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OPEN PIT MINE
PLANNING & DESIGN

3RD EDITION

1. FUNDAMENTALS

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A BALKEMA BOOK

OPEN PIT MINE PLANNING & DESIGN
VOLUME 1 – FUNDAMENTALS

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OPEN PIT MINE PLANNING & DESIGN

Volume 1 – Fundamentals

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Preface to the 3rd Edition

The first edition of *Open Pit Mine Planning and Design* appeared in 1995. Volume 1, the "Fundamentals", consisted of six chapters

1. Mine Planning
2. Mining Revenues and Costs
3. Orebody Description
4. Geometrical Considerations
5. Pit Limits
6. Production Planning

totaling 636 pages. Volume 2, the "CSMine Software Package" was written in support of the student- and engineer-friendly CSMine pit generation computer program included on a CD enclosed in a pocket inside the back cover. This volume, which contained six chapters and 200 pages, consisted of (1) a description of a small copper deposit in Arizona to be used for demonstrating and applying the mine planning and design principles, (2) the CSMine tutorial, (3) the CSMine user's manual, and (4) the VarioC tutorial, user's manual and reference guide. The VarioC microcomputer program, also included on the CD, was to be used for the statistical analysis of the drill hole data, calculation of experimental variograms, and interactive modeling involving the variogram. The main purpose of the CSMine software was as a learning tool. Students could learn to run it in a very short time and they could then focus on the pit design principles rather than on the details of the program. CSMine could handle 10,000 blocks which was sufficient to run relatively small problems.

We were very pleased with the response received and it became quite clear that a second edition was in order. In Volume 1, Chapters 1 and 3 through 6 remained largely the same but the reference lists were updated. The costs and prices included in Chapter 2 "Mining Costs and Revenues" were updated. Two new chapters were added to Volume 1:

7. Reporting of Mineral Resources and Ore Reserves
8. Responsible Mining

To facilitate the use of this book in the classroom, review questions and exercises were added at the end of Chapters 1 through 8. The "answers" were not, however, provided. There were several reasons for this. First, most of the answers could be found by the careful reading, and perhaps re-reading, of the text material. Secondly, for practicing mining engineers, the answers to the opportunities offered by their operations are seldom provided in advance. The fact that the answers were not given should help introduce the student to the real world of mining problem solving. Finally, for those students using the book under the guidance of a professor, some of the questions will offer discussion possibilities. There is no single "right" answer for some of the included exercises.

In Volume 2, the CSMine software included in the first edition was written for the DOS operating system which was current at that time. Although the original program does work in the Windows environment, it is not optimum. Furthermore, with the major advances in computer power that occurred during the intervening ten-year period, many improvements could be incorporated. Of prime importance, however, was to retain the user friendliness of the original CSMine. Its capabilities were expanded to be able to involve 30,000 blocks.

A total of eight drill hole data sets involving three iron properties, two gold properties and three copper properties were included on the distribution CD. Each of these properties was described in some detail. It was intended that, when used in conjunction with the CSMine software, these data sets might form the basis for capstone surface mine designs. It has been the experience of the authors when teaching capstone design courses that a significant problem for the student is obtaining a good drillhole data set. Hopefully the inclusion of these data sets has been of some help in this regard.

The second edition was also well received and the time arrived to address the improvements to be included in this, the 3rd edition. The structure and fundamentals have withstood the passage of time and have been retained. The two-volume presentation has also been maintained.

However, for those of you familiar with the earlier editions, you will quickly notice one major change. A new author, in the form of Randy Martin, has joined the team of Bill Hustrulid and Mark Kuchta in preparing this new offering. Randy is the "Mother and Father" of the very engineer-friendly and widely used MicroMODEL open pit mine design software. As part of the 3rd edition, he has prepared an "academic" version of his software package. It has all of the features of his commercial version but is limited in application to six data sets:

- Ariz_Cu: the same copper deposit used with CSMine (36,000 blocks)
- Andina_Cu: a copper deposit from central Chile (1,547,000 blocks)
- Azul: a gold deposit from central Chile (668,150 blocks)
- MMdemo: a gold deposit in Nevada (359,040 blocks)
- Norte_Cu: a copper deposit in northern Chile (3,460,800 blocks)
- SeamDemo: a thermal coal deposit in New Mexico (90,630 blocks).

Our intention has been to expose the student to more realistic applications once the fundamentals have been learned via the CSMine software (30,000 block limitation). The MicroMODEL V8.1 Academic version software is included on the CD together with the 6 data sets. The accompanying tutorial has been added as Chapter 16. Our idea is that the student will begin their computer-aided open pit mine design experience using CSMine and the Ariz_Cu data set and then progress to applying MicroMODEL to the same set with help from the tutorial.

The new chapter makeup of Volume 2 is

14. The CSMine Tutorial
15. CSMine User's Guide
16. The MicroMODEL V8.1 Mine Design Software
17. Orebody Case Examples

Volume 1, "Fundamentals", has also experienced some noticeable changes. Chapters 1 and 3 through 8 have been retained basically as presented in the second edition. The prices and costs provided in Chapter 2 have been revised to reflect those appropriate for today (2012). The reference list included at the end of each chapter has been revised. In the earlier editions, no real discussion of the basic unit operations was included. This has now been corrected with the addition of:

9. Blasting
10. Rotary Drilling
11. Shovel Loading
12. Truck Haulage
13. Equipment Availability and Utilization

Each chapter has a set of “Review Questions and Exercises”.

The authors would like to acknowledge the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) for permission to include their ‘Estimation of Mineral Resources and Mineral Reserves: Best Practices Guidelines’ in Chapter 7. The Australasian Institute of Mining and Metallurgy (AusIMM) was very kind to permit our inclusion of the ‘JORC-2004 Code’ in Chapter 7. The current commodity prices were kindly supplied by Platt’s Metals Week, the Metal Bulletin, Minerals Price Watch, and Skillings Mining Review. The Engineering News-Record graciously allowed the inclusion of their cost indexes. The CMJ Mining Sourcebook, Equipment Watch (a Penton Media Brand), and InfoMine USA provided updated costs. Thomas Martin kindly permitted the inclusion of materials from the book “Surface Mining Equipment”. The authors drew very heavily on the statistics carefully compiled by the U.S. Department of Labor, the U.S. Bureau of Labor Statistics, and the U.S. Geological Survey. Mining equipment suppliers Atlas Copco, Sandvik Mining, Komatsu America Corporation, Terex Inc., Joy Global (P&H), Siemens Industry, Inc., and Varel International have graciously provided us with materials for inclusion in the 3rd edition. Ms. Jane Olivier, Publications Manager, Society for Mining, Metallurgy and Exploration (SME) has graciously allowed inclusion of materials from the 3rd edition, Mining Engineering Handbook. Otto Schumacher performed a very thorough review of the materials included in chapters 9 through 13. Last, but not least, Ms. Arlene Chafe provided us access to the publications of the International Society of Explosive Engineers (ISEE).

The drill hole sets included in Chapter 17 were kindly supplied by Kennecott Barneys Canyon mine, Newmont Mining Corporation, Minnesota Department of Revenue, Minnesota Division of Minerals (lronton Office), Geneva Steel and Codelco.

Finally, we would like to thank those of you who bought the first and second editions of this book and have provided useful suggestions for improvement.

The result is what you now hold in your hands. We hope that you will find some things of value. In spite of the changes that have taken place in the content of the book over the years, our basic philosophy has remained the same – to produce a book which will form an important instrument in the process of learning/teaching about the engineering principles and application of them involved in the design of open pit mines.

Another important “consistency” with this 3rd edition is the inclusion of the Bingham Pit on the cover. Obviously the pit has also changed over the years but this proud lady which was first mined as an open pit in 1906 is still a remarkable beauty! Kennecott Utah Copper generously provided the beautiful photo of their Bingham Canyon mine for use on the cover.

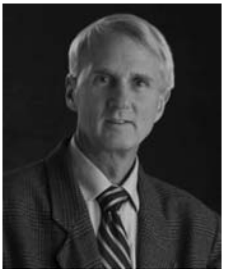
Important Notice – Please Read

This book has been primarily written for use as a textbook by students studying mining engineering, in general, and surface mining, in particular. The focus has been on presenting the concepts and principles involved in a logical and easily understood way. In spite of great efforts made to avoid the introduction of mistakes both in understanding and presentation, they may have been inadvertently/unintentionally introduced. The authors would be pleased if you, the reader, would bring such mistakes to their attention so that they may be corrected in subsequent editions.

Neither the authors nor the publisher shall, in any event, be liable for any damages or expenses, including consequential damages and expenses, resulting from the use of the information, methods, or products described in this textbook. Judgments made regarding the suitability of the techniques, procedures, methods, equations, etc. for any particular application are the responsibility of the user, and the user alone. It must be recognized that there is still a great deal of 'art' in successful mining and hence careful evaluation and testing remains an important part of technique and equipment selection at any particular mine.

About the Authors

William Hustrulid studied Minerals Engineering at the University of Minnesota. After obtaining his Ph.D. degree in 1968, his career has included responsible roles in both mining academia and in the mining business itself. He has served as Professor of Mining Engineering at the University of Utah and at the Colorado School of Mines and as a Guest Professor at the Technical University in Luleå, Sweden. In addition, he has held mining R&D positions for companies in the USA, Sweden, and the former Republic of Zaire. He is a Member of the U.S. National Academy of Engineering (NAE) and a Foreign Member of the Swedish Royal Academy of Engineering Sciences (IVA). He currently holds the rank of Professor Emeritus at the University of Utah and manages Hustrulid Mining Services in Spokane, Washington.



Mark Kuchta studied Mining Engineering at the Colorado School of Mines and received his Ph.D. degree from the Technical University in Luleå, Sweden. He has had a wide-ranging career in the mining business. This has included working as a contract miner in the uranium mines of western Colorado and 10 years of experience in various positions with LKAB in northern Sweden. At present, Mark is an Associate Professor of Mining Engineering at the Colorado School of Mines. He is actively involved in the education of future mining engineers at both undergraduate and graduate levels and conducts a very active research program. His professional interests include the use of high-pressure waterjets for rock scaling applications in underground mines, strategic mine planning, advanced mine production scheduling and the development of user-friendly mine software.

Randall K. "Randy" Martin studied Metallurgical Engineering at the Colorado School of Mines and later received a Master of Science in Mineral Economics from the Colorado School of Mines. He has over thirty years of experience as a geologic modeler and mine planner, having worked for Amax Mining, Pincock, Allen & Holt, and Tetratech. Currently he serves as President of R.K. Martin and Associates, Inc. His company performs consulting services, and also markets and supports a variety of software packages which are used in the mining industry. He is the principal author of the MicroMODEL® software included with this textbook.

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CHAPTER 1

Mine planning

1.1 INTRODUCTION

1.1.1 *The meaning of ore*

One of the first things discussed in an Introduction to Mining course and one which students must commit to memory is the definition of 'ore'. One of the more common definitions (USBM, 1967) is given below:

Ore: A metalliferous mineral, or an aggregate of metalliferous minerals, more or less mixed with gangue which from the standpoint of the miner can be mined at a profit or, from the standpoint of a metallurgist can be treated at a profit.

This standard definition is consistent with the custom of dividing mineral deposits into two groups: metallic (ore) and non-metallic. Over the years, the usage of the word 'ore' has been expanded by many to include non-metallics as well. The definition of ore suggested by Banfield (1972) would appear to be more in keeping with the general present day usage.

Ore: A natural aggregate of one or more solid minerals which can be mined, or from which one or more mineral products can be extracted, at a profit.

In this book the following, somewhat simplified, definition will be used:

Ore: A natural aggregation of one or more solid minerals that can be mined, processed and sold at a profit.

Although definitions are important to know, it is even more important to know what they mean. To prevent the reader from simply transferring this definition directly to memory without being first processed by the brain, the 'meaning' of ore will be expanded upon.

The key concept is 'extraction leading to a profit'. For engineers, profits can be expressed in simple equation form as

$$\text{Profits} = \text{Revenues} - \text{Costs} \tag{1.1}$$

The revenue portion of the equation can be written as

$$\text{Revenues} = \text{Material sold (units)} \times \text{Price/unit} \tag{1.2}$$

The costs can be similarly expressed as

$$\text{Costs} = \text{Material sold (units)} \times \text{Cost/unit} \tag{1.3}$$

Combining the equations yields

$$\text{Profits} = \text{Material sold (units)} \times (\text{Price/unit} - \text{Cost/unit}) \tag{1.4}$$

As has been the case since the early Phoenician traders, the minerals used by modern man come from deposits scattered around the globe. The price received is more and more being set by world wide supply and demand. Thus, the price component in the equation is largely determined by others. Where the mining engineer can and does enter is in doing something about the unit costs. Although the development of new technology at your property is one answer, new technology easily and quickly spreads around the world and soon all operations have the 'new' technology. Hence to remain profitable over the long term, the mining engineer must continually examine and assess smarter and better site specific ways for reducing costs at the operation. This is done through a better understanding of the deposit itself and the tools/techniques employed or employable in the extraction process. Cost containment/reduction through efficient, safe and environmentally responsive mining practices is serious business today and will be even more important in the future with increasing mining depths and ever more stringent regulations. A failure to keep up is reflected quite simply by the profit equation as

$$\text{Profits} < 0 \tag{1.5}$$

This, needless to say, is unfavorable for all concerned (the employees, the company, and the country or nation). For the mining engineer (student or practicing) reading this book, the personal meaning of ore is

$$\text{Ore} \equiv \text{Profits} \equiv \text{Jobs} \tag{1.6}$$

The use of the mathematical equivalence symbol simply says that 'ore' is equivalent to 'profits' which is equivalent to 'jobs'. Hence one important meaning of 'ore' to us in the minerals business is jobs. Probably this simple practical definition is more easily remembered than those offered earlier. The remainder of the book is intended to provide the engineer with tools to perform even better in an increasingly competitive world.

1.1.2 *Some important definitions*

The exploration, development, and production stages of a mineral deposit (Banfield & Havard, 1975) are defined as:

Exploration: The search for a mineral deposit (prospecting) and the subsequent investigation of any deposit found until an orebody, if such exists, has been established.

Development: Work done on a mineral deposit, after exploration has disclosed ore in sufficient quantity and quality to justify extraction, in order to make the ore available for mining.

Production: The mining of ores, and as required, the subsequent processing into products ready for marketing.

It is essential that the various terms used to describe the nature, size and tenor of the deposit be very carefully selected and then used within the limits of well recognized and accepted definitions.

Over the years a number of attempts have been made to provide a set of universally accepted definitions for the most important terms. These definitions have evolved somewhat as the technology used to investigate and evaluate orebodies has changed. On February 24, 1991, the report, 'A Guide for Reporting Exploration Information, Resources and Reserves' prepared by Working Party No. 79 – 'Ore Reserves Definition' of the Society of Mining, Metallurgy and Exploration (SME), was delivered to the SME Board of Directors (SME, 1991). This report was subsequently published for discussion. In this section, the 'Definitions' and 'Report Terminology' portions of their report (SME, 1991) are included. The interested reader is encouraged to consult the given reference for the detailed guidelines. The definitions presented are tied closely to the sequential relationship between exploration information, resources and reserves shown in Figure 1.1.

With an increase in geological knowledge, the exploration information may become sufficient to calculate a resource. When economic information increases it may be possible to convert a portion of the resource to a reserve. The double arrows between reserves and resources in Figure 1.1 indicate that changes due to any number of factors may cause material to move from one category to another.

Definitions

Exploration information. Information that results from activities designed to locate economic deposits and to establish the size, composition, shape and grade of these deposits. Exploration methods include geological, geochemical, and geophysical surveys, drill holes, trial pits and surface underground openings.

Resource. A concentration of naturally occurring solid, liquid or gaseous material in or on the Earth's crust in such form and amount that economic extraction of a commodity from the concentration is currently or potentially feasible. Location, grade, quality, and quantity are known or estimated from specific geological evidence. To reflect varying degrees of geological certainty, resources can be subdivided into measured, indicated, and inferred.

– Measured. Quantity is computed from dimensions revealed in outcrops, trenches, workings or drill holes; grade and/or quality are computed from the result of detailed sampling. The sites for inspection, sampling and measurement are spaced so closely and the geological character is so well defined that size, shape, depth and mineral content of the resource are well established.

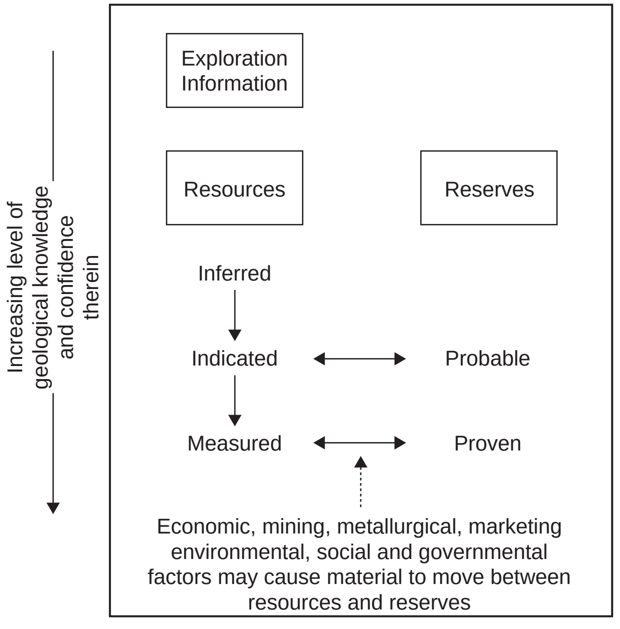


Figure 1.1. The relationship between exploration information, resources and reserves (SME, 1991).

– Indicated. Quantity and grade and/or quality are computed from information similar to that used for measured resources, but the sites for inspection, sampling, and measurements are farther apart or are otherwise less adequately spaced. The degree of assurance, although lower than that for measured resources, is high enough to assume geological continuity between points of observation.

– Inferred. Estimates are based on geological evidence and assumed continuity in which there is less confidence than for measured and/or indicated resources. Inferred resources may or may not be supported by samples or measurements but the inference must be supported by reasonable geo-scientific (geological, geochemical, geophysical, or other) data.

Reserve. A reserve is that part of the resource that meets minimum physical and chemical criteria related to the specified mining and production practices, including those for grade, quality, thickness and depth; and can be reasonably assumed to be economically and legally extracted or produced at the time of determination. The feasibility of the specified mining and production practices must have been demonstrated or can be reasonably assumed on the basis of tests and measurements. The term reserves need not signify that extraction facilities are in place and operative.

The term economic implies that profitable extraction or production under defined investment assumptions has been established or analytically demonstrated. The assumptions made must be reasonable including assumptions concerning the prices and costs that will prevail during the life of the project.

The term 'legally' does not imply that all permits needed for mining and processing have been obtained or that other legal issues have been completely resolved. However, for a reserve to exist, there should not be any significant uncertainty concerning issuance of these permits or resolution of legal issues.

Reserves relate to resources as follows:

– Proven reserve. That part of a measured resource that satisfies the conditions to be classified as a reserve.

– Probable reserve. That part of an indicated resources that satisfies the conditions to be classified as a reserve.

It should be stated whether the reserve estimate is of in-place material or of recoverable material. Any in-place estimate should be qualified to show the anticipated losses resulting from mining methods and beneficiation or preparation.

Reporting terminology

The following terms should be used for reporting exploration information, resources and reserves:

1. Exploration information. Terms such as 'deposit' or 'mineralization' are appropriate for reporting exploration information. Terms such as 'ore,' 'reserve,' and other terms that imply that economic extraction or production has been demonstrated, should not be used.

2. Resource. A resource can be subdivided into three categories:
   (a) Measured resource;
   (b) Indicated resource;
   (c) Inferred resource.

The term 'resource' is recommended over the terms 'mineral resource, identified resource' and 'in situ resource.' 'Resource' as defined herein includes 'identified resource,' but excludes 'undiscovered resource' of the United States Bureau of Mines (USBM) and United

States Geological Survey (USGS) classification scheme. The 'undiscovered resource' classification is used by public planning agencies and is not appropriate for use in commercial ventures.

3. Reserve. A reserve can be subdivided into two categories:
   (a) Probable reserve;
   (b) Proven reserve.

The term 'reserve' is recommended over the terms 'ore reserve,' 'minable reserve' or 'recoverable reserve.'

The terms 'measured reserve' and 'indicated reserve,' generally equivalent to 'proven reserve' and 'probable reserve,' respectively, are not part of this classification scheme and should not be used. The terms 'measured,' 'indicated' and 'inferred' qualify resources and reflect only differences in geological confidence. The terms 'proven' and 'probable' qualify reserves and reflect a high level of economic confidence as well as differences in geological confidence.

The terms 'possible reserve' and 'inferred reserve' are not part of this classification scheme. Material described by these terms lacks the requisite degree of assurance to be reported as a reserve.

The term 'ore' should be used only for material that meets the requirements to be a reserve.

It is recommended that proven and probable reserves be reported separately. Where the term reserve is used without the modifiers proven or probable, it is considered to be the total of proven and probable reserves.

1.2 MINE DEVELOPMENT PHASES

The mineral supply process is shown diagrammatically in Figure 1.2. As can be seen a positive change in the market place creates a new or increased demand for a mineral product.

In response to the demand, financial resources are applied in an exploration phase resulting in the discovery and delineation of deposits. Through increases in price and/or advances in technology, previously located deposits may become interesting. These deposits must then be thoroughly evaluated regarding their economic attractiveness. This evaluation process will be termed the 'planning phase' of a project (Lee, 1984). The conclusion of this phase will be the preparation of a feasibility report. Based upon this, the decision will be made as to whether or not to proceed. If the decision is 'go', then the development of the mine and concentrating facilities is undertaken. This is called the implementation, investment, or design and construction phase. Finally there is the production or operational phase during which the mineral is mined and processed. The result is a product to be sold in the marketplace. The entrance of the mining engineer into this process begins at the planning phase and continues through the production phase. Figure 1.3 is a time line showing the relationship of the different phases and their stages.

The implementation phase consists of two stages (Lee, 1984). The design and construction stage includes the design, procurement and construction activities. Since it is the period of major cash flow for the project, economies generally result by keeping the time frame to a realistic minimum. The second stage is commissioning. This is the trial operation of the individual components to integrate them into an operating system and ensure their readiness

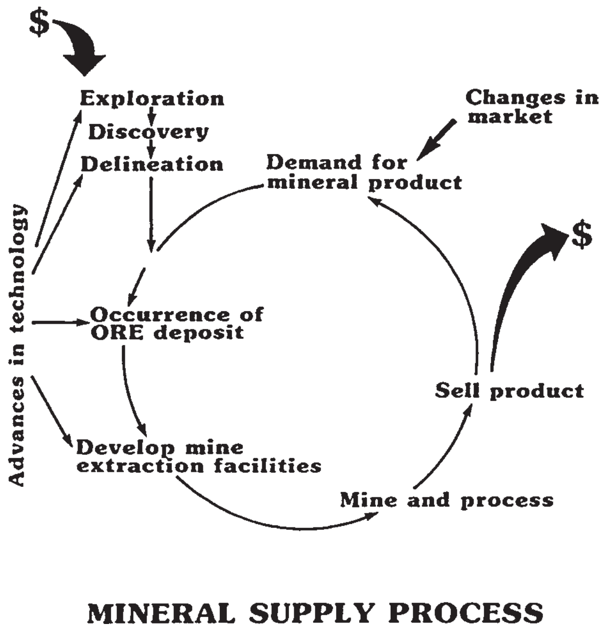


Figure 1.2. Diagrammatic representation of the mineral supply process (McKenzie, 1980).

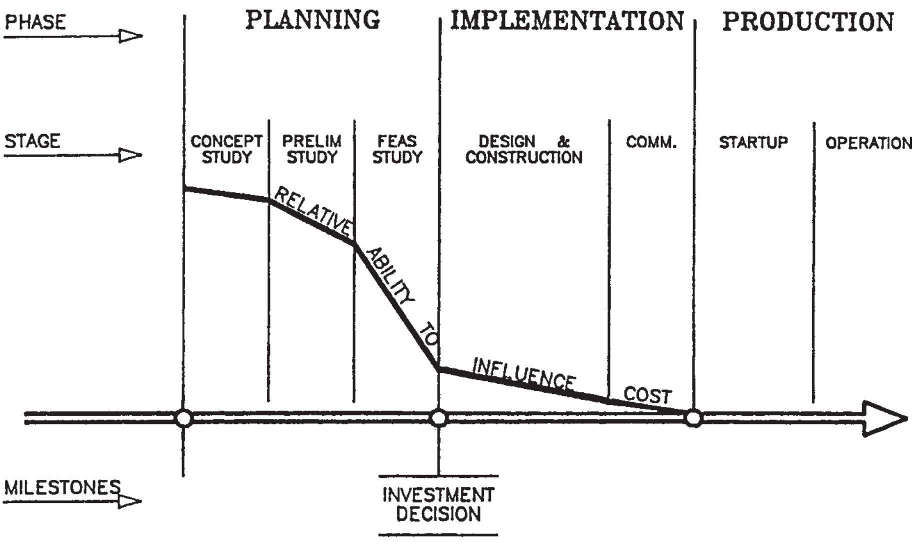


Figure 1.3. Relative ability to influence costs (Lee, 1984).

for startup. It is conducted without feedstock or raw materials. Frequently the demands and costs of the commissioning period are underestimated.

The production phase also has two stages (Lee, 1984). The startup stage commences at the moment that feed is delivered to the plant with the express intention of transforming it into product. Startup normally ends when the quantity and quality of the product is sustainable at the desired level. Operation commences at the end of the startup stage.

As can be seen in Figure 1.3, and as indicated by Lee (1984),

the planning phase offers the greatest opportunity to minimize the capital and operating costs of the ultimate project, while maximizing the operability and profitability of the venture. But the opposite is also true: no phase of the project contains the potential for instilling technical or fiscal disaster into a developing project, that is inherent in the planning phase. . . .

At the start of the conceptual study, there is a relatively unlimited ability to influence the cost of the emerging project. As decisions are made, correctly or otherwise, during the balance of the planning phase, the opportunity to influence the cost of the job diminishes rapidly.

The ability to influence the cost of the project diminishes further as more decisions are made during the design stage. At the end of the construction period there is essentially no opportunity to influence costs.

The remainder of this chapter will focus on the activities conducted within the planning stage.

1.3 AN INITIAL DATA COLLECTION CHECKLIST

In the initial planning stages for any new project there are a great number of factors of rather diverse types requiring consideration. Some of these factors can be easily addressed, whereas others will require in-depth study. To prevent forgetting factors, checklist are often of great value. Included below are the items from a 'Field Work Program Checklist for New Properties' developed by Halls (1975). Student engineers will find many of the items on this checklist of relevance when preparing mine design reports.

Checklist items (Halls, 1975)

1. Topography
   - (a) USGS maps
   - (b) Special aerial or land survey
     - Establish survey control stations
     - Contour
2. Climatic conditions
   - (a) Altitude
   - (b) Temperatures
     - Extremes
     - Monthly averages
   - (c) Precipitation
     - Average annual precipitation
     - Average monthly rainfall
     - Average monthly snowfall
     - Run-off
       - Normal
       - Flood
     - Slides – snow and mud
   - (d) Wind
     - Maximum recorded
     - Prevailing direction
     - Hurricanes, tornados, cyclones, etc.

(e) Humidity
Effect on installations, i.e. electrical motors, etc.
(f) Dust
(g) Fog and cloud conditions

3. Water – potable and process
(a) Sources
Streams
Lakes
Wells
(b) Availability
Ownership
Water rights
Cost
(c) Quantities
Monthly availability
Flow rates
Drought or flood conditions
Possible dam locations
(d) Quality
Present sample
Possibility of quality change in upstream source water
Effect of contamination on downstream users
(e) Sewage disposal method

4. Geologic structure
(a) Within mine area
(b) Surrounding areas
(c) Dam locations
(d) Earthquakes
(e) Effect on pit slopes
Maximum predicted slopes
(f) Estimate on foundation conditions

5. Mine water as determined by prospect holes
(a) Depth
(b) Quantity
(c) Method of drainage

6. Surface
(a) Vegetation
Type
Method of clearing
Local costs for clearing
(b) Unusual conditions
Extra heavy timber growth
Muskeg
Lakes
Stream diversions
Gravel deposits

7. Rock type – overburden and ore
   (a) Submit sample for drillability test
   (b) Observe fragmentation features
       Hardness
       Degree of weathering
       Cleavage and fracture planes
       Suitability for road surface
8. Locations for concentrator – factors to consider for optimum location
   (a) Mine location
       Haul uphill or downhill
   (b) Site preparation
       Amount of cut and/or fill
   (c) Process water
       Gravity flow or pumping
   (d) Tailings disposal
       Gravity flow or pumping
   (e) Maintenance facilities
       Location
9. Tailings pond area
   (a) Location of pipeline length and discharge elevations
   (b) Enclosing features
       Natural
       Dams or dikes
       Lakes
   (c) Pond overflow
       Effect of water pollution on downstream users
       Possibility for reclaiming water
   (d) Tailings dust
       Its effect on the area
10. Roads
   (a) Obtain area road maps
   (b) Additional road information
       Widths
       Surfacing
       Maximum load limits
       Seasonal load limits
       Seasonal access
       Other limits or restrictions
       Maintained by county, state, etc.
   (c) Access roads to be constructed by company (factors considered)
       Distance
       Profile
       Cut and fill
       Bridges, culverts
       Terrain and soil conditions

11. Power
    (a) Availability
        Kilovolts
        Distance
        Rates and length of contract
    (b) Power lines to site
        Who builds
        Who maintains
        Right-of-way requirements
    (c) Substation location
    (d) Possibility of power generation at or near site
12. Smelting
    (a) Availability
    (b) Method of shipping concentrate
    (c) Rates
    (d) If company on site smelting – effect of smelter gases
    (e) Concentrate freight rates
    (f) Railroads and dock facility
13. Land ownership
    (a) Present owners
    (b) Present usage
    (c) Price of land
    (d) Types of options, leases and royalties expected
14. Government
    (a) Political climate
        Favorable or unfavorable to mining
        Past reactions in the area to mining
    (b) Special mining laws
    (c) Local mining restrictions
15. Economic climate
    (a) Principal industries
    (b) Availability of labor and normal work schedules
    (c) Wage scales
    (d) Tax structure
    (e) Availability of goods and services
        Housing
        Stores
        Recreation
        Medical facilities and unusual local disease
        Hospital
        Schools
    (f) Material costs and/or availability
        Fuel oil
        Concrete
        Gravel
        Borrow material for dams

   (g) Purchasing
       Duties
16. Waste dump location
   (a) Haul distance
   (b) Haul profile
   (c) Amenable to future leaching operation
17. Accessibility of principal town to outside
   (a) Methods of transportation available
   (b) Reliability of transportation available
   (c) Communications
18. Methods of obtaining information
   (a) Past records (i.e. government sources)
   (b) Maintain measuring and recording devices
   (c) Collect samples
   (d) Field observations and measurements
   (e) Field surveys
   (f) Make preliminary plant layouts
   (g) Check courthouse records for land information
   (h) Check local laws and ordinances for applicable legislation
   (i) Personal inquiries and observation on economic and political climates
   (j) Maps
   (k) Make cost inquiries
   (l) Make material availability inquiries
   (m) Make utility availability inquiries

1.4 THE PLANNING PHASE

In preparing this section the authors have drawn heavily on material originally presented in papers by Lee (1984) and Taylor (1977). The permission by the authors and their publisher, The Northwest Mining Association, to include this material is gratefully acknowledged.

1.4.1 *Introduction*

The planning phase commonly involves three stages of study (Lee, 1984).

Stage 1: *Conceptual study*
A conceptual (or preliminary valuation) study represents the transformation of a project idea into a broad investment proposition, by using comparative methods of scope definition and cost estimating techniques to identify a potential investment opportunity. Capital and operating costs are usually approximate ratio estimates using historical data. It is intended primarily to highlight the principal investment aspects of a possible mining proposition. The preparation of such a study is normally the work of one or two engineers. The findings are reported as a preliminary valuation.

Stage 2: *Preliminary or pre-feasibility study*
A preliminary study is an intermediate-level exercise, normally not suitable for an investment decision. It has the objectives of determining whether the project concept justifies a detailed

analysis by a feasibility study, and whether any aspects of the project are critical to its viability and necessitate in-depth investigation through functional or support studies.

A preliminary study should be viewed as an intermediate stage between a relatively inexpensive conceptual study and a relatively expensive feasibility study. Some are done by a two or three man team who have access to consultants in various fields others may be multi-group efforts.

Stage 3: *Feasibility study*
The feasibility study provides a definitive technical, environmental and commercial base for an investment decision. It uses iterative processes to optimize all critical elements of the project. It identifies the production capacity, technology, investment and production costs, sales revenues, and return on investment. Normally it defines the scope of work unequivocally, and serves as a base-line document for advancement of the project through subsequent phases.

These latter two stages will now be described in more detail.

1.4.2 *The content of an intermediate valuation report*

The important sections of an intermediate valuation report (Taylor, 1977) are:

– Aim;
– Technical concept;
– Findings;
– Ore tonnage and grade;
– Mining and production schedule;
– Capital cost estimate;
– Operating cost estimate;
– Revenue estimate;
– Taxes and financing;
– Cash flow tables.

The degree of detail depends on the quantity and quality of information. Table 1.1 outlines the contents of the different sections.

1.4.3 *The content of the feasibility report*

The essential functions of the feasibility report are given in Table 1.2.

Due to the great importance of this report it is necessary to include all detailed information that supports a general understanding and appraisal of the project or the reasons for selecting particular processes, equipment or courses of action. The contents of the feasibility report are outlined in Table 1.3.

The two important requirements for both valuation and feasibility reports are:

1. Reports must be easy to read, and their information must be easily accessible.
2. Parts of the reports need to be read and understood by non-technical people.

According to Taylor (1977):

There is much merit in a layered or pyramid presentation in which the entire body of information is assembled and retained in three distinct layers.

Layer 1. Detailed background information neatly assembled in readable form and adequately indexed, but retained in the company's office for reference and not included in the feasibility report.

Layer 2. Factual information about the project, precisely what is proposed to be done about it, and what the technical, physical and financial results are expected to be.

Layer 3. A comprehensive but reasonably short summary report, issued preferably as a separate volume.

The feasibility report itself then comprises only the second and third layers. While everything may legitimately be grouped into a single volume, the use of smaller separated volumes makes for easier reading and for more flexible forms of binding. Feasibility reports always need to be reviewed by experts in various specialities. The use of several smaller volumes makes this easier, and minimizes the total number of copies needed.

Table 1.1. The content of an intermediate valuation report (Taylor, 1977).

Aim: States briefly what knowledge is being sought about the property, and why, for guidance in exploration spending, for joint venture negotiations, for major feasibility study spending, etc. Sources of information are also conveniently listed.

Concept: Describes very briefly where the property is located, what is proposed or assumed to be done in the course of production, how this may be achieved, and what is to be done with the products.

Findings: Comprise a summary, preferably in sequential and mainly tabular forms, of the important figures and observations from all the remaining sections. This section may equally be termed Conclusions, though this title invites a danger of straying into recommendations which should not be offered unless specially requested.

Any cautions or reservations the authors care to make should be incorporated in one of the first three sections. The general aim is that the non-technical or less-technical reader should be adequately informed about the property by the time he has read the end of Findings.

Ore tonnage and grade: Gives brief notes on geology and structure, if applicable, and on the drilling and sampling accomplished. Tonnages and grades, both geological and minable and possibly at various cut-off grades, are given in tabular form with an accompanying statement on their status and reliability.

Mining and production schedule: Tabulates the mining program (including preproduction work), the milling program, any expansions or capacity changes, the recoveries and product qualities (concentrate grades), and outputs of products.

Capital cost estimate: Tabulates the cost to bring the property to production from the time of writing including the costs of further exploration, research and studies. Any prereport costs, being sunk, may be noted separately.

An estimate of postproduction capital expenditures is also needed. This item, because it consists largely of imponderables, tends to be underestimated even in detailed feasibilities studies.

Operating cost estimate: Tabulates the cash costs of mining, milling, other treatment, ancillary services, administration, etc. Depreciation is not a cash cost, and is handled separately in cash flow calculations. Postmine treatment and realization costs are most conveniently regarded as deductions from revenues.

Revenue estimate: Records the metal or product prices used, states the realization terms and costs, and calculates the net smelter return or net price at the deemed point of disposal. The latter is usually taken to be the point at which the product leaves the mine's plant and is handed over to a common carrier. Application of these net prices to the outputs determined in the production schedule yields a schedule of annual revenues.

Financing and tax data: State what financing assumptions have been made, all equity, all debt or some specified mixture, together with the interest and repayment terms of loans. A statement on the tax regime specifies tax holidays (if any), depreciation and tax rates, (actual or assumed) and any special

(Continued)

Table 1.1. (Continued).

features. Many countries, particularly those with federal constitutions, impose multiple levels of taxation by various authorities, but a condensation or simplification of formulae may suffice for early studies without involving significant loss of accuracy.

Cash flow schedules: Present (if information permits) one or more year-by-year projections of cash movements in and out of the project. These tabulations are very informative, particularly because their format is almost uniformly standardized. They may be compiled for the indicated life of the project or, in very early studies, for some arbitrary shorter period.
Figures must also be totalled and summarized. Depending on company practice and instructions, investment indicators such as internal rate of return, debt payback time, or cash flow after payback may be displayed.

Table 1.2. The essential functions of the feasibility report (Taylor, 1977).

1. To provide a comprehensive framework of established and detailed facts concerning the mineral project.
2. To present an appropriate scheme of exploitation with designs and equipment lists taken to a degree of detail sufficient for accurate prediction of costs and results.
3. To indicate to the project's owners and other interested parties the likely profitability of investment in the project if equipped and operated as the report specifies.
4. To provide this information in a form intelligible to the owner and suitable for presentation to prospective partners or to sources of finance.

Table 1.3. The content of a feasibility study (Taylor, 1977).

General:
- Topography, climate, population, access, services.
- Suitable sites for plant, dumps, towns, etc.

Geological (*field*):
- Geological study of structure, mineralization and possibly of genesis.
- Sampling by drilling or tunnelling or both.
- Bulk sampling for checking and for metallurgical testing.
- Extent of leached or oxidized areas (frequently found to be underestimated).
- Assaying and recording of data, including check assaying, rock properties, strength and stability.
- Closer drilling of areas scheduled for the start of mining.
- Geophysics and indication of the likely ultimate limits of mineralization, including proof of non-mineralization of plan and dump areas.
- Sources of water and of construction materials.

Geological and mining (*office*):
- Checking, correcting and coding of data for computer input.
- Manual calculations of ore tonnages and grades.
- Assay compositing and statistical analysis.
- Computation of mineral inventory (geological reserves) and minable reserves, segregated as needed by orebody, by ore type, by elevation or bench, and by grade categories.
- Computation of associated waste rock.
- Derivation of the economic factors used in the determination of minable reserves.

Mining:
- Open pit layouts and plans.
- Determination of preproduction mining or development requirements.
- Estimation of waste rock dilution and ore losses.

(Continued)

Table 1.3. (Continued).

- Production and stripping schedules, in detail for the first few years but averaged thereafter, and specifying important changes in ore types if these occur.
- Waste mining and waste disposal.
- Labor and equipment requirements and cost, and an appropriate replacement schedule for the major equipment.

Metallurgy (*research*):
- Bench testing of samples from drill cores.
- Selection of type and stages of the extraction process.
- Small scale pilot plant testing of composited or bulk samples followed by larger scale pilot mill operation over a period of months should this work appear necessary.
- Specification of degree of processing, and nature and quality of products.
- Provision of samples of the product.
- Estimating the effects of ore type or head grade variations upon recovery and product quality.

Metallurgy (*design*):
- The treatment concept in considerable detail, with flowsheets and calculation of quantities flowing.
- Specification of recovery and of product grade.
- General siting and layout of plant with drawings if necessary.

Ancillary services and requirements:
- Access, transport, power, water, fuel and communications.
- Workshops, offices, changehouse, laboratories, sundry buildings and equipment.
- Labor structure and strength.
- Housing and transport of employees.
- Other social requirements.

Capital cost estimation:
- Develop the mine and plant concepts and make all necessary drawings.
- Calculate or estimate the equipment list and all important quantities (of excavation, concrete, building area and volume, pipework, etc.).
- Determine a provisional construction schedule.
- Obtain quotes of the direct cost of items of machinery, establish the costs of materials and services, and of labor and installation.
- Determine the various and very substantial indirect costs, which include freight and taxes on equipment (may be included in directs), contractors' camps and overheads plus equipment rental, labor punitive and fringe costs, the owner's field office, supervision and travel, purchasing and design costs, licenses, fees, customs duties and sales taxes.
- Warehouse inventories.
- A contingency allowance for unforeseen adverse happenings and for unestimated small requirements that may arise.
- Operating capital sufficient to pay for running the mine until the first revenue is received.
- Financing costs and, if applicable, preproduction interest on borrowed money.

A separate exercise is to forecast the major replacements and the accompanying provisions for postproduction capital spending. Adequate allowance needs to be made for small requirements that, though unforeseeable, always arise in significant amounts.

Operating cost estimation:
- Define the labor strength, basic pay rates, fringe costs.
- Establish the quantities of important measurable supplies to be consumed – power, explosives, fuel, grinding steel, reagents, etc. – and their unit costs.
- Determine the hourly operating and maintenance costs for mobile equipment plus fair performance factors.
- Estimate the fixed administration costs and other overheads plus the irrecoverable elements of townsite and social costs.

(Continued)

Table 1.3. (Continued).

Only cash costs are used thus excluding depreciation charges that must be accounted for elsewhere. As for earlier studies, post-mine costs for further treatment and for selling the product are best regarded as deductions from the gross revenue.

Marketing:
- Product specifications, transport, marketing regulations or restrictions.
- Market analysis and forecast of future prices.
- Likely purchasers.
- Costs for freight, further treatment and sales.
- Draft sales terms, preferably with a letter of intent.
- Merits of direct purchase as against toll treatment.
- Contract duration, provisions for amendment or cost escalation.
- Requirements for sampling, assaying and umpiring.

The existence of a market contract or firm letter of intent is usually an important prerequisite to the loan financing of a new mine.

Rights, ownership and legal matters:
- Mineral rights and tenure.
- Mining rights (if separated from mineral rights).
- Rents and royalties.
- Property acquisitions or securement by option or otherwise.
- Surface rights to land, water, rights-of-way, etc.
- Licenses and permits for construction as well as operation.
- Employment laws for local and expatriate employees separately if applicable.
- Agreements between partners in the enterprise.
- Legal features of tax, currency exchange and financial matters.
- Company incorporation.

Financial and tax matters:
- Suggested organization of the enterprise, as corporation, joint-venture or partnership.
- Financing and obligations, particularly relating to interest and repayment on debt.
- Foreign exchange and reconversion rights, if applicable.
- Study of tax authorities and regimes, whether single or multiple.
- Depreciation allowances and tax rates.
- Tax concessions and the negotiating procedure for them.
- Appropriation and division of distributable profits.

Environmental effects:
- Environmental study and report; the need for pollution or related permits, the requirements during construction and during operation.
- Prescribed reports to government authorities, plans for restoration of the area after mining ceases.

Revenue and profit analysis:
- The mine and mill production schedules and the year-by-year output of products.
- Net revenue at the mine (at various product prices if desired) after deduction of transport, treatment and other realization charges.
- Calculation of annual costs from the production schedules and from unit operating costs derived previously.
- Calculation of complete cash flow schedules with depreciation, taxes, etc. for some appropriate number of years – individually for at least 10 years and grouped thereafter.
- Presentation of totals and summaries of results.
- Derived figures (rate of return, payback, profit split, etc.) as specified by owner or client.
- Assessment of sensitivity to price changes and generally to variation in important input elements.

1.5 PLANNING COSTS

The cost of these studies (Lee, 1984) varies substantially, depending upon the size and nature of the project, the type of study being undertaken, the number of alternatives to be investigated, and numerous other factors. However, the order of magnitude cost of the technical portion of studies, excluding such owner's cost items as exploration drilling, special grinding or metallurgical tests, environmental and permitting studies, or other support studies, is commonly expressed as a percentage of the capital cost of the project:

Conceptual study:	0.1 to 0.3 percent
Preliminary study:	0.2 to 0.8 percent
Feasibility study:	0.5 to 1.5 percent

1.6 ACCURACY OF ESTIMATES

The material presented in this section has been largely extracted from the paper 'Mine Valuation and Feasibility Studies' presented by Taylor (1977).

1.6.1 *Tonnage and grade*

At feasibility, by reason of multiple sampling and numerous checks, the average mining grade of some declared tonnage is likely to be known within acceptable limits, say ±5%, and verified by standard statistical methods. Although the ultimate tonnage of ore may be known for open pit mines if exploration drilling from surface penetrates deeper than the practical mining limit, in practice, the ultimate tonnage of many deposits is nebulous because it depends on cost-price relationships late in the project life. By the discount effects in present value theory, late life tonnage is not economically significant at the feasibility stage. Its significance will grow steadily with time once production has begun. It is not critical that the total possible tonnage be known at the outset. What is more important is that the grade and quality factors of the first few years of operation be known with assurance.

Two standards of importance can be defined for most large open pit mines:

1. A minimum ore reserve equal to that required for all the years that the cash flows are projected in the feasibility report must be known with accuracy and confidence.

2. An ultimate tonnage potential, projected generously and optimistically, should be calculated so as to define the area adversely affected by mining and within which dumps and plant buildings must not encroach.

1.6.2 *Performance*

This reduces to two items – throughput and recovery. Open pit mining units have well established performance rates that can usually be achieved if the work is correctly organized and the associated items (i.e. shovels and trucks) are suitably matched. Performance suffers if advance work (waste stripping in a pit) is inadequate. Care must be taken that these tasks are adequately scheduled and provided for in the feasibility study.

The throughput of a concentrator tends to be limited at either the fine crushing stage or the grinding stages. The principles of milling design are well established, but their application requires accurate knowledge of the ore's hardness and grindability. These qualities must therefore receive careful attention in the prefeasibility test work. Concentrator performance is part of a three way relationship involving the fineness of grind, recovery, and the grade of concentrate or product. Very similar relations may exist in metallurgical plants of other types. Again, accuracy can result only from adequate test-work.

1.6.3 *Costs*

Some cost items, notably in the operating cost field, differ little from mine to mine and are reliably known in detail. Others may be unique or otherwise difficult to estimate. Generally, accuracy in capital or operating cost estimating goes back to accuracy in quantities, reliable quotes or unit prices, and adequate provision for indirect or overhead items. The latter tend to form an ever increasing burden. For this reason, they should also be itemized and estimated directly whenever possible, and not be concealed in or allocated into other direct cost items.

Contingency allowance is an allowance for possible over expenditures contingent upon unforeseen happenings such as a strike or time delaying accident during construction, poor plant foundation conditions, or severe weather problems. To some extent the contingency allowance inevitably allows for certain small expenditures always known to arise but not foreseeable nor estimable in detail. Caution is needed here. The contingency allowance is not an allowance for bad or inadequate estimating, and it should never be interpreted in that manner.

The accuracy of capital and operating cost estimates increases as the project advances from conceptual to preliminary to feasibility stage. Normally acceptable ranges of accuracy are considered to be (Lee, 1984):

Conceptual study:	$\pm$30 percent
Preliminary study:	$\pm$20 percent
Feasibility study:	$\pm$10 percent

It was noted earlier that the scope of work in the conceptual and preliminary studies is not optimized. The cost estimate is suitable for decision purposes, to advance the project to the next stage, or to abort and minimize losses.

1.6.4 *Price and revenue*

The revenue over a mine's life is the largest single category of money. It has to pay for everything, including repayment of the original investment money. Because revenue is the biggest base, measures of the mine's economic merit are more sensitive to changes in revenue than to changes of similar ratio in any of the expenditure items.

Revenue is governed by grade, throughput, recovery, and metal or product price. Of these, price is: (a) by far the most difficult to estimate and (b) the one quantity largely outside the estimator's control. Even ignoring inflation, selling prices are widely variable with time. Except for certain controlled commodities, they tend to follow a cyclic pattern.

The market departments of major metal mining corporations are well informed on supply/demand relationships and metal price movements. They can usually provide forecasts of average metal prices in present value dollars, both probable and conservative, the latter being with 80% probability or better. Ideally, even at the conservative product price, the proposed project should still display at least the lowest acceptable level of profitability.

1.7 FEASIBILITY STUDY PREPARATION

The feasibility study is a major undertaking involving many people and a variety of specialized skills. There are two basic ways through which it is accomplished.

1. The mining company itself organizes the study and assembles the feasibility report. Various parts or tasks are assigned to outside consultants.
2. The feasibility work is delegated to one or more engineering companies.

Contained on the following pages is an eleven step methodology outlining the planning (Steps 1–4) organizing (Steps 5–10) and execution (Step 11) steps which might be used in conducting a feasibility study. It has been developed by Lee (1984, 1991).

Phase A. Planning

Step 1: *Establish a steering committee*. A steering committee consisting of managers and other individuals of wide experience and responsibility would be formed to overview and evaluate the direction and viability of the feasibility study team. One such steering committee might be the following:

– Vice-President (Chairman);
– General Manager, mining operations;
– Vice-President, finance;
– Chief Geologist, exploration;
– Vice-President, technical services;
– Consultant(s).

Step 2: *Establish a project study team.* The criteria for selection of the study team members would emphasize these qualities:

– Competent in their respective fields.
– Considerable experience with mining operations.
– Complementary technical abilities.
– Compatible personalities – strong interpersonal qualities.
– Commitment to be available through the implementation phase, should the prospect be viable.

The team members might be:

– Project Manager;
– Area Supervisor, mining;
– Area Supervisor, beneficiation;
– Area Supervisor, ancillaries.

Step 3: *Develop a work breakdown structure*. The Work Breakdown Structure (WBS) is defined by the American Association of Cost Engineers (AACE) as:

a product-oriented family tree division of hardware, software, facilities and other items which organizes, defines and displays all of the work to be performed in accomplishing the project objectives.

The WBS is a functional breakdown of all elements of work on a project, on a geographical and/or process basis. It is a hierarchy of work packages, or products, on a work area basis. The WBS is project-unique, reflecting the axiom that every project is a unique event.

A WBS is a simple common-sense procedure which systematically reviews the full scope of a project (or study) and breaks it down into logical packages of work. The primary challenge is normally one of perspective. It is imperative that the entire project be visualized as a sum of many parts, any one of which could be designed, scheduled, constructed, and priced as a single mini-project.

There are a number of categories which can be used to construct a work breakdown structure. These include:

(1) Components of the product;
(2) Functions;
(3) Organizational units;
(4) Geographical areas;
(5) Cost accounts;
(6) Time phases;
(7) Configuration characteristics;
(8) Deliverables;
(9) Responsible persons;
(10) Subpurposes.

It is not a rigid system. WBS categories can be used in any sequence desired, including using the same category several times. A sample WBS is shown in Figures 1.4 and 1.5.

An alternative to this is the Work Classification Structure (WCS). This commodity-based classification of goods and services is commonly used by construction contractors and consulting engineering firms as the primary cost-collection system. The specific intent of the WCS is to provide a consistent reference system for storage, comparison and evaluation of technical, man-hour and cost data from work area to work area within a project; and from project to project; and from country to country. The WCS may have different names in different organizations, but it is the 'original' costing system. It is the basis for virtually all of the estimating manuals and handbooks which identify unit costs for commodities such as concrete, or piping, or road construction, or equipment installation. The WCS provides a commodity based method to estimate and control costs. The key to the success of the WCS system within an organization is the absolute consistency with which it is used.

The WBS is of primary interest to owners and project managers – both of whom are interested in tracking cost and schedule on a work area basis. The WCS is primarily of interest to construction contractors and engineering consultants, who measure actual performance against forecast performance on a commodity basis.

Professional project managers and cost engineers normally use cost coding systems which encode both the commodity and the work package. This allows them to evaluate job-to-date performance, then forecast cost or productivity trends for the balance of the project.

Step 4: *Develop an action plan for the study.* An action plan in its simplest form, is just a logical (logic-oriented) time-bar plan listing all of the activities to be studied. Figure 1.6 is one example of such a time-bar graph. A more general action plan would have these characteristics:

1. Purpose: the action plan serves as a control document during the execution of the feasibility study. It functions as a master reference, against which change can be measured and resolved. It provides a visual communication of the logic and progress of the study.

2. Methodology: it may be possible for one person, working in isolation, to develop an action plan. However, it is substantially more desirable to have the project study team develop their plan on a participative, interactive basis. (Texas Instruments' Patrick Haggerty insisted that 'those who implement plans must make the plans'). This interaction

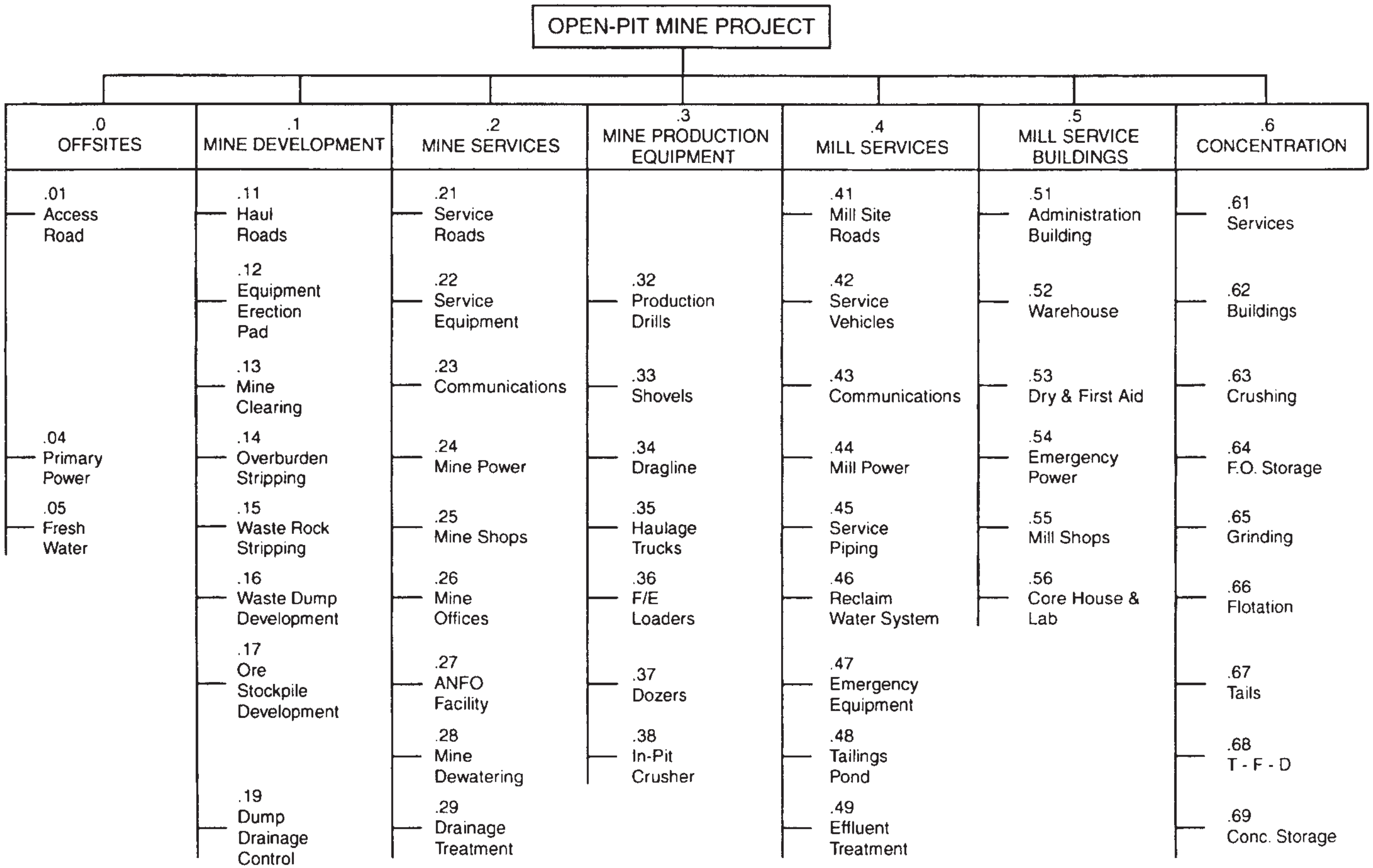


Figure 1.4. Typical work breakdown structure (WBS) directs for an open pit mining project (Lee, 1991).

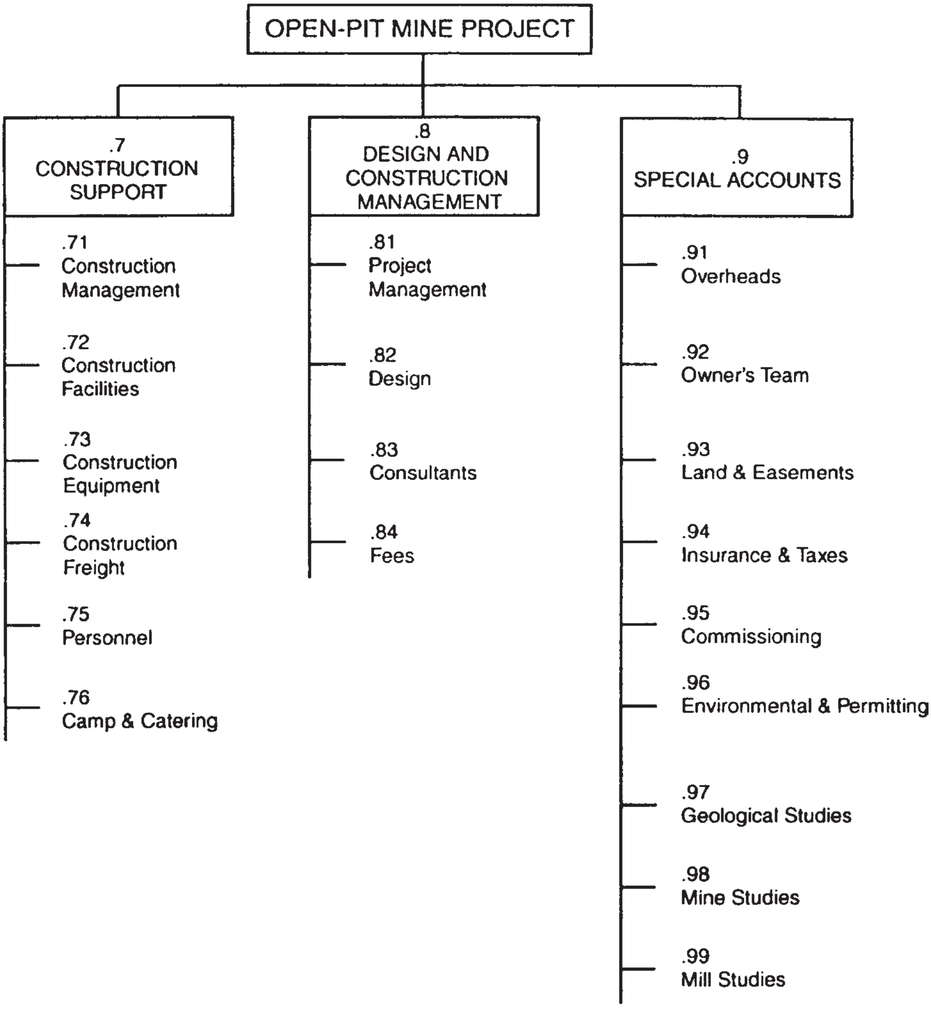


Figure 1.5. Typical work breakdown structure (WBS) indirects for an open pit mining project (Lee, 1991).

fosters understanding and appreciation of mutual requirements and objectives; even more importantly, it develops a shared commitment.

3. Format: a simple master time-bar schedule would be produced, displaying the study activities in a logic-oriented fashion. Brief titles and a reference number would be attached to each activity. For a simple in-house job, an operating company would probably stop at this point. However, for a major study on a new mining operation, the activity reference numbers and titles would be carried into a separate action plan booklet. Each activity would be described briefly, and a budget attached to it.

4. WBS reference: the most convenient way to organize these activities is by referencing them to the first and second levels of the WBS.

5. Number of activities: the practical limitation on the number of individual study activities would be of the order of one hundred.

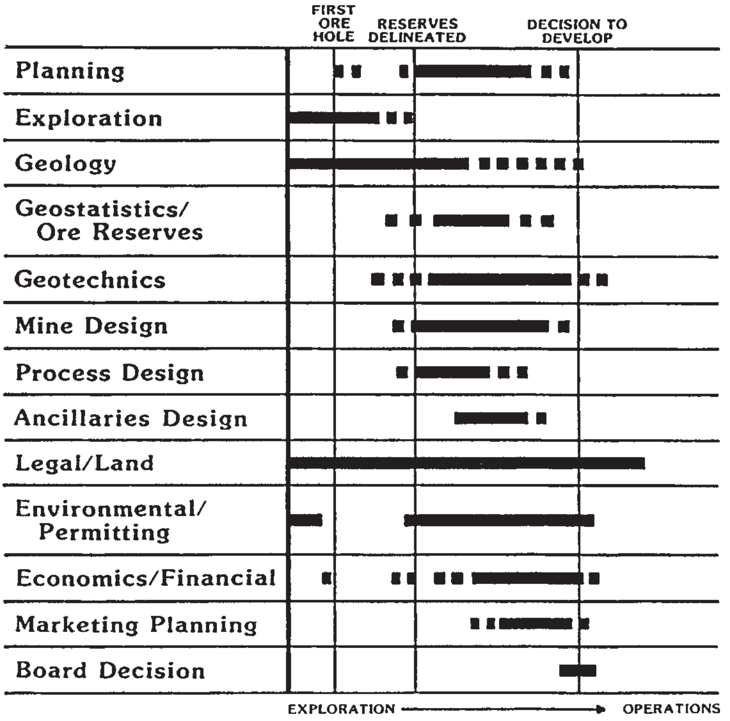


Figure 1.6. Bar chart representation for a mine feasibility/decision-making sequence (McKelvey, 1984).

Phase B. Organizing
Step 5: *Identify additional resource requirements*. While developing a comprehensive action plan, needs for additional resources normally become apparent.

Step 6: *Identify secondary project team members*.

Step 7: *Develop organization chart and responsibilities*. There are a number of ways to organize a project study team for a large study. A separate task force can be established by removing personnel from existing jobs and developing a project-oriented hierarchy, military style. This can work effectively, but can discourage broad participation in the evolution of the project. In a large company, a matrix system can be used very successfully, if used very carefully and with understanding. The management of a matrix organization is based on the management of intentional conflict; it works exceedingly well in a positive environment, and is an unequivocal disaster in an unfavorable environment.

Step 8: *Develop second-level plans and schedules*. Using the master time-bar, the action plan and the WBS as primary references, the enlarged project study team develops second-level plans and schedules, thus establishing their objectives and commitments for the balance of the study. These schedules are oriented on an area-by-area basis, with the primary team members providing the leadership for each area.

Step 9: *Identify special expertise required*. The project study team after reviewing their plan, with the additional information developed during Step 8, may identify a number of areas of the job which require special expertise. Such items may be packaged as separate Requests For Proposals (RFP's), and forwarded to pre-screened consultants on an invitation basis. The scope of work in each RFP should be clearly identified, along with the objectives for the work. A separate section provides explicit comments on the criteria for selection of the successful bidder; this provides the bidder with the opportunity to deliver proposals which can be weighted in the directions indicated by the project team.

Step 10: *Evaluate and select consultants*. Evaluation of the consultant's bids should be thorough, objective, and fair. The evaluations and decisions are made by the use of spread sheets which compare each bidder's capability to satisfy each of the objectives for the work as identified in the RFP. The objectives should be pre-weighted to remove bias from the selection process.

Phase C. Execution

Step 11: *Execute*, *monitor*, *control*. With the project study team fully mobilized and with the specialist consultants engaged and actively executing well-defined contracts, the primary challenge to the project manager is to ensure that the study stays on track.

A number of management and reporting systems and forms may be utilized, but the base-line reference for each system and report is the scope of work, schedule and cost for each activity identified in the action plan. The status-line is added to the schedule on a bi-weekly basis, and corrections and modifications made as indicated, to keep the work on track.

1.8 CRITICAL PATH REPRESENTATION

Figure 1.7 is an example of a network chart which has been presented by Taylor (1977) for a medium sized, open pit base metal mine. Each box on the chart contains:

- activity number,
- activity title,
- responsibility (this should be a person/head of section who would carry the responsibility for budget and for progress reports),
- starting date,
- completion date,
- task duration.

The activities, sequential relationships and critical/near critical paths can be easily seen.

Figure 1.8 is the branch showing the basic mining related activities. This progression will be followed through the remainder of the book.

1.9 MINE RECLAMATION

1.9.1 *Introduction*

In the past, reclamation was something to be considered at the end of mine operations and not in the planning stage. Today, in many countries at least, there will be no mine without first thoroughly and satisfactorily addressing the environmental aspects of the proposed project. Although the subject of mine reclamation is much too large to be covered in this brief chapter, some of the factors requiring planning consideration will be discussed. In the western United States, a considerable amount of mineral development takes place on federal and Indian lands. The Bureau of Land Management (BLM) of the U.S. Department of the Interior has developed the *Solid Minerals Reclamation Handbook* (BLM, 1992) with the objective being 'to provide the user with clear guidance which highlights a logical sequence for managing the reclamation process and a summary of key reclamation principles.'

The remaining sections of this chapter have been extracted from the handbook. Although they only pertain directly to those lands under BLM supervision, the concepts have more general application as well. Permission from the BLM to include this material is gratefully acknowledged.

1.9.2 *Multiple-use management*

Multiple-use management is the central concept in the Federal Land Policy and Management Act (FLPMA) of 1976. FLPMA mandates that 'the public lands be managed in a manner that will protect the quality of scientific, scenic, historical, ecological, environmental, air and atmospheric, water resource and archeological values.' Multiple-use management is defined in FLPMA (43 USC 1702(c)) and in regulations (43 CFR 1601.0-5(f)) as, in part, the 'harmonious and coordinated management of the various resources without permanent impairment of the productivity of the lands and the quality of the environment with consideration being given to the relative values of the resources and not necessarily to the combination of uses that will give the greatest economic return or the greatest unit output.' In addition, FLPMA mandates that activities be conducted so as to prevent 'unnecessary or undue degradation of the lands' (43 USC 1732 (b)).

The Mining and Minerals Policy Act of 1970 (30 USC 21(a)) established the policy for the federal government relating to mining and mineral development. The Act states that it is policy to encourage the development of 'economically sound and stable domestic mining, minerals, metal and mineral reclamation industries.' The Act also states, however, that the government should also promote the 'development of methods for the disposal, control, and reclamation of mineral waste products, and the reclamation of mined land, so as to lessen any adverse impact of mineral extraction and processing upon the physical environment that may result from mining or mineral activities.'

In accordance with the National Environmental Policy Act (NEPA), an environmental document will be prepared for those mineral actions which propose surface disturbance. The requirements and mitigation measures recommended in an Environmental Assessment (ERA) or Environmental Impact Statement (EIS) shall be made a part of the reclamation plan.

It is a statutory mandate that BLM ensure that reclamation and closure of mineral operations be completed in an environmentally sound manner. The BLM's long-term reclamation goals are to shape, stabilize, revegetate, or otherwise treat disturbed areas in order to provide a self-sustaining, safe, and stable condition that provides a productive use of the land which conforms to the approved land-use plan for the area. The short-term reclamation goals are to stabilize disturbed areas and to protect both disturbed and adjacent undisturbed areas from unnecessary or undue degradation.

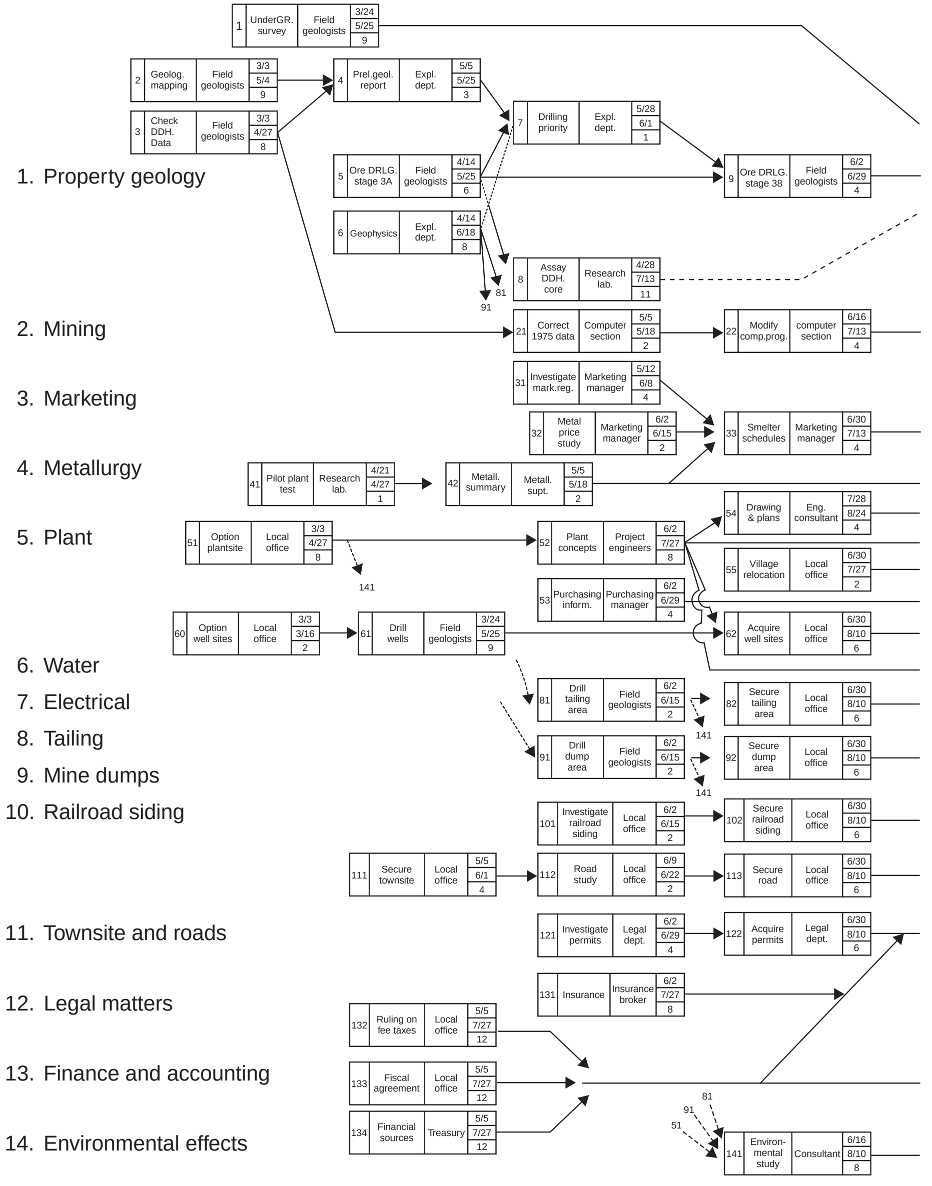
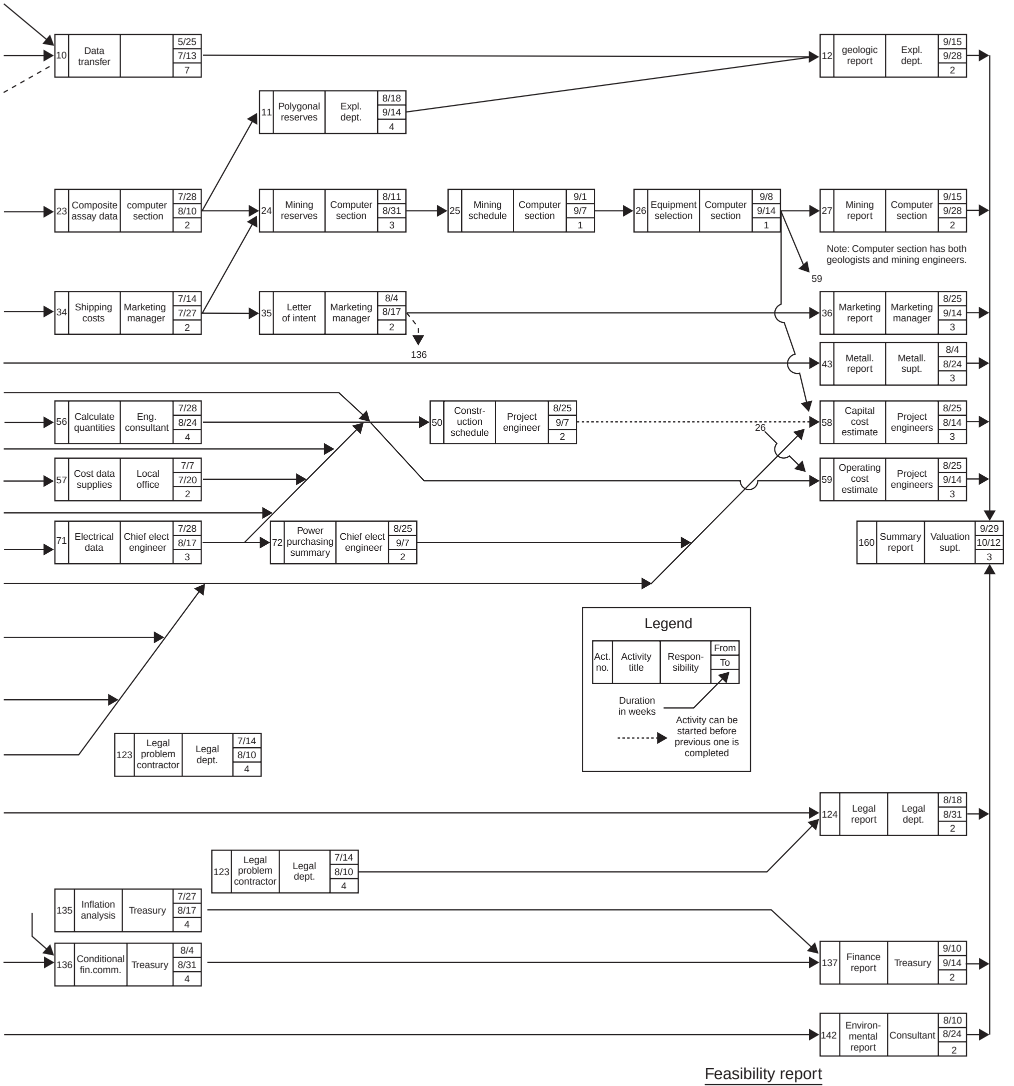


Figure 1.7. Activity network for a feasibility study (Taylor, 1977).



Feasibility report

Volumes:

1. Geology, ore reserves & mining
2. Metallurgy
3. Capital & operating cost estimates
4. Legal, finance & marketing
5. Summary & economic analysis

Figure 1.8. Simplified flow sheet showing the mining department activities.

1.9.3 *Reclamation plan purpose*

The purposes of the reclamation plan are as follows:

1. Reclamation plans provide detailed guidelines for the reclamation process and fulfill federal, state, county and other local agencies requirements. They can be used by regulatory agencies in their oversight roles to ensure that the reclamation measures are implemented, are appropriate for the site, and are environmentally sound.

2. Reclamation plans will be used by the operator throughout the operational period of the project and subsequent to cessation of exploration, mining, and processing activities. In turn, responsible agencies, including the BLM, will use the reclamation plan as a basis to review and evaluate the success of the reclamation program.

3. Reclamation plans should provide direction and standards to assist in monitoring and compliance evaluations.

1.9.4 *Reclamation plan content*

The reclamation plan should be a comprehensive document submitted with the plan of operations notice, exploration plan, or mining plan. A reclamation plan should provide the following:

1. A logical sequence of steps for completing the reclamation process.
2. The specifics of how reclamation standards will be achieved.
3. An estimate of specific costs of reclamation.
4. Sufficient information for development of a basis of inspection and enforcement of reclamation and criteria to be used to evaluate reclamation success and reclamation bond release.

The reclamation plan shall guide both the operator and the BLM toward a planned future condition of the disturbed area. This requires early coordination with the operator to produce a comprehensive plan. The reclamation plan will serve as a binding agreement between the operator and the regulatory agencies for the reclamation methodology and expected reclamation condition of the disturbed lands and should be periodically reviewed and modified as necessary.

Although the operator will usually develop the reclamation plan, appropriate pre-planning, data inventory, and involvement in the planning process by the regulatory agencies, is essential to determine the optimum reclamation proposal. Most determinations as to what is expected should be made before the reclamation plan is approved and implemented.

It is expected that there will be changes to planned reclamation procedures over the life of the project. Any changes will generally be limited to techniques and methodology needed to attain the goals set forth in the plan. These changes to the plan may result from oversights or omissions from the original reclamation plan, permitted alterations of project activities, procedural changes in planned reclamation as a result of information developed by on-site revegetation research undertaken by the operator and studies performed elsewhere, and/or changes in federal/state regulations. Specific requirements are given in the next section.

In preparing and reviewing reclamation plans, the BLM and the operator must set reasonable, achievable, and measurable reclamation goals which are not inconsistent with the established land-use plans. Achievable goals will ensure reclamation and encourage operators to conduct research on different aspects of reclamation for different environments. These goals should be based on available information and techniques, should offer incentives to both parties, and should, as a result, generate useful information for future use.

1.9.5 *Reclamation standards*

An interdisciplinary approach shall be used to analyze the physical, chemical, biological, climatic, and other site characteristics and make recommendations for the reclamation plan. In order for a disturbed area to be considered properly reclaimed, the following must be complied with:

1. Waste management. All undesirable materials (e.g. toxic subsoil, contaminated soil, drilling fluids, process residue, refuse, etc.) shall be isolated, removed, or buried, or otherwise disposed as appropriate, in a manner providing for long-term stability and in compliance with all applicable state and federal requirements:

(a) The area shall be protected from future contamination resulting from an operator's mining and reclamation activities.

(b) There shall be no contaminated materials remaining at or near the surface.

(c) Toxic substances that may contaminate air, water, soil, or prohibit plan growth shall be isolated, removed, buried or otherwise disposed of in an appropriate manner.

(d) Waste disposal practices and the reclamation of waste disposal facilities shall be conducted in conformance to applicable federal and state requirements.

2. Subsurface. The subsurface shall be properly stabilized, holes and underground workings properly plugged, when required, and subsurface integrity ensured subject to applicable federal and state requirements.

3. Site stability.

(a) The reclaimed area shall be stable and exhibit none of the following characteristics:
- Large rills or gullies.
- Perceptible soil movement or head cutting in drainages.
- Slope instability on or adjacent to the reclaimed area.

(b) The slope shall be stabilized using appropriate reshaping and earthwork measures, including proper placement of soils and other materials.

(c) Appropriate water courses and drainage features shall be established and stabilized.

4. Water management. The quality and integrity of affected ground and surface waters shall be protected as a part of mineral development and reclamation activities in accordance with applicable federal and state requirements:

(a) Appropriate hydrologic practices shall be used to protect and, if practical, enhance both the quality and quantity of impacted waters.

(b) Where appropriate, actions shall be taken to eliminate ground water co-mingling and contamination.

(c) Drill holes shall be plugged and underground openings, such as shafts, slopes, stopes, and adits, shall be closed in a manner which protects and isolates aquifers and prevents infiltration of surface waters, where appropriate.

(d) Waste disposal practices shall be designed and conducted to provide for long-term ground and surface water protection.

5. Soil management. Topsoil, selected subsoils, or other materials suitable as a growth medium shall be salvaged from areas to be disturbed and managed for later use in reclamation.

6. Erosion prevention. The surface area disturbed at any one time during the development of a project shall be kept to the minimum necessary and the disturbed areas reclaimed as soon as is practical (concurrent reclamation) to prevent unnecessary or undue degradation resulting from erosion:

(a) The soil surface must be stable and have adequate surface roughness to reduce run-off, capture rainfall and snow melt, and allow for the capture of windblown plant seeds.

(b) Additional short-term measures, such as the application of mulch or erosion netting, may be necessary to reduce surface soil movement and promote revegetation.

(c) Soil conservation measures, including surface manipulation, reduction in slope angle, revegetation, and water management techniques, shall be used.

(d) Sediment retention structures or devices shall be located as close to the source of sediment generating activities as possible to increase their effectiveness and reduce environmental impacts.

7. Revegetation. When the final landform is achieved, the surface shall be stabilized by vegetation or other means as soon as practical to reduce further soil erosion from wind or water, provide forage and cover, and reduce visual impacts. Specific criteria for evaluating revegetation success must be site-specific and included as a part of the reclamation plan:

(a) Vegetation production, species diversity, and cover (on unforested sites), shall approximate the surrounding undisturbed area.

(b) The vegetation shall stabilize the site and support the planned post-disturbance land use, provide natural plant community succession and development, and be capable of renewing itself. This shall be demonstrated by:

- Successful on-site establishment of the species included in the planting mixture and/or other desirable species.
- Evidence of vegetation reproduction, either spreading by rhizomatous species or seed production.
- Evidence of overall site stability and sustainability.

(c) Where revegetation is to be used, a diversity of vegetation species shall be used to establish a resilient, self-perpetuating ecosystem capable of supporting the postmining land use. Species planted shall includes those that will provide for quick soil stabilization, provide litter and nutrients for soil building, and are self-renewing. Except in extenuating circumstances, native species should be given preference in revegetation efforts.

(d) Species diversity should be selected to accommodate long-term land uses, such as rangeland and wildlife habitat, and to provide for a reduction in visual contrast.

(e) Fertilizers, other soil amendments, and irrigation shall be used only as necessary to provide for establishment and maintenance of a self-sustaining plant community.

(f) Seedlings and other young plants may require protection until they are fully established. Grazing and other intensive uses may be prohibited until the plant community is appropriately mature.

(g) Where revegetation is impractical or inconsistent with the surrounding undisturbed areas, other forms of surface stabilization, such as rock pavement, shall be used.

8. Visual resources. To the extent practicable, the reclaimed landscape should have characteristics that approximate or are compatible with the visual quality of the adjacent area with regard to location, scale, shape, color, and orientation of major landscape features.

9. Site protection. During and following reclamation activities the operator is responsible for monitoring and, if necessary, protecting the reclaimed landscape to help ensure reclamation success until the liability and bond are released.

10. Site-specific standards. All site-specific standards must be met in order for the site to be properly and adequately reclaimed.

1.9.6 *Surface and ground water management*

The hydrologic portion of the reclamation plan shall be designed in accordance with all federal, state, and local water quality standards, especially those under the Clean Water Act National Pollutant Discharge Elimination System (NPDES) point source and non point source programs.

The baseline survey should be conducted to identify the quantity and quality of all surface and subsurface waters which may be at risk from a proposed mineral operation. All aspects of an operation which may cause pollution need to be investigated, so that every phase of the operation can be designed to avoid contamination. It is better to avoid pollution rather than subsequently treat water. The diversion of water around chemically reactive mining areas or waste dumps must be considered during the planning stage. Site selection must be considered during the planning stage. Site selection for waste dumps should be conducted to minimize pollution.

Reclamation plans should be prepared to include a detailed discussion of the proposed surface water run-off and erosion controls including how surface run-off will be controlled during the ongoing operations, during interim shutdowns, and upon final closure.

Reclamation plans should also include a properly designed water monitoring program to ensure operator compliance with the approved plan. The purpose of the monitoring program is to determine the quantities and qualities of all waters which may be affected by mineral operations.

Operators should consider controlling all surface flows (i.e. run-on and run-off) with engineered structures, surface stabilization and early vegetative cover. Where the threat to the downstream water quality is high, the plan should provide for total containment, treatment, or both, if necessary, of the surface run-off on the project site. Sediment retention devices or structures should be located as near as possible to sediment source.

The physical control of water use and routing is a major task for mining projects.
The analysis includes the need to:

- Minimize the quantity of water used in mining and processing.
- Prevent contamination and degradation of all water.
- Intercept water so that it does not come in contact with pollutant generating sources.
- Intercept polluted water and divert it to the appropriate treatment facility.

Control may be complicated by the fact that many sources of water pollution are non point sources and the contaminated water is difficult to intercept.

1.9.7 *Mine waste management*

Handling of the waste materials generated during mining has a direct and substantial effect on the success of reclamation. Materials which will comprise the waste should be sampled and characterized for acid generation potential, reactivity, and other parameters of concern. Final waste handling should consider the selective placement of the overburden, spoils, or waste materials, and shaping the waste disposal areas. Creating special subsurface features (rock drains), sealing toxic materials, and grading or leveling the waste dumps are all waste handling techniques for enhancing reclamation. Any problems with the placement of waste discovered after the final handling will be very costly to rectify. Therefore, the selective placement of wastes must be considered during the mine plan review process in order to mitigate potential problems. Waste materials generated during mining are either placed in external waste dumps, used to backfill mined out pits, or used to construct roads, pads, dikes, etc. The design of waste management practices must be conducted in cooperation with the State, the Environmental Protection Agency (EPA), the BLM, other involved federal agencies and the operator.

The most common types of waste dumps include: (1) head of valley fills, (2) cross valley fills, (3) side hill dumps, and (4) flat land pile dumps. In the design and construction of large waste dumps it is important to consider appropriate reclamation performance standards for stability, drainage, and revegetation. Some guidance to consider during the mine plan review process includes the following:

1. Waste dumps should not be located within stream drainages or groundwater discharge areas unless engineered to provide adequate drainage to accommodate the expected maximum flow.

2. Waste dumps will be graded or contoured and designed for mass stability. Design criteria should include a geotechnical failure analysis. It is also recommended that prior to the construction of large waste dumps, a foundation analysis and geophysical testing be conducted on the dump site to ensure basal stability, especially on side hill dump locations. The effects of local groundwater conditions and other geohydrologic factors must be considered in the siting and designing of the dump.

3. Cross valley fills should provide for stream flow through the base of the dump. This is usually done using a rubble drain or french drain. At a minimum, the drain capacity should be capable of handling a design storm flow. To be effective, the drain must extend from the head of the upstream fill to the toe of the downstream face and should be constructed of coarse durable rock which will pass a standard slake test. Toxic or acid-producing materials should not be placed in valley fills.

4. Drainage should be diverted around or through head of valley and sidehill dumps.

5. Drains must be constructed of durable, nonslaking rock or gravel.

6. Topsoil or other suitable growth media should be removed from the proposed dump site and stockpiled for future use in reclamation.

7. Placement of coarse durable materials at the base and toe of the waste dump lowers the dump pore pressure and provides for additional internal hydrologic stability. An exception to this guidance would be in the case where the spoils materials exhibit high phytotoxic properties and the spoils must be sealed to prevent water percolation.

8. The finer textured waste materials which are more adaptable for use as a growing medium should be placed on the outside or mantel of the waste dump.

9. After the waste dump has been shaped, scarified, or otherwise treated to enhance reclamation, available topsoil or other selected subsoils should be spread over the surfaces of the dump as a growing medium. Grading and scarification may be required.

10. The dump should be designed to provide for controlled water flow which minimizes erosion and enhances structural stability.

11. Control erosion on long face slopes by requiring some form of slope-break mitigation, such as benches to intercept the flow of water or rock/brush terraces to slow down the velocity of the run-off.

12. Waste dump benches should be bermed or constructed wide enough to handle the peak design flows and to prevent overflowing onto the face of the dump in the event of freezing conditions. Dump benches should be constructed to allow for mass settling of the dump.

Safety requirements must be calculated for large waste dumps or waste embankments.

1.9.8 *Tailings and slime ponds*

Tailings and slime ponds consist of impounded mill wastes. Slime ponds are tailings ponds with high percentages of silts and clays, which cause very slow sediment drying conditions. Slime ponds are commonly associated with phosphate and bauxite processing operations. Reclamation of slime ponds is complicated by the slow dewatering.

Tailings impoundments are typically placed behind dams. Dams and the impounded wastes may require sealing on a case-by-case basis to avoid seepage below the dam or contamination of the groundwater. This measure only may be done before emplacement of the wastes. Long-term stability of the structure must be assured in order to guarantee ultimate reclamation success.

The nature of the tailings to be impounded should be determined as early as possible during the development of any plan. Tailings exhibiting phytotoxic or other undesirable physical or chemical properties will require a more complex reclamation plan. Analysis should include a thorough review of groundwater flow patterns in the area and a discussion of potential groundwater impacts. An impermeable liner or clay layer may be required to avoid contamination of groundwater. Where tailings include cyanide, final reclamation may include either extensive groundwater monitoring or pumpback wells and water treatment facilities to assure (ensure) groundwater quality is protected. The presence of cyanide in the tailings will not normally complicate reclamation of the surface.

1.9.9 *Cyanide heap and vat leach systems*

Dilute solutions of sodium cyanide (NaCN) or potassium cyanide (KCN) are used to extract precious metals from ores. Concentrations of cyanide solution utilized range from 300 to 500 ppm for heap leach operations to 2000 ppm (0.2%) for vat leach systems.

Low-grade ores can be economically leached in heaps placed on impermeable pads where a cyanide solution is sprinkled onto the ore. The solution preferentially collects the metals as it percolates downward and is recovered at the bottom of the heap through various means. Other metals besides gold and silver are mobilized by cyanide solutions.

Higher grade ores may be crushed, ground and agitated with cyanide solution in vats or tanks. The solids are then separated from the gold or silver-bearing (pregnant) solution. The precious metals are recovered from the pregnant solution and the solids are transferred to a tailings impoundment. The tailings are often deposited in a slurry form and may contain several hundred parts per million of cyanide.

Part of the overall mine reclamation plan includes cyanide detoxification of residual process solutions, ore heaps, tailings impoundments, and processing components.

A key to reclamation of cyanide facilities is planning for the solution neutralization process. The first step is to set a detoxification performance standard. This will have to be site specific dependent on the resources present and their susceptibility to cyanide and metal contamination. A minimum requirement would have to be the specific state standard. BLM may need to require more stringent standards if sensitive resources are present. Other considerations include the health advisory guideline used by EPA of 0.2 mg/l for cyanide in drinking water; and the freshwater chronic standard of 0.0052 mg/l for aquatic organisms. Some species of fish are especially sensitive to cyanide. Likewise metals, and other constituent levels, should be specified for detoxification of cyanide solutions.

There are a variety of methods for achieving detoxification of cyanide solutions. These range from simple natural degradation, to active chemical or physical treatment of process waters. A thorough understanding of the metallurgical process generating the waste, and of the chemistry of the waste stream is necessary to select the most effective cyanide destruction technique.

1.9.10 *Landform reclamation*

Shaping, grading, erosion control, and visual impact mitigation of an affected site are important considerations during review of the reclamation plan. The review process not only ensures that the topography of the reclaimed lands blend in as much as possible with the surrounding landforms, natural drainage patterns, and visual contrasts, but also enhances the success of revegetation.

The final landform should:

– be mechanically stable,
– promote successful revegetation,
– prevent wind and water erosion,
– be hydrologically compatible with the surrounding, landforms, and
– be visually compatible with the surrounding landforms.

Pit backfilling provides an effective means for reclamation of the disturbed lands to a productive post-mining land use. However, development of some commodities and deposit types may not be compatible with pit backfilling.

Open pit mine optimization is achieved by extending the pit to the point where the cost of removing overlying volumes of unmineralized 'waste' rock just equal the revenues (including profit) from the ore being mined in the walls and bottom of the pit. Because there is usually mineralization remaining, favorable changes in an economic factor (such as an increase in the price of the commodity or new technology resulting in a reduced operating cost) can result in a condition where mining can be expanded, or resumed at a future time. This economically determined pit configuration is typical of the open pit metal mining industry and is of critical importance in efforts to maximize the recovery of the mineral resource. To recover all the known ore reserves the entire pit must remain exposed through progressively deeper cuts. Backfilling where technologically and economically feasible, can not begin until the ore reserves within the specific pit are depleted at the conclusion of mining. Additionally, some waste material is not suitable for use as backfilling material.

Depending upon the size of the open pit, backfilling can extend the duration of operations from a few months to several years.

Final highwall configuration, including consideration of overall slope angle, bench width, bench height, etc., should be determined during the review of the plan. The maximum height of the highwall should be determined using site-specific parameters such as rock type and morphology. In most cases, the maximum height is regulated by various state agencies.

The normal procedures are to either leave the exposed highwall or to backfill and bury the highwall either totally or partially. Appropriate fencing or berming at the top of the highwall is necessary to abate some of the hazards to people and animals.

It is important that the backfill requirements be determined during the plan review process and included in the approved plan.

1.10 ENVIRONMENTAL PLANNING PROCEDURES

As described by Gilliland (1977), environmental planning consists of two distinct phases:

– Initial project evaluation,
– The strategic plan.

The components involved in each of these as extracted from the Gilliland paper will be outlined below.

1.10.1 *Initial project evaluation*

1. Prepare a detailed outline of the proposed action. This should include such items as drawings of land status, general arrangement of facilities, emission points and estimates of emission composition and quantities, and reclamation plans. It is also helpful to have information on the scope of possible future development and alternatives that might be available which could be accommodated within the scope of the proposed action.

For example, are there other acceptable locations for tailings disposal if the initial location cannot be environmentally marketed? A schedule for engineering and construction of the proposed action and possible future development should also be available.

2. Identify permit requirements. Certain permits can take many months to process and must be applied for well in advance of construction. Further, some permits will require extensive data, and very long lead times may be encountered in the collection of such data.

For example, biotic studies for environmental impact statements require at least a year, and sometimes longer, to evaluate seasonal changes in organisms. Are there points of conflict between permit requirements and the nature of the proposed action? Can the proposed action be altered to overcome these discrepancies or to avoid the need for permits that could be particularly difficult or significantly time-consuming to obtain? For example, a 'zero' effluent discharge facility could well avoid the Federal Water Pollution Control Action requirement for an Environmental Impact Statement (EIS).

3. Identify major environmental concerns. This includes potential on-site and off-site impacts of the proposed action and from possible future development. Land use and socio-economic issues as well as those of pollutional character must be taken into account. Although there may be little concern about the impacts of an exploratory activity itself, when bulldozers and drill rigs begin to move onto a property, it becomes apparent to the public that there may indeed ultimately be a full development of the property. Public concern may surface from speculation about the possible impacts of full development, and this could result in considerable difficulty in obtaining even the permits necessary to proceed with the proposed activity.

4. Evaluate the opportunity for and likelihood of public participation in the decision-making process. Recent administrative reforms provide for expanded opportunity for public participation in the decision-making process. Projects to be located in areas of minimal environmental sensitivity may stir little public interest and permits will not be delayed beyond their normal course of approval. A project threatening material impact to an area where the environmental resources are significant, however, will probably receive careful public scrutiny and may be challenged every step in the permit process.

5. Consider the amount and effect of delay possibly resulting from public participation during each state of the project. This could also be called intervention forecasting. When can a hearing be requested? When would it be possible for a citizen to bring suit? How long would it take to secure a final court action? Could the plaintiffs enjoin work on the project during the pendency of litigation? Can the project tolerate such delays? Can the project schedule be adjusted to live with such delays?

6. Evaluate the organization and effectiveness of local citizens groups. Attitudes are also part of this evaluation. Local citizen groups can be a powerful ally in positive communications with the public. They can also be effective adversaries. This evaluation should be extended to all groups which could have a significant voice in opinion making within a community. The working relationship of local groups with state or national counterpart groups should also be assessed.

7. Determine the attitudes and experience of governmental agencies. Identify any inter-agency conflicts. New ventures face an intricate web of federal, state, and local laws and regulations which are often complicated by inconsistencies in the policy goals which underlie these laws, and overlapping jurisdiction of the regulatory agencies.

Sometimes you must deal with personnel who have little knowledge of the business world or of the nature of operations being proposed. A company must be prepared, therefore, to dedicate considerable time and effort in promoting and understanding of the project.

Further, it is imperative for a company to recognize that government agency personnel have a public responsibility to see that the various laws and regulations within their jurisdiction are complied with. They may not always agree that the requirements of the law are practical, fair or equitable, but it is their job to ensure their applicability. Sometimes areas of apparent frustration or conflict will resolve themselves by re-evaluating your position with regard to the role that must be performed by regulatory personnel.

8. Consider previous industry experience in the area. This involves a determination of public attitudes toward previous or existing industry in the area and the posture and performance of these industries as a responsible member of the community. It is extremely helpful to the cause of your project if industry enjoys the status of being a good citizen. Where negative attitudes prevail, is there something about your project that could invite similar censure or could it be so designed to change these public attitudes?

9. Consider recent experience of other companies. Have new industries located or tried to locate within the area? Were there any issues involved relative to their success or failure to locate that might also be issues of concern to the proposed activity?

10. Identify possible local consultants and evaluate their ability and experience. Local consultants can be invaluable in assisting the company in many areas of inquiry. Their familiarity with the local scene on environmental, legal, socioeconomic, land use and other matters can enhance the credibility of a company's planning efforts and acceptability within a community.

11. Consider having a local consultant check the conclusions of the initial evaluation.

This initial project evaluation is essentially an identification procedure which in many instances can be largely produced in-house with possibly some modest assistance from outside consultants. Correspondingly, the cost could range from a thousand dollars or less to several thousand dollars depending upon the familiarity of personnel with this type of work and the amount of outside consultant help needed.

1.10.2 *The strategic plan*

Following the initial project evaluation the next step is to prepare a strategy or game plan for dealing with the identified issues. The elements would include:

1. Outline of technical information needed to obtain permits and to address legitimate environmental, land use and socioeconomic concerns. There are good reasons for the reluctance of planners to develop hard data before they are sure that they will be permitted to proceed with a project.

However, if a project is worthwhile, every practical effort must be made to develop information that demonstrates impacts have been carefully assessed, legitimate environmental concerns have been addressed, and controls and mitigation measures will be adequate to meet all existing standards and to protect the environment. In cases where standards are stringent and controls are not demonstrated technology, substantial extra effort may have to be made to develop predictions of performance. In case where better data cannot be developed without delaying construction, plans may have to include a proposal for eventually securing such data and adjusting permit requirements before operations begin.

Where predictive data are not practically obtainable, a plan might provide for operational monitoring with post-startup alteration of permit requirements if problems arise. This plan element, therefore, provides a specific checklist for the information gathering system.

2. Categorically assign responsibilities for the acquisition of the technical information and hire necessary consultants. Coordinate this work with governmental agencies when appropriate.

The primary responsibility for each element of data collection should be clearly designated so that misunderstandings do not arise. Governmental agencies can be an important source of background information on air quality, water quality and other pertinent data. Further, government studies may be intended or in progress which in scope would include the location and environmental concerns of the company's proposed action. Data collection by the company could complement these studies and vice versa.

3. Prepare a schedule for obtaining information and data and for submitting permit applications to the appropriate agencies. Firm target dates must be established for the finalization of reports, permit applications or other necessary authorizations. Interim reporting periods should also be set to ascertain the status of progress and to provide whatever adjustments are necessary to keep on the appropriate schedule. A critical path chart would include a display of this sequence. If a project is properly planned, its proponents require nothing more from government except even handed operation of the approval mechanism.

4. Select local legal, technical and public relations consultants. Sometimes the local consultants may be those who will be directly involved in the data development. In other instances, these consultants would have more of a role in planning, data evaluation and public communications.

5. Avoid hostile confrontations with environmental groups. There is nothing to be gained from a shouting match where both sides become so highly polarized that reason and credibility cannot be maintained. No-growth advocates will probably continue to be unyielding in their opposition no matter how much progress is made in devising effective environmental controls.

Project planners who view citizen opposition as monolithic and implacable miss, however, an opportunity to reduce the risks of intervention and delay. Citizen attitudes are subject to change, and many citizen activists are sincerely, and very properly, seeking to secure for themselves and others the maintenance of a quality environment.

If the proposed activity is demonstrably sound, both industrially and environmentally, and the public has access to all the facts, it is likely that people will make sound judgements and that mineral development will be permitted.

6. Develop a consistent program for the generation of credible factual information. Good factual information needed to refute or substantiate concerns regarding possible impacts of the proposed action or future development is not always available. Such deficiencies are not uncommon or unacceptable if they are honestly faced and a program is designed to acquire the necessary information. Many projects have been seriously delayed or stopped because of a company's failure to admit that a concern exists. This can become a focal point for attacking the credibility of a company's entire program.

1.10.3 *The environmental planning team*

The environmental planning effort, due to the wide diversity of tasks involved requires the participation of many specialists drawn from various functional areas of mining organizations and from outside consulting firms. To coordinate this effort there must be a team leader who has the perspective to understand the requirements of the disciplines involved and the eventual use of the information evolved. This team leader must also have the acknowledged responsibility and authority for the performance of this coordinating role.

Table 1.4. Types of permits and approvals which may be required for the Kensington Gold Project (Forest Service, 1990).

1. *Federal government*

Forest Service
1. NEPA compliance and record of decision on EIS
2. Plan of operations
3. Special use permits

Environmental Protection Agency
1. National Pollutant Discharge Elimination System (NPDES)
2. Spill Prevention Control and Countermeasure (SPCC) plan
3. Review of section 404 Permit
4. Notification of hazardous wages activity
5. NEPA compliance and record of decision on EIS (cooperating agency)

Army Corps of Engineers
1. Section 404 Permit – Clean Water Act (dredge and fill)
2. Section 10 Permit – Rivers and Harbor Act
3. NEPA compliance and record of decision on EIS (cooperating agency)

Coast Guard
1. Notice of fueling operations
2. Permit to handle hazardous materials
3. Application for private aids to navigation

Federal Aviation Administration
1. Notice of landing area and certification of operation
2. Determination of no hazard

Federal Communications Commission
1. Radio and microwave station authorizations

Treasury Department (Dept of Alcohol, Tobacco & Firearms)
1. Explosives user permit

Mine Safety and Health Administration
1. Mine I.D. number
2. Legal identity report
3. Miner training plan approval

U.S. Fish and Wildlife Service
1. Threatened and endangered species clearance
2. Bald Eagle Protection Act clearance

National Marine Fisheries Services
1. Threatened and endangered species clearance

2. *State of Alaska*

Alaska Division of Government Coordination
1. Coastal project questionnaire
2. Coastal management program certification

Alaska Department of Environmental Conservation
1. Air quality permit
2. Burning permit
3. Certification of reasonable assurance
4. Solid Waste Management permit
5. Oil facilities approval of financial responsibility
6. Oil facilities discharge contingency plan
7. Water and sewer plan approval
8. Food service permit

Alaska Department of Natural Resources
1. Water rights permits
2. Tidelands lease
3. Right-of-way permit
4. Permit to construct or modify a dam
5. Land use permit

Alaska Department of Fish and Game
1. Fishway or fish passage permit
2. Anadromous fish protection permit

Alaska Department of Public Safety
1. Life and fire safety plan check

Alaska Department of Labor
1. Fired and unfired pressure vessel certificate
2. Elevator certificate of operation

Alaska Department of Revenue
1. Affidavit for non-resident business taxation
2. Alaska business license
3. Alaska mining license

Alaska Department of Health and Social Services
1. Health care facilities construction license
2. Certificate of need (townsite with health care facilities)

3. *Local government*

City and Bureau of Juneau
1. Mining permit
2. Grading permit
3. Building permits
4. Burning permits
5. Explosive permits

The team members include such personnel as the project manager, project engineers, attorneys, environmental specialists, technical and public relations experts.

1.11 A SAMPLE LIST OF PROJECT PERMITS AND APPROVALS

The 'Final Scoping Document, Environmental Impact Statement' for the Kensington Gold Project located near Juneau, Alaskawas published by the U.S. Forest Service (Juneau Ranger District) in July 1990 (Forest Service, 1990). To provide the reader with an appreciation for the level of effort involved just in the permitting process, a listing of the various federal, state, and local government permits/approvals which may be required for this underground gold mine/mill, is given in Table 1.4.

REFERENCES AND BIBLIOGRAPHY

Anonymous. 1976. Principles of the Mineral Resource Classification System of the U.S. Bureau of Mines and U.S. Geological Survey. Geological Survey Bulletin 1450-A.Washington: U.S. Government Printing Office.

Anonymous. 1980. Principles of a Resource/Reserve Classification for Minerals. Geological Survey Circular 831. Arlington, VA: U.S. Geological Survey.

Anonymous. 1998. A heavy metal harvest for the millennium. E/MJ. 199(11): 32YY–32ZZ.

AusIMM. 1995. Code and guidelines for technical assessment and/or valuation of mineral and petroleum assets and mineral and petroleum securities for independent expert reports (The VALMIN Code). http://www.ausimm.com.au/codes/valmin/valcode0.asp.

AusIMM. 1998a. The revised and updated VALMIN Code: Code and guidelines for technical assessment and/or valuation of mineral and petroleum assets and mineral and petroleum securities for independent expert reports (The VALMIN Code). http://www.ausimm.com.au/codes/valmin.

AusIMM. 1998b. The revised VALMIN Code and guidelines: An aide memoire to assist its interpretation. http://www.ausimm.com.au/codes/valmin/eexp11.asp.

AusIMM. 2004. JORC information. www.jorc.org/main.

AusIMM. 2005. Australasian Code for the reporting of exploration results, mineral resources and ore reserves: The JORC Code – 2004 Edition. www.jorc.org/pdf/jorc2004print.pdf.

Banfield, A.F. 1972. Ore reserves, feasibility studies and valuations of mineral properties. Paper presented at the AIME Annual Meeting, San Francisco, CA February 20–24, 1972. Society of Mining Engineers of AIME, Preprint 72-AK-87.

Banfield, A.F., and J.F. Havard. 1975. Let's define our terms in mineral valuation. *Mining Engineering* 27(7): 74–78.

Barfoot, G. 2007. Challenges for project management on long-term iron ore expansion projects. Iron Ore 2007 Conference, Proceedings, AusIMM Publication Series 6/2007. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 223–224.

BLM (Bureau of Land Management) 1992. *Solid Minerals Reclamation Handbook* (BLM Manual Handbook H-3042-1). U.S. Department of the Interior.

Böde, K. 1999. KUBUS – Estimating and controlling system for project management in the construction and construction-related industry. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 155–164. Colorado School of Mines:CSM.

Bourgouin, F. 2011. The politics of large-scale mining in Africa: domestic policy, donors, and global economic processes. JSAIMM. 111(07):525–529.

Brennan, J., and A. Lawrence. 2005. Taking it to the bank – Making your mining project bankable. E/MJ. 206(7):80–83. September.

Bullock, R.L. 2011. Accuracy of feasibility study evaluation would improve accountability. Mining Engineering. 63(4):78–85. April.

Camus, J. 2011. Value creation in the mining business. Mining Engineering. 63(3):43–52. March.

Cawood, F.T. 2011. Threats to the South African minerals sector – an independent view on the investment environment for mining. JSAIMM. 111(7):469–474.
Chow, J. 2007. Cadia East open pit project mining study, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 37–44.
CIM. 2000. Exploration Best Practice Guidelines. Aug 20. www.cim.org/definitions/explorationBEST PRACTICE.pdf.
CIM. 2003a. Standards and guidelines for valuation of mineral properties. February. www.cim.org/committees/CIMVal_final_standards.pdf.
CIM. 2003b. Guidelines for the reporting of diamond exploration results. Mar 9. www.cim.org/committees/diamond_exploration_final.cfm.
CIM. 2003c. Estimation of mineral resources and mineral reserves: Best practice guidelines. Nov 23. www.cim.org/committees/estimation.cfm.
CIM. 2004. Definition Standards – On mineral resources and mineral reserves. www.cim.org/committees/StdsAppNovpdf.
CIM. 2005. Standards and guidelines. www.cim.org/committees/guidelinesStandards_main.cfm.
Clegg, A.M. 2005. Risk in project preparation – 'Determining the project needs and the effects on the management strategy'. JSAIMM. 105(2):81–94.
Danilkewich, H., Mann, T., and G. Wahl. 2002. Preparing a feasibility study request for proposal in the 21st century. Pre-print Paper 02-101. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.
Danni, J.W. 1992. Mineral Hill Mine – A case study in corporate environmentalism. *Mining Engineering*. 44(1): 50–53.
Davis, G. 1994. U.S. share of world mineral markets: Where are we headed. *Mining Engineering*. 46(9): 1067–1069.
Dessureault, S., Scoble, M., and S. Dunbar. 2002. Activity based costing and information engineering: Formulation of an annual budget. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 601–614. Alaska:SME.
De Voto, R.H., and T.P. McNulty. 2000. Banning cyanide use at McDonald – an attack on open-pit mining. *Mining Engineering.* 52(12): 19–27.
Evans, D. 2008. Analyzing the risk of bankable feasibility studies in today's mining supercycle. E/MJ. 209(7):92–94. September.
Forest Service (Juneau Ranger District) 1990. Kensington Gold Project, Alaska: Final Scoping Document, Environmental Impact Statement. U.S. Department of Agriculture (7).
Francis, D. 1997. Bre-X: The Inside Story. Key Porter Books, Toronto. 240 pp.
Gavelan, Z., and S. Dessureault. 2006. Probabilistic approach to project-specific political risk analysis for mineral projects. Mining Engineering. 58(1):43–49. January.
Gilliland, J.C. 1977. The environmental requirements of mine planning. *Mineral Industry Costs.* Northwest Mining Association: 57–65.
Goold, D., and A. Willis. 1997. Bre-X Fraud. McClelland & Stewart. 272 pp.
Grace, K.A. 1984. Reserves, resources and pie-in-the-sky. *Mining Engineering* 36(10): 1446–1450; 1985. Discussion. *Mining Engineering* 37(8): 1069–1072.
Halls, J.L. 1975. *Personal communication*. Kennecott Copper Company.
Hatton, R. 2005. Computer simulations to enable continuous blending to two plants at Sishen mine From the medium-term point of view, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 281–289.
Heuberger, R. 2005. Risk analysis in the mining industry. JSAIMM. 105(2):75–80.
Hickson, R.J. 2000. Project management for dummies, or "How to improve your project success ratio in the new millennium". Paper 00-133. SME Annual Meeting. Feb 28–Mar 1. SLC, UT.
Hustrulid, W. 2002. 2002 Jackling Lecture: You can't make a silk purse out of a sow's ear. *Mining Engineering*. 54(9): 41–48.
Hutson, H. 2009. Application of the best available technology to reclamation design and integration with mine planning. Paper 09-085. SME Annual Meeting. Feb 22–25. Denver, CO.
Jacus, J.R., and T.E. Root. 1991. The emerging federal law of mine waste: administrative, judicial and legislative developments. Land and Water Law Review, University of Wyoming, vol. XXVI, n. 2, 1991.
Johnson, K.R. 2011. Visual computing and mine design. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 39–46.

Jorgenson, J.D. 2005. Challenges facing the North American iron ore industry. Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 395–403.

Kelly, R. 2009. The changing face of project evaluation – Meeting the demands of changing projects, project owners and evaluation criteria. Project Evaluation 2009. Melbourne, Vic. April 21–22. pp. 189–197.

Kelly, T.D. 2002. Raw materials and technology fuel U.S. economic growth. *Mining Engineering*. 54(12): 17–21.

King, B.M. 1998. Impact of rehabilitation and closure costs on production rate and cut-off grade strategy. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 617–630. London:IMM.

Kirk, S.J., and T.J. O'Neil. 1983. Preplanning of end land uses for an open pit copper mine. *Mining Engineering* 35(8): 1191–1195.

Kirk, W.S. 1998. Iron ore reserves and mineral resource classification systems. Skillings Mining Review. June 6. pp. 4–11.

Kral, S. 1993. Risk assessment/management in the environmental planning of mines. *Mining Engineering* 45(2): 151–154.

Lai, F.J., and W. Strange. 2009. Using value at risk for integrated project risk evaluation. Project Evaluation Conference. Melbourne, Vic. April 21–22. pp. 223–232.

Lee, T.D. 1984. Planning and mine feasibility study – An owners perspective. In: *Proceedings of the 1984 NWMA Short Course 'Mine Feasibility – Concept to Completion'* (G.E. McKelvey, compiler). Spokane, WA.

Lee, T.D. 1991. Personal communication. 12/10/91.

Leon, P.S.G. 2011. International best practice and resource nationalism: the International Bar Association. JSAIMM. 111(7):517–524.

Leonardson, R.W., Weakly, C.G., Lander, A.M., and P.B. Zohar. 2006. Exploring between drill holes yields new ounces at Goldstrike. Mining Engineering. 58(12):37–40. December.

Lilford, E.V. 2006. The corporate cost of capital. JSAIMM. 106(2):139–146.

Love, S. 2009. On the problem of sponsor bias and the unacceptable variance between projected and actual outcomes for mining projects. AusIMM Journal. Issue 1. Feb.

Marcus, J. 1990. Mining environment – regulatory control of mining at the federal level, part 1. *E&MJ* 191(6).

Martinez, L.A. 2009. Why Accounting for uncertainty and risk can improve final decision-making in strategic open pit mine evaluation. Project Evaluation Conference, Melbourne, Vic, April 21–22. pp. 113–124.

McKelvey, G.E. 1984. Mineral Exploration. *Mine Feasibility – Concept to Completion* (G.E. McKelvey, compiler). Spokane, WA: Northwest Mining Association.

McKenzie, B.W. 1980. Looking for the improbable needle in a haystack. *The Economics of Base Metal Exploration in Canada.* Queens University, Ontario: Working Paper No. 19.

Micaux, S. 2011. Three caveats that will change your design culture. AusIMM Journal. Issue 6. Dec.

Miller, J.D. 1984. Legal considerations in mine planning and feasibility. *Mine Feasibility – Concept to Completion* (G.E. McKelvey, compiler). Spokane, WA: Northwest Mining Association.

Moore, R.T. 1984. Environmental/Permitting. *Mine Feasibility – Concept to Completion* (G.E. McKelvey, compiler). Northwest Mining Association.

Noble, A.C. 1993. Geologic resources vs. ore reserves. *Mining Engineering* 45(2): 173–176.

Omizzolo, S. 2010. Conference report: Seventh large open pit mining conference 2010. AusIMM Journal. Issue 5. Oct.

Park, S-J., and I.I. Matunhire. 2011. Investigation of factors influencing the determination of discount rate in the economic evaluation of mineral development projects. JSAIMM. 111(11):773–814.

Paulsen, K.R. 1982. Environmental and regulatory costs of mining projects. *Mineral Industry Costs* (J.R. Hoskins, compiler): 27–32. Northwest Mining Association.

Ramsden, M., Bitencourt R., and J. Austin. 2005. Geological solutions to short-term planning in iron ore – A Pilbara iron example, Mt Tom Price. Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 129–135.

Rendu, J-M. 2003. SME meets with the SEC – resources and reserves reporting discussed. *Mining Engineering*. July. P35.

Rogers, J.P. 1999. ProjectWorks® – Bechtel's automated solution for projects. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 165–172. Colorado School of Mines:CSM.

SAIMM. 2000. South African Code for reporting of mineral resources and mineral reserves (the SAMREC Code). www.saimm.co.za/pages/comppages/samrec_version.pdf.
Samimi, N.F., Shahriar, K., Dehghan J., and H. Abadi. 2007. Feasibility Study and Economic Evaluation for Deposit No. 3 of Gol-e-Gohar Iron Ore Deposit, AusIMM, Iron Ore 2007 Conference Proceedings, Perth, WA, August 2007, pp. 381–385.
Sarder, M. 2009. Rio Tinto and BHP Billiton iron ore joint venture: The deal and what others said. AusIMM Journal. Issue 4. Aug.
SEC. 2005. Description of property by users engaged or to be engaged in significant mining operations. Guide 7. www.sec.gov/divisions/corpfin/forms/industry.htm#secguide7/.
SME (Society of Mining, Metallurgy and Exploration) 1991. A guide for reporting exploration information, resources and reserves. *Mining Engineering* 43(4): 379–384.
SME. 1991. A guide for reporting exploration information, resources and reserves. *Mining Engineering* April. pp. 379–384.
SME. 1999. A guide for reporting exploration information, mineral resources, and mineral reserves. www.smenet.org.digital_library/index.cfm.
SME. 2003. Reporting mineral resources and reserves: Regulatory, financial, legal, accounting, managerial and other aspects. Conference October 1–3, 2003. SME.
Solomon, M.H. 2011. A conceptual approach to evaluating the political economics of mining in Africa and the sector. JSAIMM. 111(7):475–492.
Stewart, D.H. 2007. Old Iron, New Iron – A Story of Revival at Frances Creek, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 19–26.
Sykes, T. 1978. The Money Miners: Australia's Mining Boom 1969–1970. Wildcat Press, Sydney. 388 pages.
Sykes, T. 1988. Two Centuries of Panic: A History of Corporate Collapses in Australia. Allen & Unwin, London. 593 pages.
Sykes, T. 1996. The Official History of Blue Sky Mines. Australian Financial Review. 165 pages.
Taylor, H.K. 1977. Mine valuation and feasibility studies. *Mineral Industry Costs* (J.R. Hoskins & W.R. Green, editors): 1–17. Spokane, WA: Northwest Mining Association.
U.S. Department of Agriculture (Forest Service) 1988. Best minerals management practices – a guide to resource management and reclamation of mined lands in the Black Hills of South Dakota and Wyoming.
U.S. Department of the Interior (Office of Surface Mining Reclamation and Enforcement) 1988. Surface Mining Control and Reclamation Act of 1977. (Public Law 95-87).
USBM. 1967. *A Dictionary of Mining, Mineral and Related Terms* (Paul W. Thrush, editor). Washington: U.S. Department of the Interior.
USGS. 1976. Principles of the mineral resource classification system of the U.S. Bureau of Mines and U.S. Geological Survey. Bulletin 1450-A.
USGS. 1980. Principles of a resource/reserve classification for minerals. USGS Circular 831. http://pubs.er.usgs.gov/pubs/cir/cir831/ or http://imcg.wr.usgs.gov/usbmak/c831.html.
Wilkinson, W.A., Jr., and J. Buenemann. 2010. Case study – Benefits of building an efficient mine planning process. Paper 10–037. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.
Willis, W.B. 2005. The Development of a Niche Iron Ore Producer, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 441–445.
Wober, H.H., and P.J. Morgan. 1993. Classification of ore reserves based on geostatistical and economic parameters. *CIM Bulletin* 86(966): 73–76.

REVIEW QUESTIONS AND EXERCISES

1. What is meant by ore?
2. Express the meaning of 'profit' in your own words. How does it relate to your future opportunities?
3. Define
   - Exploration
   - Development
   - Production

4. Discuss the changes that have occurred between the SME 1991 and SME 1999 guidelines regarding the 'Reporting of Exploration Information, Resources and Reserves.' See the Reference section of this chapter.
5. Distinguish the meanings of 'Resource' and 'Reserve.'
6. Using Figure 1.1 discuss the basis upon which 'Resources' and 'Reserves' change category.
7. The U.S. Securities and Exchange Commission (SEC) have their own guidelines regarding the public reporting of resources and reserves. Referring to the website provided in the References, summarize their requirements. How do they compare to the SME 1991 guidelines? To the 1999 guidelines?
8. The most recent version of the USGS/USBM classification system is published as USGS Circular 831. Download the Circular from their website (see References). What was the main purpose of these guidelines? Who was the intended customer?
9. What is meant by 'Hypothetical Resources'?
10. What is meant by 'Undiscovered Resources'?
11. In the recent Bre-X scandal, the basis for their resource/reserve reporting was indicated to be USGS Circular 831. In which of the classification categories would the Bre-X resources/reserves fall? Explain your answer. See the References for the website.
12. Compare the 1999 SME guidelines with those provided in the JORC code included in Chapter 7.
13. Compare the 1999 SME guidelines with those provided in the CIM guidelines included in Chapter 7.
14. Discuss the relevance of the Mineral Supply Process depicted in Figure 1.2 to iron ore for the period 2002 to 2005.
15. Discuss the relevance of the Mineral Supply Process depicted in Figure 1.2 to molybdenum for the period 2002 to 2005.
16. Discuss the relevance of the Mineral Supply Process depicted in Figure 1.2 to copper for the period 2002 to 2005.
17. Figure 1.3 shows diagrammatically the planning, implementation and production phases for a new mining operation. What are the planning stages? What are the implementation stages? What are the production stages?
18. Does the 'Relative Ability to Influence Cost' curve shown in Figure 1.3 make sense? Why or why not?
19. What is the fourth phase that should be added to Figure 1.3?
20. In the initial planning stages for any new project there are a great number of factors of rather diverse nature that must be considered. The development of a 'checklist' is often a very helpful planning tool. Combine the items included in the checklist given in section 1.3 with those provided by Gentry and O'Neil on pages 395–396 of the SME Mining Engineering Handbook (2nd edition, Volume 1).
21. How might the list compiled in problem 20 be used to guide the preparation of a senior thesis in mining engineering?
22. What is the meaning of a 'bankable' mining study?
23. Summarize the differences between a conceptual study, a pre-feasibility study and a feasibility study.
24. Assume that the capstone senior mine design course extends over two semesters each of which is 16 weeks in duration. Using the information provided in Tables 1.3 and 1.4 regarding the content of an intermediate valuation report (pre-feasibility study) and a

feasibility study, respectively, develop a detailed series of deliverables and milestones. It is suggested that you scan the two tables and cut-and-paste/edit to arrive at your final product.

25. Assume that the estimated capital cost for an open pit project is $500 million. How much would you expect the conceptual study, the preliminary study and the feasibility study to cost?
26. Section 1.6 concerns the accuracy of the estimates provided. These are discussed with respect to tonnage and grade, performance, costs, and price and revenue. Summarize each.
27. Discuss what is meant by the contingency allowance. What is it intended to cover? What is it not meant to cover?
28. In section 1.6.4 it is indicated that both probable and average metal prices expressed in present value dollars need to be provided. The 'conservative' price is considered to be that with an 80% probability of applying. Choose a mineral commodity and assign a probable price and a conservative price for use in a pre-feasibility study. Justify your choices.
29. What are the two common ways for accomplishing a feasibility study?
30. Summarize the steps involved in performing a feasibility study. What is the function of the steering committee? Who are the members?
31. Who are the members of the project teams?
32. What is meant by a Work Breakdown Structure? What is its purpose?
33. What is the difference between a Work Breakdown Structure and a Work Classification Structure?
34. Construct a bar chart for the activities listed in the project schedule developed in problem 24. It should be of the type shown in Figure 1.6.
35. What is meant by an RFP? How should they be structured?
36. What is the goal of a Critical Path representation?
37. Section 1.9 deals with mine reclamation. What is the rationale for including this material in Chapter 1 of this book and not later?
38. What is the concept of multiple use management? What is its application to minerals?
39. What is the practical implication of the statement from the Mining and Minerals Policy act of 1970?
40. What is the U.S. National Materials and Mineral Policy, Research and Development Act of 1980 (Public Law 96-479)? Is it being followed today?
41. Define the following acronyms:
    a. BLM
    b. FLPMA
    c. CFR
    d. NEPA
    e. EA
    f. EIS
    g. NPDES
    h. EPA
42. Summarize the purposes of a reclamation plan.
43. What should a reclamation plan contain?
44. For a disturbed area to be properly reclaimed, what must be achieved? Summarize the major concepts.

45. Discuss the most important concepts regarding surface and ground water management.
46. Discuss the important concepts regarding mine waste management.
47. Tailings and slime ponds. What is the difference between them? What are the engineering concerns?
48. What means are available for the detoxification of cyanide heap and vat leach systems?
49. What is meant by landform reclamation?
50. Do open pit mines have to be refilled? Discuss the pro's and con's regarding the backfilling of open pit mines.
51. In section 1.10 'Environmental Planning Procedures' Gilliland divided environmental planning into two distinct phases: (1) Initial project evaluation and (2) The strategic plan. Summarize the most important aspects of each.
52. What members would be part of an environmental planning team?
53. In section 1.11 a list of the project permits and approvals required for the Kensington Gold project in Alaska has been provided. List the permits and approvals required for a new mining project in your state/country.

CHAPTER 2

Mining revenues and costs

2.1 INTRODUCTION

For one to know whether the material under consideration is 'ore' or simply 'mineralized rock', both the revenues and the costs must be examined. It is the main objective of this chapter to explore in some detail each of these topics.

2.2 ECONOMIC CONCEPTS INCLUDING CASH FLOW

In Chapter 6, the production planning portion of this text, an economic basis will be used to select production rate, mine life, etc. This section has been included to support that chapter. It is not intended to be a textbook complete in itself but rather to demonstrate some of the important concepts and terms.

2.2.1 *Future worth*

If someone puts \$1 in a savings account today at a bank paying 10% simple interest, at the end of year 1 the depositor would have \$1.10 in his account. This can be written as

$$\mathrm{FW} = \mathrm{PV}(1 + i) \tag{2.1}$$

where FW is the future worth, PV is the present value, i is the interest rate.

If the money is left in the account, the entire amount (principal plus interest) would draw interest. At the end of year 2, the account would contain \$1.21. This is calculated using

$$\mathrm{FW} = \mathrm{PV}(1 + i)(1 + i)$$

At the end of year n, the accumulated amount would be

$$\mathrm{FW} = \mathrm{PV}(1 + i)^n \tag{2.2}$$

In this case if $n = 5$ years, then

$$FW = \$1(1 + 0.10)^5 = \$1.61$$

2.2.2 *Present value*

The future worth calculation procedure can now be reversed by asking the question 'What is the present value of \$1.61 deposited in the bank 5 years hence assuming an interest rate of 10%?' The formula is rewritten in the form

$$PV = \frac{FW}{(1+i)^n} \tag{2.3}$$

Substituting FW = \$1.61, $i = 0.10$, and $n = 5$ one finds as expected that the present value is

$$PV = \frac{\$1.61}{(1+0.10)^5} = \$1$$

2.2.3 *Present value of a series of uniform contributions*

Assume that \$1 is to be deposited in the bank at the end of 5 consecutive years. Assuming an interest rate of 10%, one can calculate the present value of each of these payments. These individual present values can then be summed to get the total.

Year 1: Payment

$$PV_1 = \frac{\$1}{(1.10)^1} = \$0.909$$

Year 2: Payment

$$PV_2 = \frac{\$1}{(1.10)^2} = \$0.826$$

Year 3: Payment

$$PV_3 = \frac{\$1}{(1.10)^3} = \$0.751$$

Year 4: Payment

$$PV_4 = \frac{\$1}{(1.10)^4} = \$0.683$$

Year 5: Payment

$$PV_5 = \frac{\$1}{(1.10)^5} = \$0.621$$

The present value of these 5 payments is

$$PV = \$3.790$$

The general formula for calculating the present value of such equal yearly payments is

$$PV = FW\left[\frac{(1+i)^n - 1}{i(1+i)^n}\right] \tag{2.4}$$

Applying the formula in this case yields

$$\mathrm{PV} = \$1\left[\frac{(1.10)^5 - 1}{(0.10)(1.10)^5}\right] = \$3.791$$

The difference in the results is due to roundoff.

2.2.4 *Payback period*

Assume that \$5 is borrowed from the bank today (time = 0) to purchase a piece of equipment and that a 10% interest rate applies. It is intended to repay the loan in equal yearly payments of \$1. The question is 'How long will it take to repay the loan?' This is called the payback period. The present value of the loan is

$$\mathrm{PV\ (loan)} = -\$5$$

The present value of the payments is

$$\mathrm{PV\ (payments)} = \$1\left[\frac{(1.10)^n - 1}{0.10(1.10)^n}\right]$$

The loan has been repaid when the net present value

$$\text{Net present value (NPV)} = \text{PV (loan)} + \text{PV (payments)}$$

is equal to zero. In this case, one substitutes different values of n into the formula

$$\mathrm{NPV} = -\$5 + \$1\left[\frac{(1.10)^n - 1}{0.10(1.10)^n}\right]$$

For $n = 5$ years NPV = −\$1.209; for $n = 6$ years NPV = −\$0.645; for $n = 7$ years NPV = −\$0.132; for $n = 8$ years NPV = \$0.335.

Thus the payback period would be slightly more than 7 years ($n \cong 7.25$ years).

2.2.5 *Rate of return on an investment*

Assume that \$1 is invested in a piece of equipment at time = 0. After tax profits of \$1 will be generated through its use for each of the next 10 years. If the \$5 had been placed in a bank at an interest rate of i then its value at the end of 10 years would have been using Equation (2.2).

$$\mathrm{FW} = \mathrm{PV}(1 + i)^n = \$5(1 + i)^{10}$$

The future worth (at the end of 10 years) of the yearly \$1 after tax profits is

$$\mathrm{FW} = A_m\left[\frac{(1 + i)^n - 1}{i}\right] \tag{2.5}$$

where A_m is the annual amount and $[(1+i)^n - 1]/i$ is the uniform series compound amount factor.

The interest rate i which makes the future worths equal is called the rate of return (ROR) on the investment.

In this case

$$\$5(1+i)^{10} = \$1\left[\frac{(1+i)^{10}-1}{i}\right]$$

Solving for i one finds that

$$i \cong 0.15$$

The rate of return is therefore 15%. One can similarly find the interest rate which makes the net present value of the payments and the investment equal to zero at time $t = 0$.

$$\text{NPV} = -\$5 + \$1\left[\frac{(1.10)^{10} - 1}{i(1+i)^{10}}\right] = 0$$

$$i \cong 0.15$$

The answer is the same.

The process of bringing the future payments back to time zero is called 'discounting'.

2.2.6 *Cash flow (CF)*

The term 'cash flow' refers to the net inflow or outflow of money that occurs during a specific time period. The representation using the word equation written vertically for an elementary cash flow calculation is

Gross revenue
– Operating expense
= Gross profit (taxable income)
– Tax
= Net profit
– Capital costs

= Cash flow

A simple example (after Stermole & Stermole, 1987) is given in Table 2.1.

In this case there is a capital expense of \$200 incurred at time $t = 0$ and another \$100 at the end of the first year. There are positive cash flows for years 2 through 6.

Table 2.1. Simple cash flow example (Stermole & Stermole, 1987).

Year	0	1	2	3	4	5	6
Revenue			170	200	230	260	290
– Operating cost			−40	−50	−60	−70	−80
– Capital costs	−200	−100					
– Tax costs			−30	−40	−50	−60	−70
Project cash flow	−200	−100	+100	+110	+120	+130	+140

2.2.7 *Discounted cash flow (DCF)*

To 'discount' is generally used synonymously with 'to find the present value'. In the previous example, one can calculate the present values of each of the individual cash flows. The net present value assuming a minimum acceptable discount rate of 15% is

$$
\begin{array}{llcr}
\text{Year 0} & \text{NPV}_0 = -200 & = & -200.00 \\[6pt]
\text{Year 1} & \text{NPV}_1 = \dfrac{-100}{1.15} & = & -86.96 \\[6pt]
\text{Year 2} & \text{NPV}_2 = \dfrac{100}{(1.15)^2} & = & 75.61 \\[6pt]
\text{Year 3} & \text{NPV}_3 = \dfrac{110}{(1.15)^3} & = & 73.33 \\[6pt]
\text{Year 4} & \text{NPV}_4 = \dfrac{120}{(1.15)^4} & = & 68.61 \\[6pt]
\text{Year 5} & \text{NPV}_5 = \dfrac{130}{(1.15)^5} & = & 64.63 \\[6pt]
\text{Year 6} & \text{NPV}_6 = \dfrac{140}{(1.15)^6} & = & 60.53 \\
\hline
 & \text{Discounted cash flow} = & & \$55.75
\end{array}
$$

The summed cash flows equal $55.75. This represents the additional capital expense that could be incurred in year 0 and still achieve a minimum rate of return of 15% on the invested capital.

2.2.8 *Discounted cash flow rate of return (DCFROR)*

To calculate the net present value, a discount rate had to be assumed. One can however calculate the discount rate which makes the net present value equal to zero. This is called the discounted cash flow rate of return (DCFROR) or the internal rate of return (ROR). The terms DCFROR or simply ROR will be used interchangeably in this book. For the example given in Subsection 2.2.6, the NPV equation is

$$\text{NPV} = -200 - \frac{100}{1+i} + \frac{100}{(1+i)^2} + \frac{110}{(1+i)^3} + \frac{120}{(1+i)^4} + \frac{130}{(1+i)^5} + \frac{140}{(1+i)^6} = 0$$

Solving for i one finds that

$$i \cong 0.208$$

In words, the after tax rate of return on this investment is 20.8%.

2.2.9 *Cash flows, DCF and DCFROR including depreciation*

The cash flow calculation is modified in the following way when a capital investment is depreciated over a certain time period.

  Gross revenue
− Operating expense
− Depreciation
= Taxable income
− Tax
= Profit
+ Depreciation
− Capital costs

= Cash flow

In this book no attempt will be made to discuss the various techniques for depreciating a capital asset. For this example it will be assumed that the investment (Inv) has a Y year life with zero salvage value. Standard straight line depreciation yields a yearly depreciation value (Dep) of

$$\text{Dep} = \frac{\text{Inv}}{Y} \tag{2.6}$$

The procedure will be illustrated using the example adapted from Stermole & Stermole (1987).

Example. A \$100 investment cost has been incurred at time $t = 0$ as part of a project having a 5 year lifetime. The salvage value is zero. Project dollar income is estimated to be \$80 in year 1, \$84 in year 2, \$88 in year 3, \$92 in year 4, and \$96 in year 5. Operating expenses are estimated to be \$30 in year 1, \$32 in year 2, \$34 in year 3, \$36 in year 4, and \$38 in year 5. The effective income tax rate is 32%.

The cash flows are shown in Table 2.2.

The net present value (NPV) of these cash flows assuming a discount rate of 15% is \$43.29

$$\begin{aligned} \text{NPV} &= -100 - \frac{40.4}{1.15} + \frac{41.8}{(1.15)^2} + \frac{43.1}{(1.15)^3} + \frac{44.5}{(1.15)^4} + \frac{45.8}{(1.15)^5} \\ &= \$43.29 \end{aligned}$$

Table 2.2. Cash flow example including depreciation (Stermole & Stermole, 1987).

Year	0	1	2	3	4	5	Cumulative
Revenue		80.0	84.0	88.0	92.0	96.0	440.0
− Oper costs		−30.0	−32.0	−34.0	−36.0	−38.0	−170.0
− Depreciation		−20.0	−20.0	−20.0	−20.0	−20.0	−100.0
= Taxable		30.0	32.0	34.0	36.0	38.0	170.0
− Tax @ 32%		−9.6	−10.2	−10.9	−11.5	−12.2	−54.4
= Net income		20.4	21.8	23.1	24.5	25.8	115.6
+ Depreciation		20.0	20.0	20.0	20.0	20.0	100.0
− Capital costs	−100.0	–	–	–	–	–	−100.0
Cash flow	−100.0	40.4	41.8	43.1	44.5	45.8	115.6

The DCFROR is the discount rate which makes the net present value equal to zero. In this case

$$\text{NPV} = -100 - \frac{40.4}{1+i} + \frac{41.8}{(1+i)^2} + \frac{43.1}{(1+i)^3} + \frac{44.5}{(1+i)^4} + \frac{45.8}{(1+i)^5} = 0$$

The value of i is about

$$i \cong 0.315$$

2.2.10 *Depletion*

In the U.S. special tax consideration is given to the owner of a mineral deposit which is extracted (depleted) over the production life. One might consider the value of the deposit to 'depreciate' much the same way as any other capital investment. Instead of 'depreciation', the process is called 'depletion'. The two methods for computing depletion are:

(1) cost depletion,

(2) percentage depletion.

Each year both methods are applied and that which yields the greatest tax deduction is chosen. The method chosen can vary from year to year. For most mining operations, percentage depletion normally results in the greatest deduction.

To apply the cost depletion method, one must first establish the cost depletion basis. The initial cost basis would normally include:

- the cost of acquiring the property including abstract and attorney fees.
- exploration costs, geological and geophysical survey costs.

To illustrate the principle, assume that this is \$10. Assume also that there are 100 tons of reserves and the yearly production is 10 tons. The \$10 cost must then be written off over the 100 total tons. For the calculation of cost depletion the cost basis at the end of any year (not adjusted by the current years depletion) is divided by the estimated remaining ore reserve units plus the amount of ore removed during the year. This gives the unit depletion. In this simple case, for year 1

$$\text{Unit depletion} = \frac{\$10}{100} = \$0.10$$

The unit depletion is then multiplied by the amount of ore extracted during the year to arrive at the depletion deduction,

$$\text{Depletion deduction} = 10 \text{ tons} \times \$0.10 = \$1$$

The new depletion cost basis is the original cost basis minus the depletion to date. Thus for the year 2 calculation:

$$\begin{aligned}\text{Depletion cost basis} &= \$10 - \$1 = \$9\\ \text{Remaining reserves} &= 90 \text{ tons}\end{aligned}$$

The year 2 unit depletion and depletion deduction are:

$$\text{Unit depletion} = \frac{\$9}{90} = \$0.10$$

$$\text{Depletion deduction} = 10 \times \$0.10 = \$1$$

Table 2.3. Percentage depletion rates for the more common minerals. A complete list of minerals and their percentage depletion rates are given in section 613(b) of the Internal Revenue Code.

Deposits	Rate
Sulphur, uranium, and, if from deposits in the United States, asbestos, lead ore, zinc ore, nickel ore, and mica	22 %
Gold, silver, copper, iron ore, and certain oil shale, if from deposits in the United States	15 %
Borax, granite, limestone, marble, mollusk shells, potash, slate, soapstone, and carbon dioxide produced from a well	14 %
Coal, lignite, and sodium chloride	10 %
Clay and shale used or sold for use in making sewer pipe or bricks or used or sold for use as sintered or burned lightweight aggregates	7½ %
Clay used or sold for use in making drainage and roofing tile, flower pots, and kindred products, and gravel, sand, and stone (other than stone used or sold for use by a mine owner or operator as dimension or ornamental stone)	5 %

IRS 2011. Depletion.
Ref: www.irs.gov/publications/p535/ch09.html

Once the initial cost of the property has been recovered, the cost depletion basis is zero. Obviously, the cost depletion deduction will remain zero for all succeeding years.

The percent depletion deduction calculation is a three step process. In the first step, the percent deduction is found by multiplying a specified percentage times the gross mining income (after royalties have been subtracted) resulting from the sale of the minerals extracted from the property during the tax year. According to Stermole & Stermole (1987):

'Mining' includes, in addition to the extraction of minerals from the ground, treatment processes considered as mining applied by the mine owner or operator to the minerals or the ore, and transportation that is not over 50 miles from the point of extraction to the plant or mill in which allowable treatment processes are applied. Treatment processes considered as mining depend upon the ore or mineral mined, and generally include those processes necessary to bring the mineral or ore to the stage at which it first becomes commercially marketable; this usually means to a shipping grade and form. However, in certain cases, additional processes are specified in the Internal Revenue Service regulations, and are considered as mining. Net smelter return or its equivalent is the gross income on which mining percentage depletion commonly is based. Royalty owners get percentage depletion on royalty income so companies get percentage depletion on gross income after royalties.

As shown in Table 2.3, the percentage which is applied varies depending on the type of mineral being mined.

In step 2, the taxable income (including all deductions except depletion and carry forward loss) is calculated for the year in question. Finally in step 3, the allowable percentage depletion deduction is selected as the lesser of the percent depletion (found in step 1) and 50% of the taxable income (found in step 2).

With both the allowable cost depletion and percentage depletion deductions now calculated, they are compared. The larger of the two is the 'allowed depletion deduction'. The overall process is summarized in Figure 2.1.

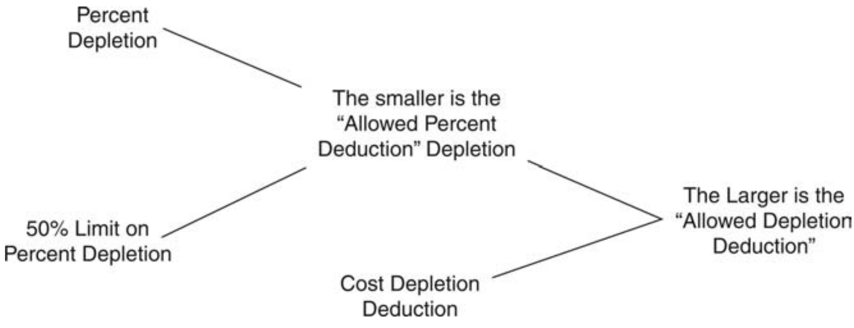


Figure 2.1. Flow sheet for determining the depletion deduction (Stermole & Stermole, 1987).

2.2.11 *Cash flows, including depletion*

As indicated the depletion allowance works exactly the same way in a cash flow calculation as depreciation. With depletion the cash flow becomes:

Gross revenue
− Operating expense
− Depreciation
− Depletion
− Taxable income
− Tax
= Profit
+ Depreciation
+ Depletion
− Capital costs

= Cash flow

The following simplified example adapted from Stermole & Stermole (1987) illustrates the inclusion of depletion in a cash flow calculation.

Example. A mining operation has an annual sales revenue of $1,500,000 from a silver ore. Operating costs are $700,000, the allowable depreciation is $100,000 and the applicable tax rate is 32%. The cost depletion basis is zero. The cash flow is:

(*1*) *Preliminary*. Calculation without depletion.

Gross revenue	$1,500,000
− Operating expense	−700,000
− Depreciation	−100,000
= Taxable income before depletion	$700,000

(*2*) *Depletion calculation*. Since the depletion basis is zero, percentage depletion is the only one to be considered. One must then choose the smaller of:
(a) 50% of the taxable income before depletion and carry-forward-losses
(b) 15% of the gross revenue

In this case the values are:

(a) $0.50 \times \$700{,}000 = \$350{,}000$

(b) $0.15 \times \$1{,}500{,}000 = \$225{,}000$

Hence the depletion allowance is $225,000.

(*3*) *Cash flow calculation.*

Gross revenue	$1,500,000
− Operating expense	−700,000
− Depreciation	−100,000
− Depletion	−225,000
= Taxable income	$475,000
− Tax @ 32%	−152,000
= Profit	$323,000
+ Depreciation	+100,000
+ Depletion	+225,000
= Cash flow	$648,000

The cash flow calculation process is expressed in words (Laing, 1976) in Table 2.4.

Laing (1976) has summarized (see Table 2.5) the factors which should be considered when making a cash flow analysis of a mining property.

The distinction between the 'exploration' and 'development' phases of a project is often blurred in actual practice. However from a tax viewpoint a sharp distinction is often made. The distinction made by the U.S. Internal Revenue Service (1988a) is paraphrased below.

1. The exploration stage involves those activities aimed at ascertaining the existence, location, extent or quality of any deposit of ore or other mineral (other than oil or gas). Exploration expenditures paid for or incurred before the beginning of the development stage of the mine or other natural deposit may for tax purposes be deducted from current income. If, however, a producing mine results, these expenditures must be 'recaptured' and capitalized. These are later recovered through either depreciation or cost depletion.

2. The development stage of the mine or other natural deposit will be deemed to begin at the time when, in consideration of all the facts and circumstances, deposits of ore or other mineral are shown to exist in sufficient quantity and quality to reasonably justify exploitation. Expenditures on a mine after the development stage has been reached are treated as operating expenses.

For further information concerning the economic models the interested reader is referred to Stermole & Stermole (2012).

2.3 ESTIMATING REVENUES

2.3.1 *Current mineral prices*

Current mineral prices may be found in a number of different publications. *Metals Week, Mining Magazine, Metal Bulletin* and *Industrial Minerals* are four examples. Spot prices

Table 2.4. Components of an annual cash flow analysis for a mining property (Laing, 1976).

Calculation	Component
	Revenue
less	Operating costs
equals	Net income before depreciation and depletion
less	Depreciation and amortization allowance
equals	Net income after depreciation and amortization
less	Depletion allowance
equals	Net taxable income
less	State income tax
equals	Net federal taxable income
less	Federal income tax
equals	Net profit after taxes
add	Depreciation and amortization allowances
add	Depletion allowance
equals	Operating cash flow
less	Captial expenditures
less	Working capital
equals	Net annual cash flow

Table 2.5. Factors for consideration in a cash flow analysis of a mining property (Laing, 1976).

Preproduction period	
Exploration expenses	Land and mineral rights
Water rights	Environmental costs
Mine and plant capital requirements	Development costs
Sunk costs	Financial structure
Working capital	Administration
Production period	
Price	Capital investment-replacement and expansions
Processing costs	Royalty
Recovery	Mining cost
Post concentrate cost	Development cost
Reserves and percent removable	Exploration cost
Grade	General and administration
Investment tax credit	Insurance
State taxes	Production rate in tons per year
Federal taxes	Financial year production begins
Depletion rate	Percent production not sent to processing plant
Depreciation schedule	Operating days per year
Post production period	
Salvage value	Contractual and reclamation expenditures

for the major metals are listed in the business sections of many daily newspapers together with futures prices. Tables 2.6 through 2.8 contain 1992–93 prices for certain:
– metals
– nonmetallic minerals
– miscellaneous metals
– ferro alloys
– ores and concentrates.

In reviewing the tables it is seen that there is considerable variation in how the prices are quoted. In general, the prices depend upon:
– quality
– quantity
– source
– form
– packaging.

The units in which the prices are expressed also vary. Some examples in this regard are presented below.

1. For many minerals, the 'ton' is unit of sale. There are three different 'tons' which might be used. They are:

$$\begin{aligned}
\text{1 short ton (st)} &= \text{2000 lbs} = \text{0.9072 metric tons} \\
\text{1 long ton (lt)} &= \text{2240 lbs} = \text{1.01605 metric tons} \\
\text{1 metric ton (mt or tonne)} &= \text{2204.61 lbs} \\
&= \text{1000 kilograms} \\
&= \text{0.9842 long tons} \\
&= \text{1.1023 short tons}
\end{aligned}$$

Iron ore, sulfur, and manganese ore are three materials normally sold by the long ton. The prices for iron ore and manganese ore are expressed in X dollars (or cents) per long ton unit (ltu). A 'unit' refers to the unit in which the quality of the mineral is expressed. For iron ore the quality is expressed in $Y\%$ Fe. Therefore one unit means 1%.

If 1 long ton (2240 lbs) of iron ore contained 1% iron (22.40 lbs), then it would contain 1 long ton unit (1 ltu) of iron. If the long ton assayed at 65% iron then it would contain 65 ltu. If the quoted price for pellets is 70 ¢/ltu, then the price of 1 long ton of pellets running 65% iron would be:

$$\begin{aligned}
\text{price/long ton} &= 65 \times 70\,¢ = 4550\,¢/\text{lt} \\
&= \$45.50/\text{lt}
\end{aligned}$$

Metric ton units (mtu) and short ton units (stu) are dealt with in the same way.

The reason for using the 'unit' approach is to take into account varying qualities.

2. For most metals, the unit of weight is the pound (lb) or kilogram (kg).
3. Gold, silver, platinum, palladium, and rhodium are sold by the troy ounce.

Table 2.6. Metal prices (*E/MJ*, 1993).

Aluminum, ¢/lb	
Used beverage cans, 12/24/92	40–42
Comex, 99.7% closing, Mar.	58.00
May	58.00
N.Y. merchant, 99.7%, 1/11/93	57.00–57.25
Antimony, $/lb	
Merchant, 3/1/90	0.85–0.92
Antimony oxide, 3/1/90	1.10–1.20
Beryllium copper, 9/17/90	
Strip (No. 25)	9.25
Rod, bar, and wire (No. 25)	10.24
Bismuth, $/lb, ton lots	
Merchant, 3/1/92	2.40–2.45
Cadmium, $/lb, ton lots	
Producers, 6/1/92	1.80–2.00
Chromium, $/lb, 1/1/91	
Electrolytic metal, standard	3.75
Cobalt, 99%, $/lb	
Afrimet, F.O.B. New York	
Cathodes, etc., 1/1/92	13.00
Powder, 1/27/89	NQ
Extra fine, 1/27/89	NQ
Sherritt Gordon	
'S' power, 12/31/90	NQ
Copper, ¢/lb	
LME, grade A, closing	
cash bid	100.26
3 mo	100.58
Comex, high grade, Jan. closing	98.35
U.S. producers, cathode	100.00
Warrenton refining, wirebar	110.35
N.Y. merchant, cathodes, Mar.	101.00
Gold, $/tr oz	
Zurich avg., opg	328.00
Paris, p.m.	329.21
London, 3:00 p.m.	327.65
Handy & Harman, N.Y.	327.65
Engelhard bullion	328.86
Engelhard fabricated	345.30
Lead, ¢/lb	
U.S. and Canadian producers, 11/6/92	32.00–35.00
Secondary fabricated, 11/9/92	36.00–40.00
London fix, $/mt	
Rudolf Wolff, spot	434.18
Rudolf Wolff, 3 mo	445.88
Lithium, $/lb	
99.9%, 1,000 lb lots, 11/2/92	31.80–32.45
Carbonate tech., 11/2/92	1.91–1.96
Magnesium, ¢/lb, 5-st lots	
Ingots, 99.8%, 11/16/91	143–153
Grinding slab, 1/1/91	143
Sticks, 1.3-in.-dia, 1/2/91	223
Manganese, ¢/lb	
Electrolytic, 99.9%, 11/15/90	105
Mercury, 99.9%, $/flask	
New York prompt, 8/26/92	205–210
C.I.F. European port, 12/8/92	115–130
Molybdic oxide, $/lb	
Producer, 1/6/92	3.35
Nickel, $/lb	
Melting briquettes, 10/10/91	3.48–3.52
N.Y. merchant, spot, 1/14/93	2.64–2.68
Platinum, $/tr oz	
London p.m. fix 1/14/93	358.75
Comex, 99.9%, Apr.	355.50
Engelhard fabricated	459.00
U.S. merchant, 1/14/93	358.00–359.00
Silver, ¢/tr oz	
Engelhard bullion	368.00
Handy & Harman, N.Y.	397.00
London fix, spot	369.15
3 mo	371.95
6 mo	374.95
12 mo	383.00
Zurich fix	369.70
Tin	
Kuala Lumpur spot, ringgit/kilo	14.81
Spot exchange, $/ringgit	0.3843
AMM N.Y. ex-dock, $/lb	2.76
Uranium, $/lb U_3O_8	
Nuexco, 12/31/92	7.85
Zinc, ¢/lb	
U.S. and foreign producers, slab, delivered in U.S., 1/14/93	
High grade	53.76–54.74
Special high grade	51.00–55.24
CGG	51.75–54.49
U.S. producers, die casting alloys	
No. 3, 7/30/91	NQ
No. 5, 7/30/91	NQ

Table 2.7. Prices for some common non-metallic minerals (*Industrial Minerals*, December 1992). Copyright 1992. Industrial Minerals, reproduced with permission.

Asbestos		
All prices quoted are F.O.B. mine		
Canadian chrysotile		
Group No. 3	C$1,450–1,750	
Group No. 4	C$1,080–1,400	
Group No. 5	C$645–850	
Group No. 6	C$525–575	
Group No. 7	C$180–350	
South African chrysotile		
Group No. 5	$360–410	
Group No. 6	$300–390	
Group No. 7	$180–220	
South African amosite		
Long	$660–1,000	
Medium	$610–700	
Short	$425–625	
South African crocidolite		
Long	$720–880	
Medium	$645–715	
Short	$640–695	
Bentonite		
Wyoming, foundry grade, 85% 200 mesh, bagged, 10 ton lots, del U.K.	£120–130	
F.O.B. plants, Wyoming rail hopper cars, bulk st	$18.00–35.00	
F.O.B. plants, Wyoming, bagged, rail cars, st	$33.00–45.00	
Fullers' Earth, soda ash-treated, del, U.K. foundry grade, bagged	£85–95	
Civil engineering grade, bulk	£60–70	
OCMA, bulk del U.K.	£65–70	
API, F.O.B. plant, Wyoming, rail cards, bagged, st	$34.50	
Feldspar		
Ceramic grade, powder, 300 mesh, bagged, ex-storage U.K.	£140	
Sand, 28 mesh, glass grade, ex-store U.K.	£65	
Ceramic grade, bulk, st		
F.O.B. Spruce Pine, NC, 170-250 mesh	$50.00	
F.O.B. Monticello, Ga, 200 mesh high potash	$82.50	
F.O.B. Middleton, Con, 200 mesh	$67.50	
Glass grade, bulk, st		
F.O.B. Spruce Pine, NC, 97.8% > 200 mesh	$33.50	
F.O.B. Monticello, Ga, 92% > 200 mesh, high potash	$64.75	
F.O.B. Middleton, Con, 96% > 200 mesh	$45.50	
Fluorspar		
Metallurgical, min 70% CaF_2, ex-U.K. mine	£85–90	
Acidspar, dry basis 97% CaF_2 bagged ex-works	£140–150	
Acidspar, dry, bulk ex-works tankers	£125–135	
Acidspar Chinese dry bulk, C.I.F. Rotterdam	$100–110	
Mexican, F.O.B. Tampico,		
Acidspar filtercake	$122–127	
Metallurgical	$90–95	
South African acidspar dry basis, F.O.B. Durban	$110–115	
USA, Illinois district, bulk, st acidspar	$190–195	
Iodine		
Crude iodine crystal, 50 kg drums 99.5% min, per kg del U.K.	$15–16	
Phosphates		
Florida, land pebble, run of mine, st dry basis, unground, bulk, ex-mine, avg.	Domestic	Export
60–66% BPL	$34.99	$30.36
66–70% BPL	$25.99	$34.67
70–72% BPL	$27.84	$37.38
72–74% BPL	$36.24	$41.80
74% BPL	$35.10	$50.20
Morocco, 75–77% BPL, FAS Casablanca	$48.50	
70–72% BPL, FAS Casablanca	$46	
Tunisia, 65–68% BPL, FAS Sfax	$32–38	
Nauru, 83% BPL, lt, F.O.B.	–	
Potash		
Muriate of potash, bulk, 60% K_2O		
Std, C.I.F. UP port	£71–74	
Granular, C.I.F. U.K. port	£81–84	
Std, F.O.B. Vancouver	$90–100	
F.O.B. Saskatchewan, bulk per, st		
Standard	$83	
Coarse	$87	
Granular	$89	
F.O.B. Carlsbad, bulk, per ton,		
Coarse	$90–100	
Granular	$95	
Salt		
Ground rocksalt, 15–20 tonne lots, avg. price del U.K.	£20	
Soda ash		
US natural, F.O.B. Wyoming, Dense, st	$80	
Sulphur		
US Frasch, liquid, dark ex-terminal, Tampa, lt	$88	
Canadian, liquid, bright, F.O.B. Rotterdam, tonne	$90	
French, Polish, liquid, ex-terminal Rotterdam, tonne	$105.75	
Canadian, solid/slate, F.O.B. Vancouver, spot	$65.75	
Canadian, solid/slate, F.O.B. Vancouver, contract	$65.70	

To accord with trade practices, certain prices are quoted in US$ (sterling now floating at around \$1.50–1.70 = £1). All quotations are © *Metal Bulletin* plc 1992.

Table 2.8. Prices* for some common non-ferrous ores (*Metal Bulletin*, 1993). Copyright 1993 Metal Bulletin; reproduced with permission.

Antimony	Per metric tonne unit Sb. C.I.F.
Clean sulphide conc., 60% Sb	\$14.00–\$15.50
Lump sulphide ore, 60% Sb	\$14.50–\$16.00
Chinese conc., 60% Sb, Se typically 60 ppm. Hg 30 ppm max.	\$12.00–\$13.00
Beryl	Per short ton unit of BeO
Cobbed lump min. 10% BeO C.I.F.	\$75–\$80
Chromite	Per tonne delivered
Transvaal, friably lumpy, basis 40% Cr_2O_3 F.O.B.	\$55–\$65
Albanian, hard lumpy, min. 42% F.O.B.	\$70–\$80
Albanian, conc., 51% F.O.B.	\$100–\$110
Turkish, lumpy, 48% 3:1 (scale pro rata) F.O.B.	\$160–\$180
Russian, lumpy, 40% min. 36% F.O.B.	\$75–\$95
Columbium ores	Per lb pentoxide content
Columbite min. 65% $Cb_2O_5 + Ta_2O_5$, 10:1 C.I.F.	\$2.60–\$3.05
Lead conc.	
70/80% Pb \$500–550 basis C.I.F.	\$170–\$180
Lithium ores	Per tonne
Petalite, 3.5–4.5% C.I.F.	£135–£140
Spodumene 4–7% Li_2O C.I.F.	£178–£183
Manganese ore	Metallurgical per mtu Mn
48/50% Mn Max. 0.1%P C.I.F.	\$3.35–\$3.55
Molybdenite	Per lb Mo in MoS_2
Conc. C.I.F.	\$1.95–\$2.05
Conc. C.I.F. U.S.	\$2.80–\$3.00
Monazite	Australian per tonne
Conc. Min.55% REO + Thoria, F.O.B./Fid	A\$300–A\$350
Tantalite	Per lb Ta_2O_5
25/40% basis 30% Ta_2O_5 C.I.F., max 0.5% U_3O_8 and ThO_2 combined	\$30.00–\$33.00
Greenbushes 40% basis	\$40.00
Tin conc.	T/C per tonne
20/30% Sn (including deduction)	£400–£530
30/50% Sn (including deduction)	£350–£500
50/65% Sn (including deduction)	£300–£600
65/75% Sn (including deduction)	£400–£525
Titanium ores	Australian per tonne
Rutile conc.min.95% TiO_2 bagged, F.O.B./Fid	A\$550–A\$600
Rutile bulk conc.min.95% TiO_2 F.O.B./Fid	A\$500–A\$560
Limenite bulk conc.min.54% TiO_2 F.O.B.	A\$83–A\$90
Tungsten ore	Per metric tonne unit WO_3
Min. 65% WO_3 C.I.F.	\$40–\$50
Uranium	Per lb U_3O_8
Nuexco exchange value December	\$7.85
Nuexco restricted American market penalty	\$2.10
Nukem December restricted spot	\$9.90–\$10.35
Nukem December unrestricted spot	\$7.90–\$8.00
Vanadium	Per lb V_2O_5
Highveld, fused min. 98% V_2O_5 C.I.F.	\$1.95
Other sources	\$1.75–\$1.85
Zinc conc.	T/C per metric dry tonne
Sulphide 49/55% Zn basis \$1,000 C.I.F. main port	\$188–\$190
Sulphide 56/61% Zn basis \$1,000 C.I.F. main port	\$190–\$194
Zircon	Australian per tonne
Std. min. 65% ZrO_2 F.O.B./Fid	A\$230–A\$270
Premium max. 0.05% Fe_2O_3 F.O.B./Fid	A\$250–A\$325

*Prices expressed C.I.F. Europe unless otherwise indicated.

The relationship between the troy ounce and some other units of weight are given below.

Troy weight (tr)

$$
\begin{aligned}
1 \text{ troz} &= 31.1035 \text{ grams}\\
&= 480 \text{ grains}\\
&= 20 \text{ pennyweights(dwt)}\\
&= 1.09714 \text{ oz avoird}
\end{aligned}
$$

U.S. Standard (avoirdupois)

$$
\begin{aligned}
1 \text{ ozavoird} &= 28.3495 \text{ grams}\\
&= 437.5 \text{ grains}\\
1 \text{ lb} &= 16 \text{ oz}\\
&= 14.5833 \text{ tr oz}\\
&= 453.59 \text{ grams}
\end{aligned}
$$

4. Mercury (quicksilver) is sold by the flask. A 'flask' is an iron container which holds 76 lbs of mercury.
5. Molybdenum is sometimes quoted in the oxide (roasted) form (MoO_3) or as the sulfide (MoS_2). The price given is per pound of Mo contained. For the oxide form this is about 67% and for the sulfide 60%.
6. The forms in which minerals are sold include bulk, bags, cathodes, ingots, rods, slabs, etc.
7. The purity of the mineral products often have a substantial effect on price (see for example Feldspar).
8. The fiber length and quality is extremely important to asbestos.

The point of sale also has a considerable effect on the price. Two abbreviations are often used in this regard.

The abbreviation 'F.O.B.' stands for 'free-on-board'. Thus the designation 'F.O.B. mine' means that the product would be loaded into a transport vessel (for example, rail cars) but the buyer must pay all transport charges from the mine to the final destination.

The abbreviation 'C.I.F.' means that cost, insurance and freight are included in the price.

Table 2.9 illustrates the difference in price depending on delivery point. For USX Corp. pellets, the price of 37.344 ¢/ltu is at the mine (Mountain Iron, Minnesota). The Cleveland-Cliffs price of 59.4 ¢/ltu (also for Minnesota taconite pellets) is at the hold of the ship at the upper lake port. Hence a rail charge has now been imposed. For the Oglebay Norton Co. (Minnesota) pellets, the price of 72.45 ¢/ltu is at the lower lake port. Thus it includes rail transport from the mine to the upper lake port plus ship transport to the lower lake port. Table 2.10 provides freight rates for iron ore and pellets. Lake freight rates are given in Table 2.11.

Many mineral products are sold through long term contracts arranged between supplier and customer. The prices will reflect this shared risk taking. There will often be significant differences between the short term (spot) and long term prices.

Recent prices for metals and industrial minerals are provided in Tables 2.12 and 2.13. Table 2.14 presents the prices for iron ore from various suppliers in 2005.

Table 2.9. Iron ore prices (*Skillings Mining Review*, 1993a).

Lake Superior iron ore prices	
(Per gross ton, 51.50% iron natural, at rail of vessel lower lake port)	
Mesabi non-Bessemer	$30.03-31.53
Cleveland-Cliffs Inc iron ore pellet prices	
(Per iron natural unit, at rail of vessel lower lake port)	
Cleveland-Cliffs Inc	72.45¢
Per gross ton unit at hold of vessel upper lake port	53.40¢
Wabush pellets per gross ton unit F.O.B. Pointe Noire	63.50¢
Cyprus Northshore Mining Corp. pellet price	
(Per gross ton iron unit natural F.O.B. Silver Bay)	
Cyprus Northshore pellets	48.76¢
IOC Ore Sales Co. pellet price	
(Per natural gross ton unit delivered rail of vessel lower lake port)	
Carol pellets	74.65¢
Inland Steel Mining Co. pellet price	
Per gross ton natural iron unit at hold vessel upper lake port	46.84¢
Oglebay Norton Co. pellet prices	
(Per natural gross ton unit, at rail of vessel lower lake port)	
Standard grade	72.45¢
Eveleth special	74.00¢
U.S. Steel pellet price	
Per dry gross ton iron unit at Mtn. Iron	37.344¢

Table 2.10. Rail tariff rates on iron ore and pellets (*Skillings Mining Review,* 1993b).

Rail freight rates ($/gross ton) from mines to Upper Lake Port	
Marquette Range to Presque Isle	
Pellets	$2.50
Natural ore	2.56
Pellets from Marquette Range to Escanaba delivered into vessel	3.39
Mesabi Range plants on BN to:	
Allouez delivered direct into vessel	6.16
When consigned to storage subject to storage charges	6.53
Winter ground storage charges on pellets:	
At Allouez per gross ton per month	16.0¢
At Escanaba: storage per gross ton per month	2.4¢
handling to storage	21.8¢
handling from storage	21.8¢
Dock charges on iron ore per gross ton	
Car to vessel at Duluth and Two Harbors	$1.05
All-rail freight rates to consuming districts	
Mesabi Range to:	
Chicago district	20.42
Geneva, Utah	*46.97
Granite City and East St. Louis, Ill.	19.69
Valley district	40.17
Marquette Range to Detroit	28.60
Marquette & Menominee Ranges to:	
Chicago district	**24.09
Granite City and East St. Louis, Ill.	23.18

*Conditional on tender of not less than 4800 GT nor more than 5200 GT.
**Multiple car rate.

Table 2.11. Lake freight tariff rates on iron ore, pellets and limestone (*Skillings Mining Review*, 1993c).

Lake freight rates from Upper Lake ports to Lower Lake ports	
Iron ore ($/gross ton)	Self unloading vessels
Head of Lakes to Lower Lakes	$6.50
Marquette to Lower Lakes	5.40
Escanaba to Lake Erie	4.88
Escanaba to Lake Michigan	3.90
Limestone ($/gross ton)	
Calcite, Drummond, Cedarville and Stoneport to	
Lower Lake Michigan	3.98
Lake Erie ports	4.10
Note: The above cargo rates apply after April 15 and before December 15, 1993. Winter formulas apply during other periods. Rates are further subject to surcharges, if warranted.	
Dock, handling and storage charges ($/gross ton) on iron ore at Lower Lake ports RCCR X-088C	
Ex self-unloading vessels at Cleveland, Ohio	
Dockage	$0.26
From dock receiving area into cars, via storage	1.60
From dock receiving area to cars	1.05
Rail of vessel receiving area to cars	1.16
At Conneaut, Ohio BLE	
Dockage of self-unloading vessel	$0.15
From receiving bin to storage	0.53
From storage to railcars	0.68
Ex bulk vessels at Cleveland C&P	
From hold to rail of vessel	$1.03
From rail of vessel into car	1.26
From rail of vessel via storage into car	2.25

2.3.2 *Historical price data*

Mineral prices as monitored over a time span of many years exhibit a general upward trend. However, this is not a steady increase with time but rather is characterized by cyclic fluctuations. Table 2.15 shows the average yearly prices for 10 common metals from 1900 through 2011 (USGS, 2012a). The monthly and average prices for 11 common metals for the period January 1997 through July 2012 are given in Table 2.16 (Metal Bulletin, 2012). To provide an indication of the price unpredictability, consider the case of copper. In 1900 for example the copper price was about 16.2 ¢/lb (Table 2.15). In 2000, 100 years later, the price had risen to 82 ¢/lb. The average rate of price increase per year over this period using the end point values is 1.6 percent. The price dropped to a low of 5.8 ¢/lb (1932) and reached a high of 131 ¢/lb (1989) over this period. Using the average price increase over the period of 1900 to 2000, the predicted price in 1950 should have been 36.4 ¢/lb. The actual value was 21.2 ¢/lb.

A mining venture may span a few years or several decades. In some cases mines have produced over several centuries. Normally a considerable capital investment is required to bring a mine into production. This investment is recovered from the revenues generated over the life of the mine. The revenues obviously are strongly dependent upon mineral price. If the actual price over the mine life period is less than that projected, serious revenue shortfalls would be experienced. Capital recovery would be jeopardized to say nothing of profits.

Price trends, for metals in particular, are typically cyclic. Using the metal price data from Table 2.15, Figures 2.2 through 2.6 show the prices for 10 metals over the period 1950 through 2011. The period and amplitude of the cycles varies considerably. For nickel

Table 2.12. Recent weekly metal prices (Platts Metals Week, 2012, April 16). Copyright 2012 Platts Metals Week; reproduced with permission.

Major Metals	
Aluminum	
	cts/lb
MW US Market	101.750/102.750
US Six-Months P1020	9.500
US 6063 Billet Upcharge	10.500/12.500
US UBCs	78.500/80.000
Painted Siding	79.000/80.000
US 6063 press scrap	4.500/5.500
	Eur/mt
Alloy 226 delivered European works	1700.000/ 1750.000
	$/mt
ADC12 FOB China	2310.000/2320.000
	Yuan/mt
ADC12 ex-works China	17400.000/17600.000
Caustic Soda	
	$/mt
FOB NE Asia	422.000/424.000
CFR SE Asia	476.000/478.000
Copper	
	cts/lb
MW No.1 Burnt Scrap Disc	19.000
MW No.1 Bare Bright Disc	3.000
MW No.2 Scrap Disc	39.000
NY Dealer Premium cathodes	5.500/6.500
US Producer cathodes	371.650/379.090
	$/mt
Grade A Cathode CIF R'dam	8253.000/8260.000
Grade A Premium CIF R'dam	75.000/80.000
Grade A CIF Livorno/Salerno	8228.000/8240.000
Grade A Prem CIF Livorno/Salerno	50.000/60.000
Russian Standard CIF R'dam	8178.000/8210.000
Russian Standard Prem CIF R'dam	0.000/30.000
Lead	
	cts/lb
North American Market	98.830/101.643
	$/mt
European dealer	2072.000/2088.000
European 99.985% Prem IW (R'dam)	20.000/35.000
In-Warehouse S'pore Prem	50.000/80.000
Nickel	
	$/lb
NY Dealer/Cathode	8.403/8.555
NY Dealer/Melting	8.403/8.555
NY Dealer/Plating	8.603/8.755
	cts/lb
NY Dealer/Cathode Premium	30.000
NY Dealer/Melting Premium	30.000
NY Dealer/Plating Premium	50.000
	$/mt
Plating Grade IW R'dam	18324.000/18384.000
Plating Grade Prem IW R'dam	200.000/250.000
Russia Full-Plate	18184.000/18214.000
Russia Full-Plate Prem IW R'dam	60.000/80.000
Briquette Premium IW R'dam	200.000/250.000
In-Warehouse S'pore Prem	150.000/160.000
Tin	
	$/mt
Europe 99.85% IW R'dam	22956.000/23094.000
Europe 99.85% Prem IW R'dam	300.000/400.000
Europe 99.90% IW R'dam	23156.000/23244.000
Europe 99.90% Prem IW R'dam	500.000/550.000
Zinc	
	cts/lb
US Dealer SHG	98.921
MW SHG Premium	7.250
MW SHG Galv. Prem.	7.000
MW SHG Alloyer #3 Prem.	17.000
	$/mt
Europe physical SHG IW R'dam	2109.000/2129.000
Europe physical SHG Prem IW R'dam	105.000/125.000
In-Warehouse S'pore Prem	70.000/95.000
Precious Metals	
	All PGM figures in $/tr oz
Iridium	
MW NY Dealer	1050.000/1085.000
Osmium	
MW NY Dealer	350.000/400.000
Palladium	
MW NY Dealer	630.000/655.000

(Continued)

Table 2.12. (Continued).

Platinum	
MW NY Dealer	1580.000/1630.000
Rhodium	
MW NY Dealer	1300.000/1375.000
Ruthenium	
MW NY Dealer	95.000/115.000
Minor Metals	
Antimony	
	cts/lb
MW NY Dealer	590.000/620.000
	$/mt
99.65% FOB China	13000.000/13200.000
Arsenic	
	cts/lb
MW Dealer	60.000/70.000
Bismuth	
	$/lb
MW NY Dealer	10.400/11.250
Cadmium	
	$/lb
MW NY Dealer	0.950/1.150
Free Market HG	1.000/1.200
Indium	
	$/kg
Producer: US Prod Indium Corp	785.000
MW NY Dealer	540.000/570.000
99.99% CIF Japan	560.000/570.000
Mercury	
	$/fl
Free Market International	1750.000/1950.000
U.S. Domestic	1750.000/1950.000
Rhenium	
	$/kg
MW NY Dealer	4200.000/4800.000
Selenium	
	$/lb
MW NY Dealer	61.000/66.000
Light Metals	
Magnesium	
	cts/lb
US Die Cast Alloy: Transaction	200.000/220.000
MW US Spot Western	215.000/230.000
MW US Dealer Import	200.000/210.000
	$/mt
Europe Free Market	3100.000/3200.000
Die Cast Alloy FOB China	3280.000/3340.000
99.8% FOB China	3000.000/3050.000
Silicon	
	cts/lb
553 Grade Delivered US Midwest	127.000/130.000
	$/mt
553 Grade, FOB China	2250.000/2300.000
553 Grade, CIF Japan	2280.000/2340.000
	Eur/mt
553 Grade, In-warehouse EU	1950.000/2050.000
Titanium	
	$/lb
MW US 70% Ferrotitanium	3.600/3.650
	$/kg
Eur. 70% Ferrotitanium	8.000/8.200
	$/lb
MW US Turning 0.5%	2.150/2.300
Eur. Turning .5%	2.300/2.400
Ferroalloys	
Cobalt	
MW 99.8% US Spot Cathode	*$/lb*
99.8% European	15.000/15.750
99.3% Russian	15.000/15.600
99.6% Zambian	14.800/15.200
	14.700/15.000
Ferrochrome	
	cts/lb
Charge Chrome 48–52% in-warehouse US	118.000/122.000
65% High Carbon in-warehouse US	118.000/122.000
Low Carbon 0.05% in-warehouse US	235.000/240.000
Low Carbon 0.10% in-warehouse US	218.000/222.000
Low Carbon 0.15% in-warehouse US	205.000/210.000
Charge Chrome 52% DDP NWE	103.000/110.000
65%–68% High-Carbon DDP NWE	120.000/123.000
Low Carbon 0.10% DDP NWE	215.000/220.000
High Carbon 60% FOB China	100.000/104.000
50–55% Regular CIF Japan	123.000
60–65% Spot CIF Japan	109.000/111.000
Ferromanganese	
	$/gt
High Carbon 76% in-warehouse US	1275.000/1350.000

(Continued)

Table 2.12. (Continued).

	cts/lb
Medium Carbon 85% Mn in-warehouse US	95.000/96.000
	$/mt
High Carbon 75% FOB China	1200.000/1210.000
Ferromolybdenum	
	$/lb
MW US FeMo	16.100/16.400
	$/kg
MW Europe FeMo	34.200/34.600
FOB China FeMo	34.400/34.500
Spot CIF Japan	34.900/35.000
Ferrosilicon	
	cts/lb
75% Si in-warehouse US	94.000/96.000
	$/mt
Chinese CIF Japan	1420.000/1435.000
	$/mt
75% FOB China	1420.000/1430.000
	Eur/mt
75% Std DDP NWE	1230.000/1290.000
Ferrovanadium	
	$/lb
Free Market V205	5.500/6.000
US Ferrovanadium	14.250/14.750
	$/kg
Europe Ferrovanadium	25.700/26.000
Manganese	
	$/mt
Electrolytic 99.7% FOB China	3000.000/3050.000
Molybdenum	
	$/lb
MW Dealer Oxide	14.150/14.300
Oxide Trans	14.150/14.300
Silicomanganese	
	cts/lb
65% Mn in-warehouse US	73.000/75.000
	$/mt
65% FOB China	1480.000/1500.000
Chinese CIF Japan	1400.000/1500.000
	Eur/mt
65:16 DDP NWE	1020.000/1050.000
Stainless Scrap	
	$/gt
NA FREE MKT 18-8	1792.000/1837.000
Tantalum	
	$/lb
Spot Tantalite Ore	37.000/42.000
Tungsten	
	$/stu
MW US Spot Ore	320.000/330.000
APT-US	395.000/420.000
	$/mtu
APT FOB China	400.000/410.000
	$/kg
MW Ferrotungsten	54.000/55.000
Ferrotungsten FOB	53.000/56.500

consider the period from 1983 through 1998. In 1983, the price began at \$2.12/lb, dropped to \$1.76/lb in 1986, and rose to a high of \$6.25/lb in 1988. It then dropped to \$2.40/lb in 1993 before increasing once again to \$3.73/lb in 1995 before landing in 1998 at about the same price it started at in 1983. Starting in 2002, the nickel price again began rising dramatically, reaching \$16.87/lb in 2007, falling to \$6.62/lb in 2009, and then stabilizing somewhat to around 10.00\$/lb for the years 2010 and 2011.

In 1975, silver was around \$4.50/tr oz. It shot up to nearly \$40.00/tr oz (January 1980) due to the buying of the Hunt brothers from Texas. By the end of 1991, the price had dropped back to about \$4.50/tr oz. Recently the price of silver has again been rising dramatically, reaching \$34.50/tr oz in 2011.

The price performance of iron ore fines over the period 1900–2011 is presented in Table 2.17 (USGS, 2012b). The monthly spot price for China import iron ore fines over the period January 2000 through July 2012 is presented in Table 2.18 (IMF, 2012) and in Figure 2.7. In the past, the iron ore price each year was set in January and was constant throughout the year. Starting in November 2008, the price began varying monthly reflecting that major iron ore production began moving from long term annual contracts to quarterly, monthly, and then pure spot market prices.

Table 2.13. Prices for some common industrial minerals (Industrial Minerals, 2012. Monthly Prices -June) (All prices are in US$ and quoted per tonne unless indicated.).

Alumina	
Calcined 98.5–99.5% Al_2O_3 bulk FOB US refinery	$675–725
Calcined, ground 98.5–99.5% Al_2O_3, bulk FOB US refinery	$750–850
Calcined, medium-soda Al_2O_3, bulk FOB refinery	$800–850
Alumina, fused	
Brown 95% min. Al_2O_3, FEPA F8-220 Grit, FOB China	$800–840
Brown 95.5% Al_2O_3, refractory lump & sized, FOB China	$660–730
White, 25 kg bags, CIF Europe	€850–890
Brown 94% Al_2O_3, FEPA 8-220 mesh refractory (mm) Chinese	$750–850
Andalusite	
57–58% Al_2O_3, 2,000 tonne bulk, FCA Mine-RSA	€235–280
55–59% Al_2O_3, FOB European port	€350–425
Antimony Trioxide	
Typically 99.5% SbO_3, 5 tonne lots, CIF Antwerp/Rotterdam	$11,100–11,200
Typically 99.5% SbO_3, 20 tonne lots FOB China	$11,100–11,200
Typically 99.5% SbO_3, ex-works USA	$11,100–11,200
Ingot, 99.65% min, CIF Rotterdam	$13,500–13,700
Ingot, 99.65% min, FOB China	$13,500–13,600
Baddeleyite	
Contract price, Refractory/abrasive grade, CIF main European port	$2500–3100
Contract price, Ceramic grade (98% $ZrO_2 + HfO_2$), CIF main European port	$3000–3300
Contract price, Ceramic pigment grade, CIF main European port	$3200–3500
Barytes	
Drilling grade	
Unground lump	
OCMA/API bulk, SG 4.20	
FOB Chennai	$146–150
FOB Morocco	$147–150
FOB China	$155–158
C&F North Sea (Moroccan)	$160–167
API, CIF Gulf Coast,	
Chinese	$166–182
Indian	$157–171
Ground	
OCMA bulk, del. Aberdeen	£95–105
OCMA bulk, del. Gt Yarmouth	£110–120
OCMA/API, big bags (1.5t), FOB S. Turkey	$150–155
SG 4.22, bagged, FOB Morocco	$135–147
Paint grade	
Ground white 96–98% $BaSO_4$, 350 mesh, 1–5 lots, del. UK	£195–220
Chinese lump, CIF Gulf Coast	$250–290
Ground, white, 96–98% $BaSO_4$, 325–350 mesh, 1–5 lots, ex-works USA. $/s. ton	$315–400
Chemical grade	
Chinese, CIF Gulf Coast	$170–182
Bauxite	
Refractory grade	
Chinese Al_2O_3/Fe_2O_3/BD, lumps 0–25 mm Shanxi, FOB Xingang	
Round kiln 87/2.0/3.2	$395–430
Round kiln 86/2.0/3.15–3.20	$360–440
Round kiln 85/2.0/3.15	$370–410
Rotary kiln 86/1.8/3.15	$365–435
Rotary kiln 85/1.8/3.15	$350–450
Guizhou, FOB Zhanjiang/ Fangcheng Round kiln 87/2.0/3.2	$470–525
Guyana RASC bauxite, bulk, FOB Linden	$460–510
Abrasive grade	
Chinese FOB Zhanjiang, China	$320–350
Welding grade	
FOB Zhanjiang, China	$470–480
FOB Linden, Guyana	$450–510
Bentonite	
Cat litter, grade 1–5 mm, bulk, FOB main European port	€42–60
Indian, cat litter grade, crushed, dried, loose in bulk, FOB Kandla	$34–38
OCMA/Foundry grades, crude & dried, bulk, FOB Milos	€60–80
API grade, bagged rail-car, ex-works Wyoming, per s.ton	$90–130
Foundry grade, bagged, railcars, ex-works Wyoming, per s.ton	$97–124
IOP grade, crude, bulk, ex-works Wyoming, per s.ton	$66–72
(Dried material in bulk) FOB Greece, €/tonne	€50–75
Cat litter grade, ex-works Wyoming, USA, $/s.ton	$50–60
Fullers' earth, soda ash-treated, Civil eng. grade, ex-works, South Africa	£50–70
Indian, FOB Kandla, crushed and dried, loose in bulk, Civil Engineering grade	$32–40
Indian, FOB Kandla, crushed and dried, loose in bulk, Iron ore pelletising grade	$36–38

(Continued)

Table 2.13. (Continued).

South African., ex-works Fullers' earth, soda ash-treated, Cat litter grade, 1–7 mm	£27–40
South African., ex-works Fullers' earth, soda ash-treated, foundry grade, bagged	£60–85
Foundry grade, bulk, del Japan	\$140–215
Borates/Boron minerals	
Boric Acid, FOB Chile	\$1250–1309
Colemanite, 40% B2O3, FOB Buenos Aires	\$690–730
Decathydrate Borax, FOB Buenos Aires	\$947–979
Ulexite, 40% B_2O_3, FOB Buenos Aires	\$666–697
Ulexite, 40% B_2O_3, FOB Lima	\$620–652
Ulexite, granular 40% B_2O_3, FOB Chile	\$692–734
Borax, PP bags (25 kg & 50 kg), Boric acid, gran, tech, FOB Latin America	\$1200–1310
Borax, PP bags (25 kg & 50 kg), Decahydrate borax, gran, tech, FOB Latin America	\$910–940
Boric acid, FOB Buenos Aires	\$1078–1136
Colemanite, 40–42% B_2O_3, ground, bagged, FOB Argentina	\$630–690
Ulexite 46–48% B_2O_3, FOB Lima	\$650–710
Bromine	
Purified, bulk, 99.95% Br, domestic destination, tonne lot, ex-works USA	\$1.6–1.7
Bulk, purified, 99.95% Br, ex works, CIF Europe	\$1.6–1.65
Large contract, Bulk, European	\$3300–3500
Calcium carbonate	
GCC, coated, fine grade,ex-works UK	£80–103
GCC, 50–22 microns, FOB USA	\$21–26
GCC, 22–10 microns, FOB USA	\$50–105
GCC, 3 microns (untreated), FOB USA	\$170–185
GCC, stearate coated 1.1–0.7 microns, FOB USA	\$270–400
GCC, 1.1–0.7 microns (untreated), FOB USA	\$200–290
PCC, Fine, surface treated (0.4–1 microns), FOB USA	\$275–375
GCC, coated, chalk, ex-works UK	£60–75
PCC, coated, ex-works, UK	£370–550
PCC, uncoated, ex-works, UK	£340–550
Celestite	
Turkish, 96%, $SrSO_4$, FOB Iskenderun	\$90–100
Chromite	
Chemical grade, 46% Cr_2O_3, wet bulk, FOB South Africa	\$360–410
Refractory grade, 46% Cr_2O_3, wet bulk, FOB South Africa	\$425–500
Foundry grade, 46% Cr_2O_3, wet bulk, FOB South Africa	\$390–420
South African, Northwest, Metallurgical grade, friable lumpy, 40% Cr_2O_3	\$180–210
Sand, moulding grade, 98% <30 mesh, del UK	£390–450
Foundry +47% Cr_2O_3 dried 1 tonne big bags FOB South Africa	\$450–500
Foundry, 45.8% min Cr_2O_3, wet bulk, FOB South Africa	\$390–420
Metallurgical grade, Conc' 40%, FOB Northwest, South Africa	\$180–210
Diatomite	
US, calcined filter-aid grade, FOB plant	\$575–640
US, flux-calcined filter-aid grade, FOB plant	\$580–825
Feldspar	
Turkish, Na feldspar, Crude, −10 mm size bulk, FOB Gulluk	\$22–23
Turkish, Na feldspar, Glass grade, −500 microns, bagged, FOB Gulluk	\$70
(−38 micron, FCL's bagged, >90 Brightness) FOB Durban, South Africa	\$168
(Na), ceramic grade, 170–200 mesh, bagged, ex-works USA, \$/s.ton	\$150–180
Na feldspar, floated −150 microns, bagged, FOB Gulluk, Turkey	\$53–55
Na feldspar, floated −500 microns, bulk, FOB Gulluk, Turkey	\$38–40
Fluorspar	
Acidspar filtercake, bulk	
Mexican, <5 ppm As FOB Tampico	\$540–550
Mexican, FOB Tampico	\$400–450
Chinese wet filtercake, CIF Rotterdam	\$500–530
Chinese, wet filtercake, FOB China	\$450–500
South African, dry basis, FOB Durban	\$380–450
Chinese dry basis, CIF US Gulf Port	\$550–650
Metallurgical	
Chinese, min 85% CaF_2, CIF Rotterdam	\$355–375
Mexican, FOB Tampico	\$230–270
Chinese, min 80%, wet bulk, FOB China	\$305–325
Chinese, min 85% CaF_2, bulk, FOB China	\$355–375
Chinese bulk, min. 90% CaF_2, FOB China	\$365–385

(Continued)

Table 2.13. (Continued).

Graphite	
Amorphous powder 80–85% Chinese del Europe	\$600–800
Synthetic fine 97–98% CIF Asia	\$950–1450
Synthetic fine 98–99% CIF Asia	\$1000–1500
Crystalline	
Medium flake 90%C, +100–80 mesh	\$1300–1800
Large flake, 90%C, +80 mesh	\$1900–2300
Fine, 94–97%C, −100 mesh	\$1900–2300
Medium, 94–97%C, +100–80 mesh	\$1875–2200
Large flake 94–97% C, +80 mesh CIF	\$2200–2700
Synthetic 99.95%C, \$ per kg, Swiss border	\$7–20
Ilmenite	
Australian, bulk concentrates, min 54% TiO_2, FOB	\$250–350
Australian, spot price, min 54% TiO_2, FOB	\$250–350
Iodine	
Iodine crystal, 99.5% min, drums, spot, \$/kg	\$60–90
Iodine crystal, 99.5% min, drums, contract, \$/kg	\$60–75
Iron Oxide Pigment	
Brown type 868, bagged, FOB China	\$1015–1075
Red type 130 90% Fe_2O_3, bagged, FOB China	\$1434–1637
Kaolin	
No 1 paper coating grade, Ex-Georgia plant, s.ton	\$161–209
No 2 paper coating grade, Ex-Georgia plant, s.ton	\$107.50–166.70
Kyanite	
Ex-works USA, 54–60% Al_2O_3, raw kyanite, s.ton	\$224–320
54–60% Al_2O_3, 22 ton lots, calcined	\$373–439
Leucoxene	
min. 91% TiO_2, max. 1% ZrO_2, bagged, FOB West Australia	A\$1450–1550
Lithium	
Lithium carbonate, del continental, USA large contracts, \$ per lb	\$2.5–3
Lithium hydroxide, 56.5–57.5% LiOH, large contracts, packed in drums or bags, del Europe or USA, \$/kg	\$6.5–7.5
Lithium hydroxide, Chinese, (56.5–57.5% LiOH), packed in drums or bags, large contracts, del Europe \$/kg	\$6–6.6
Petalite, 4.2% LiO_2, FOB Durban	\$165–260
Spodumene concentrate, >7.5% Li_2O, CIF USA, s.ton	\$720–770
Spodumene concentrate, 5% Li_2O, CIF USA, s.ton	\$460–510
Spodumene concentrate, 7.5% Li_2O, CIF Europe	\$750–800
Spodumene concentrate, 5% Li_2O, CIF Europe	\$440–490
Spodumene concentrate, > 7.5% Li_2O, bulk, CIF Asia	\$720–770
Spodumene concentrate, 5% Li_2O, CIF Asia	\$300–400
Magnesia	
Calcined, 90–92% MgO, lump, FOB China	\$320–360
European calcined, agricultural grade, CIF Europe	€240–350
Dead-burned	
Lump, FOB China	
90% MgO	\$350–400
92% MgO	\$430–470
94–95% MgO	\$410–480
97.5% MgO	\$560–600
Fused,	
Lump, FOB China	
96% MgO	\$790–860
97% MgO	\$930–1050
98% MgO	\$1080–1210
Magnesite	
Greek, raw, max 3.5% SiO_2, FOB East Mediterranean	€65–75
Mica	
Indian mine scrap green (Andhra Pradesh) for mica paper, FOB Madras	\$300–400
Indian wet-ground, CIF Europe	\$600–900
Micronised, FOB plant, USA	\$700–1000
Wet-ground, FOB plant, USA	\$700–1300
Flake, FOB plant, USA	\$350–500
Nitrate	
Sodium, about 98%, ex-store Chile	€550–570
Olivine	
Olivine, refractory grade, bulk, US ex-plant/mine	\$75–150
Perlite	
Coarse (filter aid) FOB east Mediterranean, bulk	€70–75
Raw, crushed, grade, big bags FOB Turkey	\$95–100
Raw, crushed, grade, bulk FOB Turkey	\$80–85
Potash	
C&F Western Europe, contract, Std.	\$400–490
Muriate, KCl, granular, bulk, ex-works, North America	\$515–535

(Continued)

Table 2.13. (Continued).

Muriate, KCl, standard, bulk, FOB Vancouver	\$460–550
Muriate, KCl, standard, bulk, FOB Baltic \$/tonne	\$350–370
Rare earth minerals	
Min 99%, large purchases, FOB China, \$/kg	\$26–32
Cerium oxide	\$1170–1370
Dysprosium oxide	\$2590–2990
Europium oxide	\$26–34
Lanthanum oxide	\$120–160
Neodymium oxide	\$120–140
Praesodymium oxide	\$88–96
Samarium oxide	
Refractory clays/Mullite	
Clay, Mulcoa 47% (sized in bulk bags), for coarse sizing, FOB USA, s.ton	\$198
Rutile	
Australian concentrate, min. 95% TiO_2, bagged, FOB	\$2500–2800
Australian concentrate, min. 95% TiO_2, large vol. for pigment, FOB	\$2050–2400
Salt	
Australian solar salt bulk CIF Shanghai,	\$50
Industrial solar salt, ex-works China	\$27–29
Industrial vacuum salt, ex-works China	\$35–40
Silica sand	
Minus 20 micron, FCL, bagged >92 brightness, FOB Durban	\$295
Glass sand, container, ex-works USA	\$20–26
Silicon carbide	
SiC, FEPA 8-220, CIF UK, black, about 99% SiC	
SiC Grade 1	€1900–2100
SiC Grade 2	€1500–1650
Refractory grade	
min 98% SiC	€1500–1800
min 95% SiC	€1350–1450
Soda ash	
Chinese synthetic soda ash, dense & light, CIF Far East	\$295–330
Chinese synthetic soda ash, dense & light, FOB China	\$260–285
Indian synthetic soda ash, dense & light, Domestic ex-works India	\$300–348
Indian synthetic soda ash, dense & light, Export C&F India	\$210–230
US natural, large contract, FOB Wyoming	\$210–230
European synthetic, dense & light, Large Contracts ex-works	€190–210
TiO_2 pigment	
Bulk volume, per tonne	
CFR Asia	\$4300–4850
CIF Northern Europe	€3260–3750
CIF USA	\$3550–4000
CIF Latin America, per lb	\$1.6–1.9
Vermiculite	
South African, bulk, FOB Antwerp	\$400–850
Wollastonite	
US ex-works, s.ton Acicular minus	
200 mesh	\$210–240
325 mesh	\$220–250
Acicular (15:-1–20:1 aspect ratio)	\$444
Chinese, FOB, tonne Acicular minus	
200 mesh	\$80–90
325 mesh	\$90–100
Zircon	
FOB Australia, bulk shipments	
Premium	\$2500–2640
Standard	\$2400–2600
FOB USA, bulk shipments	
Premium	\$2600–3000
Standard	\$2550–2750
FOB South Africa, bulk shipments	
ceramic grade	\$2300–2650
Micronised zircon	
99.5% $<4\,\mu$, average particle size $<0.95\,\mu$, C&F Asia	\$2750–2800
Fused zirconia	
Monoclinic, refractory/abrasive, contract, CIF main European port	\$6500–7800
Monoclinic, Ceramic pigment grade, Contract price, CIF main European port	\$3800–4800
Monoclinic, Structural ceramic/ electronic grade, Contract price, CIF main European port	\$4600–6000
Monoclinic, Technical ceramic, grade, Contract price, CIF main European port	\$15900–21000
Stabilised, Refractory grade, Contract price, CIF main European port	\$6500–7800
Stabilised, Technical ceramic grade, Contract price, CIF main European port	\$50000–100000
Zircon Opacifiers	
Micronised zircon, 100% <6 microns, average 1–2 microns, bagged, CFR Asia	\$2845–3400
Micronised zircon, 100% <6 microns, average 1–2 microns, bagged, ex-works Europe	\$2770–3400

The prices in Table 2.13 appeared in the June 2012 issue of Mineral Price Watch. Published by Industrial Minerals Information, a division of Metal Bulletin plc, U.K.

Table 2.14. Iron ore prices announced for year 2005 (Skillings, August 2005).

Europe (cents/mtu)	
CVRD Carajas Sinter Feed FOB Ponta da Maderia	65.00
CVRD Blast Furnace Pellets (FOB Ponta da Maderia)	118.57
CVRD Direct Reduction Pellets (FOB Ponta da Maderia)	130.43
CVRD Standard Sinter Feed (FOB Tubarao)	62.51
CVRD Blast Furnace Pellets (FOB Tubarao)	115.51
CVRD Direct Reduction Pellets (FOB Tubarao)	127.06
IOC Concentrate (FOB Sept-Iles)	66.71
IOC Pellets (FOB Sept-Iles)	120.06
Japan (cents/ltu)	
BHP Billiton Mt. Newman (DMT) Fines	61.72
BHP Billiton (DMT) Lump	78.77
CVRD Carajas Sinter Feed FOB Ponta da Maderia	57.08
CVRD Blast Furnace Pellets (FOB Ponta da Maderia)	116.86
CVRD Standard Sinter Feed (FOB Tubarao)	56.23
CVRD Blast Furnace Pellets (FOB Tubarao)	113.84
Rio Tinto Hamersley Lump	78.77
Rio Tinto Hamersley Fines	61.72
Rio Tinto Yandicoogina Ore	58.02

Note: The European and Japanese customers have negotiated somewhat different contracts.

The primary use for iron ore is in steel production, and in recent years, the iron ore market has been dominated by the phenomenal growth in steel production in China. Table 2.19 (World Steel Association, 2012) shows the annual steel production for the top twenty steel producing countries for the years 2000 through 2011, and Figure 2.8 shows the production trends for the top five steel producing countries. The annual production variation over the period for all other countries is dwarfed by the approximately sevenfold increase in annual steel production in China. Much of the recent increase in the price of molybdenum oxide (Fig. 2.4) can also be attributed to the growth in the Chinese steel production since a major use of molybdenum is in the production of steel.

The interested reader is encouraged to carefully study the price trends for each of the metals presented and try to explain the fluctuations. In some cases there are clear causes while in others an explanation is difficult to find.

For making the valuation calculations, the first problem is deciding what base price should be used. The second problem is forecasting the future price history.

Figure 2.9 shows a plot of the average price for copper as a function of time over the period of 1935 to 1992. This is based on the data given in Table 2.20. As can be seen, the price exhibits an upward trend but a cyclic variation is observed.

If the year in which the valuation was made was 1980, then the average copper price is 101.42 ¢/lb. If this current price had been selected as the base price, since it was at the peak of a cycle, the average price would never reach this base price again for many years. In this particular case until 1988. The revenue projection would have been very far off. The same would have been true if the base price for 1985 (a local low) had been selected. Here however the revenue projections would be too pessimistic and possibly the proposed project would be shelved. The conclusion is that choosing the current price as the base price for the

Table 2.15. Average annual metal prices 1900–2011 (USGS, 2012a).

Year	Al (¢/lb)	Cu (¢/lb)	Pb (¢/lb)	Zn (¢/lb)	Au ($/tr oz)	Pt ($/tr oz)	Ag ($/tr oz)	Mo ($/lb)	Ni ($/lb)	Sn ($/lb)
1900	32.7	16.2	4.5	4.4	20.67	6	0.62	NA	0.50	0.30
1901	33.0	16.1	4.4	4.1	20.67	20	0.59	NA	0.56	0.17
1902	33.0	11.6	4.1	4.8	20.67	20	0.53	NA	0.45	0.27
1903	33.0	13.2	4.2	5.4	20.67	19	0.53	NA	0.40	0.28
1904	35.0	12.8	4.3	5.1	20.67	21	0.59	NA	0.40	0.28
1905	35.0	15.6	4.5	5.9	20.67	17	0.62	NA	0.40	0.31
1906	35.8	19.3	5.7	6.1	20.67	28	0.68	NA	0.40	0.40
1907	45.0	20.0	5.4	5.8	20.67	NA	0.65	NA	0.45	0.38
1908	28.7	13.2	4.2	4.6	20.67	21	0.53	NA	0.45	0.29
1909	22.0	13.1	4.3	5.4	20.67	25	0.53	NA	0.40	0.30
1910	22.3	12.9	4.4	5.4	20.67	33	0.53	NA	0.40	0.34
1911	20.1	12.6	4.4	5.7	20.67	43	0.53	NA	0.40	0.42
1912	22.0	16.5	4.5	6.9	20.67	45	0.62	0.20	0.40	0.46
1913	23.6	15.5	4.4	5.6	20.67	45	0.61	0.30	0.42	0.44
1914	18.6	13.3	3.9	5.1	20.67	45	0.56	1.02	0.41	0.34
1915	34.0	17.5	4.7	14.2	20.67	47	0.51	1.02	0.41	0.39
1916	60.8	28.4	6.9	13.6	20.67	83	0.67	1.02	0.42	0.43
1917	51.7	29.2	8.6	8.9	20.67	103	0.84	1.43	0.42	0.62
1918	33.5	24.7	7.1	8.0	20.67	106	0.98	1.48	0.41	0.89
1919	32.1	18.2	5.8	7.0	20.67	115	1.12	1.17	0.40	0.64
1920	32.7	17.5	8.2	7.8	20.67	111	1.02	0.51	0.42	0.49
1921	22.1	12.7	4.7	4.7	20.67	75	0.63	0.71	0.42	0.30
1922	18.7	13.6	5.7	5.7	20.67	98	0.68	0.22	0.38	0.33
1923	25.4	14.7	7.4	6.7	20.67	117	0.65	0.77	0.36	0.43
1924	27.0	13.3	8.3	6.3	20.67	119	0.67	0.92	0.30	0.50
1925	27.0	14.3	9.1	7.7	20.67	119	0.69	0.41	0.33	0.58
1926	27.0	14.1	8.4	7.4	20.67	113	0.62	0.71	0.36	0.65
1927	25.4	13.1	6.8	6.3	20.67	85	0.57	0.77	0.35	0.64
1928	24.3	14.8	6.3	6.0	20.67	79	0.58	1.02	0.37	0.50
1929	24.3	18.4	6.8	6.5	20.67	68	0.53	0.51	0.35	0.45
1930	23.8	13.2	5.5	4.6	20.67	44	0.38	0.56	0.35	0.32
1931	23.3	8.4	4.3	3.6	20.67	32	0.29	0.43	0.35	0.24
1932	23.3	5.8	3.2	2.9	20.67	32	0.28	0.51	0.35	0.22
1933	23.3	7.3	3.9	4.0	20.67	31	0.35	0.76	0.35	0.39
1934	23.4	8.7	3.9	4.2	35.00	34	0.48	0.71	0.35	0.52
1935	20.0	8.9	4.1	4.4	35.00	33	0.64	0.71	0.35	0.50
1936	20.5	9.7	4.7	4.9	35.00	42	0.45	0.67	0.35	0.46
1937	19.9	13.4	6.0	6.5	35.00	47	0.45	0.69	0.35	0.54
1938	20.0	10.2	4.7	4.6	35.00	34	0.43	0.71	0.35	0.42
1939	20.0	11.2	5.0	5.1	35.00	36	0.39	0.69	0.35	0.50
1940	18.7	11.5	5.2	6.4	35.00	36	0.35	0.70	0.35	0.50
1941	16.5	12.0	5.8	7.5	35.00	36	0.35	0.69	0.35	0.52
1942	15.0	12.0	6.5	8.3	35.00	36	0.38	0.72	0.32	0.52
1943	15.0	12.0	6.5	8.3	35.00	35	0.45	0.74	0.32	0.52
1944	15.0	12.0	6.5	8.3	35.00	35	0.45	0.79	0.32	0.52
1945	15.0	12.0	6.5	8.3	35.00	35	0.52	0.78	0.32	0.52
1946	15.0	14.1	8.1	8.7	35.00	53	0.80	0.82	0.35	0.54
1947	15.0	21.3	14.7	10.5	35.00	62	0.72	0.83	0.35	0.78
1948	15.7	22.3	18.1	13.6	35.00	92	0.74	0.85	0.36	0.99

(Continued)

Table 2.15. (Continued).

1949	17.0	19.5	15.4	12.2	35.00	75	0.72	0.95	0.40	0.99
1950	17.7	21.6	13.3	13.9	35.00	76	0.74	0.98	0.45	0.96
1951	19.0	24.5	17.5	18.0	35.00	93	0.89	1.05	0.54	1.27
1952	19.4	24.5	16.5	16.2	35.00	93	0.85	1.07	0.57	1.21
1953	20.9	29.0	13.5	10.8	35.00	93	0.85	1.10	0.60	0.96
1954	21.8	29.9	14.1	10.7	35.00	88	0.85	1.16	0.61	0.92
1955	23.7	37.5	15.1	12.3	35.00	94	0.89	1.17	0.66	0.95
1956	24.0	42.0	16.0	13.5	35.00	105	0.91	1.23	0.65	1.02
1957	25.4	30.2	14.7	11.4	35.00	90	0.91	1.29	0.74	0.96
1958	24.8	26.3	12.1	10.3	35.00	66	0.89	1.34	0.74	0.95
1959	24.7	31.0	12.2	11.5	35.00	72	0.91	1.37	0.74	1.02
1960	26.0	32.3	11.9	13.0	35.00	83	0.91	1.38	0.74	1.02
1961	25.5	30.3	10.9	11.6	35.00	83	0.92	1.47	0.78	1.13
1962	23.9	31.0	9.6	11.6	35.00	83	1.09	1.50	0.80	1.15
1963	22.6	31.0	11.2	12.0	35.00	82	1.28	1.50	0.79	1.17
1964	23.7	32.3	13.6	13.6	35.00	90	1.29	1.59	0.79	1.58
1965	24.5	35.4	16.0	14.5	35.00	100	1.29	1.66	0.79	1.78
1966	24.5	36.0	15.1	14.5	35.00	100	1.29	1.65	0.79	1.64
1967	25.0	38.1	14.0	13.8	35.00	111	1.55	1.69	0.88	1.53
1968	25.6	41.2	13.2	13.5	40.12	117	2.14	1.74	0.95	1.48
1969	27.2	47.6	14.9	14.7	41.68	124	1.79	1.80	1.05	1.64
1970	28.7	58.1	15.7	15.3	36.39	133	1.77	1.77	1.29	1.74
1971	29.0	52.2	13.9	16.1	41.37	121	1.55	1.82	1.33	1.67
1972	25.0	51.3	15.0	17.7	58.47	121	1.68	1.77	1.40	1.77
1973	26.4	59.4	16.3	20.7	97.98	150	2.56	1.76	1.53	2.28
1974	43.1	77.1	22.5	36.0	159.87	181	4.71	2.12	1.74	3.96
1975	34.8	64.0	21.5	39.0	161.43	164	4.42	2.83	2.07	3.40
1976	41.2	69.4	23.1	37.0	125.35	162	4.35	3.25	2.25	3.80
1977	47.6	66.7	30.7	34.4	148.36	157	4.62	4.85	2.27	5.35
1978	50.8	65.8	33.7	31.0	193.46	261	5.40	9.21	2.04	6.30
1979	70.8	92.1	52.6	37.3	307.61	445	11.09	23.13	2.66	7.35
1980	76.2	101.2	42.5	37.4	612.74	677	20.63	9.39	2.83	8.48
1981	59.9	84.4	36.5	44.6	460.33	446	10.52	6.40	2.71	7.35
1982	46.7	73.0	25.5	38.5	376.35	327	7.95	4.09	2.18	6.53
1983	68.5	76.7	21.7	41.4	423.01	424	11.44	3.65	2.12	6.53
1984	61.2	66.7	25.6	48.5	360.80	357	8.14	3.56	2.16	6.26
1985	49.0	67.1	19.1	40.4	317.26	291	6.14	3.25	2.26	5.94
1986	55.8	66.2	22.0	38.0	367.02	461	5.47	2.87	1.76	3.83
1987	72.1	82.6	35.9	41.9	478.99	553	7.01	2.90	2.20	4.19
1988	110.2	120.7	37.1	60.3	438.56	523	6.53	3.45	6.26	4.41
1989	88.0	131.1	39.4	82.1	382.57	507	5.50	3.37	6.03	5.22
1990	73.9	122.9	45.8	74.4	385.68	467	4.82	2.85	4.02	3.86
1991	59.4	109.3	33.5	52.6	363.91	371	4.04	2.38	3.70	3.63
1992	57.6	107.5	35.1	58.5	345.25	361	3.94	2.20	3.18	4.02
1993	53.5	91.6	31.7	46.3	360.80	375	4.30	2.34	2.40	3.50
1994	71.2	111.1	37.2	49.4	385.68	411	5.29	4.76	2.88	3.69
1995	85.7	138.3	42.3	55.8	385.68	425	5.15	7.89	3.73	4.15
1996	71.2	108.9	49.0	51.3	388.79	398	5.19	3.78	3.40	4.12
1997	77.1	107.0	46.7	64.4	332.81	397	4.89	4.30	3.14	3.81
1998	65.3	78.5	45.3	51.3	295.17	375	5.54	3.40	2.10	3.73
1999	65.8	75.7	43.7	53.5	279.93	379	5.26	2.65	2.73	3.66
2000	74.4	88.0	43.6	55.8	280.24	549	5.01	2.55	3.92	3.70

(Continued)

Table 2.15. (Continued).

Year	Al (¢/lb)	Cu (¢/lb)	Pb (¢/lb)	Zn (¢/lb)	Au ($/tr oz)	Pt ($/tr oz)	Ag ($/tr oz)	Mo ($/lb)	Ni ($/lb)	Sn ($/lb)
2001	68.9	76.7	43.6	44.0	272.16	533	4.35	2.35	2.70	3.15
2002	64.9	75.7	43.6	38.6	311.03	543	4.60	3.76	3.07	2.92
2003	68.0	85.3	43.8	40.6	363.91	694	4.88	5.35	4.37	3.40
2004	83.9	133.8	55.3	52.6	410.57	849	6.44	16.65	6.26	5.49
2005	91.2	173.7	61.2	67.1	444.78	900	7.34	31.80	6.67	3.61
2006	121.6	314.8	77.6	158.8	606.52	1144	11.60	24.77	10.98	4.19
2007	122.0	327.9	123.8	154.2	696.72	1308	13.44	30.30	16.87	6.80
2008	120.7	319.3	120.2	88.9	874.01	1578	15.02	28.58	9.57	8.66
2009	79.4	241.3	87.1	78.0	973.54	1208	14.68	11.70	6.62	6.40
2010	104.3	348.3	108.9	102.1	1228.59	1616	20.00	15.80	9.89	9.53
2011	120.0	405.0	124.0	106.0	1600.00	1720	34.50	15.83	10.30	16.40

valuation is generally poor due to the cyclic behavior of the prices. The problem is shown diagrammatically in Figure 2.10.

One must decide the base price to be used as well as the trend angle and project the results over the depreciation period as a minimum. Another alternative to the selection of the current price as the base price is to use a recent price history over the past two or perhaps five years. For a valuation being done in July 1989 the price was \$1.15/lb. Averaging this value with those over the past three years would yield

Years	Base value	% change
1989	1.15	0
1988–89	1.18	−4.8
1987–89	1.06	+28.3
1986–89	0.96	+42.5

Inflation has not been accounted for in these figures. The point being that a wide range of base values can be calculated. The same is obviously true for determining the 'slope' of the trend line. This can be reflected by the percent change over the period of interest. These values have been added to the above table. They have been calculated by

$$\text{Percent change} = \left(\frac{\text{Price (1989)} - \text{Price Y}}{\text{Price Y}}\right) 100\%$$

The conclusion is that due to the cyclic nature of the prices, several cycles must be examined in arriving at both a representative base price and a trend.

There are two approaches which will be briefly discussed for price forecasting. These are:
– trend analysis,
– use of econometric models.

2.3.3 *Trend analysis*

The basic idea in trend analysis is to try and replace the actual price-time history with a mathematical representation which can be used for projection into the future. In examining

Table 2.16. Monthly Metal Prices (Metal Bulletin, 2012). Copyright 2012 Metal Bulletin; reproduced with permission.

		Al (¢/lb)	Cu (¢/lb)	Pb (¢/lb)	Zn (¢/lb)	Au ($/tr oz)	Pt ($/tr oz)	Ag ($/tr oz)	Pd ($/tr oz)	Mo Oxide ($/lb)	Ni ($/lb)	Sn ($/lb)
1997	Jan.	71	110	31.4	49	355	359	4.77	121	4.51	3.21	2.67
	Feb.	72	109	29.9	53	347	365	5.07	136	4.77	3.51	2.67
	Mar.	74	110	31.5	57	352	380	5.20	149	4.77	3.58	2.68
	Apr.	71	108	29.1	56	344	371	4.77	154	4.77	3.32	2.59
	May	74	114	28.0	59	344	390	4.76	171	4.71	3.39	2.59
	June	71	118	27.9	61	341	431	4.75	204	4.76	3.20	2.52
	July	72	111	28.8	69	324	416	4.37	188	4.73	3.10	2.47
	Aug.	78	102	27.6	75	324	425	4.50	215	4.73	3.07	2.46
	Sept.	73	96	28.8	74	323	425	4.73	191	4.51	2.95	2.49
	Oct.	73	93	27.2	58	325	424	5.03	205	4.23	2.89	2.52
	Nov.	73	87	25.5	53	307	293	5.08	208	3.99	2.79	2.57
	Dec.	69	80	23.9	50	289	367	5.79	199	3.98	2.70	2.50
	Avg.	*73*	*103*	*28.3*	*60*	*331*	*387*	*4.90*	*178*	*4.54*	*3.14*	*2.56*
1998	Jan.	67	77	24.1	50	297	375	5.88	226	3.97	2.49	2.36
	Feb.	66	75	23.4	47	289	386	6.83	237	4.04	2.44	2.38
	Mar.	65	79	25.4	47	296	399	6.24	262	4.49	2.45	2.48
	Apr.	64	82	25.9	50	308	414	6.33	321	4.40	2.45	2.59
	May	62	79	24.6	48	299	389	5.56	354	4.11	2.28	2.66
	June	59	75	23.9	46	292	356	5.27	287	4.09	2.03	2.71
	July	59	75	24.8	47	293	378	5.46	307	3.99	1.96	2.56
	Aug.	59	73	24.3	47	284	370	5.18	288	3.58	1.85	2.58
	Sept.	61	75	23.6	45	289	360	5.00	283	3.11	1.86	2.49
	Oct.	59	72	22.3	43	296	343	5.00	277	2.71	1.76	2.46
	Nov.	59	71	22.4	44	294	347	4.97	277	2.38	1.87	2.48
	Dec.	57	67	22.7	43	291	350	4.88	297	2.75	1.76	2.39
	Avg.	*62*	*75*	*24.0*	*46*	*294*	*372*	*5.55*	*285*	*3.64*	*2.10*	*2.51*
1999	Jan.	55	65	22.3	42	287	355	5.15	322	2.81	1.94	2.32
	Feb.	54	64	23.3	46	287	365	5.53	352	2.85	2.10	2.39
	Mar.	54	63	23.0	47	286	370	5.19	353	2.82	2.27	2.43
	Apr.	58	66	23.5	46	282	358	5.07	362	2.64	2.31	2.45
	May	60	69	24.5	47	277	356	5.27	330	2.61	2.45	2.56
	June	60	65	22.5	45	261	357	5.03	337	2.74	2.36	2.39
	July	64	74	22.5	49	256	349	5.18	332	2.69	2.59	2.37
	Aug.	65	75	22.8	51	257	350	5.27	340	2.73	2.93	2.37
	Sept.	68	79	23.0	54	265	372	5.23	362	2.91	3.19	2.42
	Oct.	67	78	22.5	52	311	423	5.41	387	2.81	3.32	2.46
	Nov.	67	78	21.7	52	293	435	5.16	401	2.72	3.61	2.65
	Dec.	70	80	21.7	54	284	441	5.16	425	2.71	3.67	2.60
	Avg.	*62*	*71*	*22.8*	*49*	*279*	*378*	*5.22*	*359*	*2.75*	*2.73*	*2.45*
2000	Jan.	76	84	21.4	53	284	441	5.19	452	2.67	3.77	2.69
	Feb.	76	82	20.5	50	300	517	5.25	636	2.65	4.38	2.56
	Mar.	72	79	20.0	51	286	481	5.06	667	2.65	4.66	2.48
	Apr.	66	76	19.1	51	280	498	5.06	572	2.65	4.41	2.44
	May	67	81	18.7	52	275	527	4.99	571	2.69	4.59	2.47
	June	68	79	19.0	51	286	560	5.00	647	2.93	3.82	2.48

(Continued)

Table 2.16. (Continued).

		Al (¢/lb)	Cu (¢/lb)	Pb (¢/lb)	Zn (¢/lb)	Au ($/tr oz)	Pt ($/tr oz)	Ag ($/tr oz)	Pd ($/tr oz)	Mo Oxide ($/lb)	Ni ($/lb)	Sn ($/lb)
	July	71	82	20.5	52	281	560	4.97	702	2.94	3.70	2.42
	Aug.	69	84	21.5	53	274	578	4.88	760	2.79	3.63	2.41
	Sept.	73	89	22.1	56	274	593	4.89	728	2.77	3.92	2.48
	Oct.	68	86	22.0	50	270	579	4.83	739	2.69	3.48	2.40
	Nov.	67	81	21.2	48	266	594	4.68	784	2.55	3.33	2.39
	Dec.	71	84	21.0	48	272	611	4.64	917	2.51	3.32	2.37
	Avg.	*70*	*82*	*20.6*	*51*	*279*	*545*	*4.95*	*681*	*2.71*	*3.92*	*2.47*
2001	Jan.	73	81	21.7	47	266	622	4.66	1040	2.33	3.17	2.35
	Feb.	73	80	22.7	46	262	601	4.55	975	2.38	2.96	2.32
	Mar.	68	79	22.6	46	263	586	4.40	782	2.41	2.78	2.29
	Apr.	68	75	21.6	44	261	595	4.37	696	2.41	2.87	2.24
	May	70	76	21.1	43	272	610	4.43	655	2.44	3.20	2.24
	June	66	73	20.1	41	270	580	4.36	614	2.58	3.01	2.19
	July	64	69	20.9	39	268	532	4.25	526	2.63	2.69	1.97
	Aug.	62	66	21.9	38	272	451	4.20	455	2.45	2.50	1.77
	Sept.	61	65	21.0	36	284	458	4.35	445	2.45	2.28	1.68
	Oct.	58	62	21.2	35	283	432	4.40	335	2.45	2.19	1.70
	Nov.	60	65	22.0	35	276	430	4.12	328	2.44	2.30	1.83
	Dec.	61	67	21.9	34	276	462	4.35	400	2.43	2.39	1.82
	Avg.	*65*	*72*	*21.6*	*40*	*271*	*530*	*4.37*	*604*	*2.45*	*2.70*	*2.03*
2002	Jan.	62	68	23.3	36	281	473	4.51	411	2.62	2.74	1.75
	Feb.	62	71	21.8	35	295	471	4.42	374	2.75	2.73	1.69
	Mar.	64	73	21.8	37	294	512	4.53	374	2.86	2.97	1.74
	Apr.	62	72	21.4	37	303	541	4.57	370	2.89	3.16	1.83
	May	61	72	20.5	35	314	535	4.71	357	2.99	3.07	1.88
	June	61	75	20.0	35	322	557	4.89	335	7.05	3.23	1.94
	July	61	72	20.2	36	314	526	4.92	323	6.05	3.24	1.96
	Aug.	59	67	19.2	34	310	545	4.55	324	4.76	3.05	1.74
	Sept.	59	67	19.1	34	319	555	4.55	327	4.78	3.01	1.79
	Oct.	59	67	19.0	34	317	581	4.40	317	4.71	3.09	1.92
	Nov.	62	72	20.0	35	319	588	4.51	286	4.04	3.32	1.92
	Dec.	62	72	20.1	36	333	597	4.63	243	3.44	3.26	1.92
	Avg.	*61*	*71*	*20.5*	*35*	*310*	*540*	*4.60*	*337*	*4.08*	*3.07*	*1.84*
2003	Jan.	63	75	20.1	35	357	630	4.81	255	3.62	3.64	2.01
	Feb.	65	76	21.5	36	360	682	4.65	253	3.75	3.91	2.07
	Mar.	63	75	20.7	36	341	677	4.49	226	4.51	3.80	2.09
	Apr.	60	72	19.8	34	328	625	4.49	163	5.21	3.59	2.07
	May	63	75	21.0	35	355	650	4.74	167	5.21	3.78	2.15
	June	64	76	21.2	36	356	662	4.53	180	5.67	4.03	2.13
	July	65	78	23.3	38	351	682	4.80	173	5.87	3.99	2.15
	Aug.	66	80	22.5	37	360	693	4.99	182	5.61	4.24	2.19
	Sept.	64	81	23.6	37	379	705	5.17	211	5.61	4.52	2.23
	Oct.	67	87	26.6	41	379	732	5.00	202	6.11	5.01	2.38
	Nov.	68	93	28.2	41	389	760	5.18	197	6.25	5.48	2.43
	Dec.	71	100	31.4	44	407	808	5.62	198	6.25	6.42	2.75
	Avg.	*65*	*81*	*23.3*	*38*	*364*	*692*	*4.87*	*201*	*5.31*	*4.37*	*2.22*

(Continued)

Table 2.16. (Continued).

2004	Jan.	73	110	34	46	414	851	6.32	216	7.68	6.96	2.94
	Feb.	76	125	40	49	405	846	6.44	235	8.15	6.87	3.03
	Mar.	75	136	40	50	407	899	7.23	268	8.88	6.22	3.46
	Apr.	78	134	34	47	403	882	7.06	297	12.74	5.83	4.06
	May	74	124	37	47	384	810	5.85	246	14.00	5.05	4.29
	June	76	122	39	46	392	807	5.86	229	14.97	6.14	4.18
	July	78	127	43	45	398	809	6.31	221	15.75	6.82	4.10
	Aug.	77	129	42	44	401	847	6.66	216	16.72	6.21	4.09
	Sept.	78	131	42	44	405	848	6.39	211	18.28	6.02	4.09
	Oct.	83	137	42	48	420	844	7.10	218	19.63	6.54	4.10
	Nov.	82	142	44	50	439	854	7.49	214	22.44	6.37	4.11
	Dec.	84	143	44	54	442	851	7.10	192	25.71	6.25	3.88
	Avg.	*78*	*130*	*40*	*48*	*409*	*846*	*6.65*	*230*	*15.41*	*6.27*	*3.86*
2005	Jan.	83	144	43	57	424	859	6.61	186	33.00	6.58	3.51
	Feb.	85	148	44	60	423	865	7.03	182	30.75	6.96	3.67
	Mar.	90	153	46	62	434	868	7.26	197	32.06	7.34	3.82
	Apr.	86	154	45	59	429	866	7.12	198	35.50	7.32	3.69
	May	79	147	45	56	422	867	7.02	190	35.89	7.68	3.69
	June	79	160	45	58	431	880	7.31	186	37.61	7.33	3.46
	July	81	164	39	54	424	874	7.01	184	32.44	6.61	3.25
	Aug.	85	172	40	59	438	900	7.04	187	30.00	6.76	3.26
	Sept.	83	175	42	63	456	915	7.15	189	31.67	6.45	3.08
	Oct.	87	184	46	68	470	931	7.67	208	32.13	5.63	2.91
	Nov.	93	194	46	73	477	963	7.87	246	31.25	5.50	2.79
	Dec.	102	208	51	83	510	978	8.64	265	29.67	6.09	3.05
	Avg.	*86*	*167*	*44*	*63*	*445*	*897*	*7.31*	*202*	*32.66*	*6.69*	*3.35*
2006	Jan.	108	215	57	95	550	1029	9.15	273	24.14	6.60	3.20
	Feb.	111	226	58	101	555	1041	9.53	289	23.84	6.79	3.55
	Mar.	110	231	54	110	557	1042	10.38	309	23.17	6.76	3.60
	Apr.	119	290	53	140	611	1102	12.61	353	22.93	8.14	4.02
	May	130	365	53	162	675	1264	13.45	369	24.58	9.56	4.01
	June	112	326	44	146	596	1189	10.80	315	26.22	9.41	3.58
	July	114	350	48	151	634	1229	11.23	319	25.96	12.06	3.82
	Aug.	112	349	53	152	633	1233	12.18	329	25.45	13.95	3.86
	Sept.	112	345	61	154	598	1186	11.68	324	27.03	13.67	4.10
	Oct.	120	340	69	173	586	1085	11.56	313	25.89	14.83	4.43
	Nov.	123	319	74	199	628	1183	12.93	325	25.56	14.57	4.57
	Dec.	128	303	78	200	630	1121	13.36	326	25.50	15.68	5.06
	Avg.	*117*	*305*	*59*	*149*	*604*	*1142*	*11.57*	*320*	*25.02*	*11.00*	*3.98*
2007	Jan.	127	257	76	172	631	1147	12.84	337	25.22	16.70	5.15
	Feb.	128	257	81	150	665	1205	13.91	342	25.53	18.68	5.87
	Mar.	125	293	87	148	655	1219	13.18	350	27.88	21.01	6.30
	Apr.	128	352	91	161	679	1278	13.74	369	28.50	22.80	6.37
	May	127	348	95	174	667	1303	13.15	368	30.10	23.67	6.42
	June	121	339	110	163	655	1287	13.14	368	33.63	18.92	6.40
	July	124	362	140	161	665	1304	12.91	366	32.50	15.16	6.69
	Aug.	114	341	141	148	665	1264	12.36	344	31.44	12.54	6.88
	Sept.	108	347	146	131	713	1307	12.83	335	32.00	13.40	6.81
	Oct.	111	363	169	135	755	1410	13.67	365	32.00	14.09	7.29
	Nov.	114	316	151	115	806	1449	14.70	363	32.75	13.88	7.57
	Dec.	108	299	118	107	803	1487	14.30	351	32.75	11.79	7.38
	Avg.	*120*	*323*	*117*	*147*	*697*	*1305*	*13.39*	*355*	*30.36*	*16.89*	*6.59*

(Continued)

Table 2.16. (Continued).

		Al (¢/lb)	Cu (¢/lb)	Pb (¢/lb)	Zn (¢/lb)	Au ($/tr oz)	Pt ($/tr oz)	Ag ($/tr oz)	Pd ($/tr oz)	Mo Oxide ($/lb)	Ni ($/lb)	Sn ($/lb)
2008	Jan.	111	320	118	106	890	1582	15.96	374	32.81	12.56	7.41
	Feb.	126	358	140	111	922	1995	17.57	466	32.88	12.68	7.81
	Mar.	136	383	136	114	968	2058	19.51	492	32.88	14.16	8.98
	Apr.	134	394	128	103	910	1990	17.50	447	32.88	13.05	9.82
	May	132	380	101	99	889	2055	17.05	437	32.88	11.67	10.91
	June	134	375	85	86	889	2041	16.97	449	32.88	10.23	10.08
	July	139	382	88	84	940	1916	18.03	428	32.88	9.14	10.50
	Aug.	125	346	87	78	839	1492	14.69	316	33.20	8.59	9.08
	Sept.	115	317	85	79	830	1225	12.37	250	33.75	8.07	8.33
	Oct.	96	223	67	59	807	914	10.44	191	33.75	5.51	6.53
	Nov.	84	169	59	52	761	842	9.87	207	15.43	4.85	6.19
	Dec.	68	139	44	50	816	845	10.29	177	12.38	4.39	5.10
	Avg.	*117*	*316*	*95*	*85*	*872*	*1580*	*15.02*	*353*	*29.88*	*9.58*	*8.40*
2009	Jan.	64	146	51	54	859	953	11.29	188	10.08	5.13	5.16
	Feb.	60	150	50	50	943	1038	13.41	207	9.63	4.72	5.01
	Mar.	61	170	56	55	924	1082	13.12	203	9.17	4.40	4.84
	Apr.	64	200	63	63	890	1165	12.51	227	8.55	5.06	5.33
	May	66	207	65	67	929	1134	14.03	230	9.08	5.73	6.26
	June	71	227	76	71	946	1221	14.65	246	10.39	6.79	6.80
	July	76	237	76	72	934	1161	13.36	248	11.52	7.25	6.37
	Aug.	88	280	86	83	949	1245	14.35	275	17.25	8.91	6.74
	Sept.	83	281	100	85	997	1290	16.39	293	13.88	7.93	6.74
	Oct.	85	285	102	94	1043	1333	17.24	322	13.26	8.40	6.81
	Nov.	88	303	105	99	1127	1402	17.82	352	10.75	7.71	6.78
	Dec.	99	317	106	108	1135	1448	17.67	374	11.18	7.74	7.05
	Avg.	*76*	*234*	*78*	*75*	*973*	*1206*	*14.65*	*264*	*11.23*	*6.65*	*6.16*
2010	Jan.	101	335	107	110	1118	1564	17.79	434	14.16	8.36	8.04
	Feb.	93	311	96	98	1095	1521	15.87	424	15.84	8.61	7.42
	Mar.	100	339	99	103	1113	1600	17.11	461	17.71	10.19	7.96
	Apr.	105	351	103	107	1149	1718	18.10	534	17.89	11.81	8.47
	May	93	310	85	89	1205	1630	18.42	491	17.44	9.98	7.97
	June	88	295	77	79	1233	1552	18.45	462	14.76	8.79	7.86
	July	90	306	83	84	1193	1527	17.96	456	14.33	8.85	8.25
	Aug.	96	330	94	93	1216	1541	18.36	487	15.29	9.71	9.41
	Sept.	98	350	99	98	1271	1591	20.55	538	16.25	10.27	10.30
	Oct.	106	376	108	108	1342	1689	23.39	592	15.20	10.80	11.95
	Nov.	106	384	108	104	1370	1696	26.54	683	15.66	10.39	11.58
	Dec.	107	415	109	103	1391	1712	29.35	755	16.03	10.94	11.87
	Avg.	*99*	*342*	*97*	*98*	*1225*	*1612*	*20.16*	*526*	*15.88*	*9.89*	*9.26*
2011	Jan.	111	433	118	108	1356	1787	28.40	793	16.76	11.63	12.46
	Feb.	114	448	117	112	1373	1826	30.78	821	17.58	12.82	14.30
	Mar.	116	432	119	107	1424	1768	35.81	762	17.70	12.16	13.94
	Apr.	121	430	124	108	1474	1794	41.97	771	17.36	11.94	14.72
	May	118	405	110	98	1510	1788	36.75	736	17.25	10.98	13.03
	June	116	410	114	101	1529	1772	35.80	771	17.25	10.14	11.60
	July	114	436	122	108	1573	1757	37.92	788	14.94	10.76	12.39

(Continued)

Table 2.16. (Continued).

	Aug.	109	410	109	100	1756	1806	40.30	763	14.83	10.02	11.08
	Sept.	104	377	104	94	1772	1756	38.15	713	14.71	9.25	10.28
	Oct.	99	333	88	84	1665	1535	31.97	616	13.89	8.57	9.88
	Nov.	94	343	90	87	1739	1596	33.08	626	13.26	8.11	9.64
	Dec.	92	343	92	87	1652	1465	30.41	643	13.46	8.23	8.81
	Avg.	*109*	*400*	*109*	*99*	*1569*	*1721*	*35.11*	*733*	*15.75*	*10.38*	*11.84*
2012	Jan.	97	365	95	90	1656	1507	30.77	659	13.76	8.99	9.73
	Feb.	100	382	96	93	1743	1658	34.14	702	14.52	9.28	11.04
	Mar.	99	384	94	92	1674	1658	32.95	685	14.37	8.49	10.44
	Apr.	93	375	94	91	1650	1589	31.55	657	14.22	8.12	10.02
	May	91	359	91	88	1586	1469	28.67	617	14.10	7.72	9.24
	June	85	337	84	84	1597	1447	28.05	612	13.53	7.50	8.74
	July	85	344	85	84	1594	1427	27.43	579	12.81	7.33	8.44

Al – Aluminum Settlement LME Daily Official \$ per tonne Monthly Average
Cu – Copper Settlement LME Daily Official \$ per tonne Monthly Average
Pb – Lead Settlement LME Daily Official \$ per tonne Monthly Average
Zn – Zinc Settlement LME Daily Official \$ per tonne Monthly Average
Au – Gold London Afternoon Daily Price \$ per troy oz Monthly Average
Pt – Platinum London free market Morning \$ per troy oz Monthly average
Ag – Silver Spot London Brokers Official Daily price Cents per troy oz Monthly Average
Pd – Palladium London free market daily price change Morning \$ per troy oz Monthly average
Mo – Molybdenum canned molybdic oxide United States Free market \$ per lb Mo in warehouse Monthly average
Ni – Nickel Settlement LME Daily Official \$ per tonne Monthly Average
Sn – Tin Settlement LME Daily Official \$ per tonne Monthly Average.

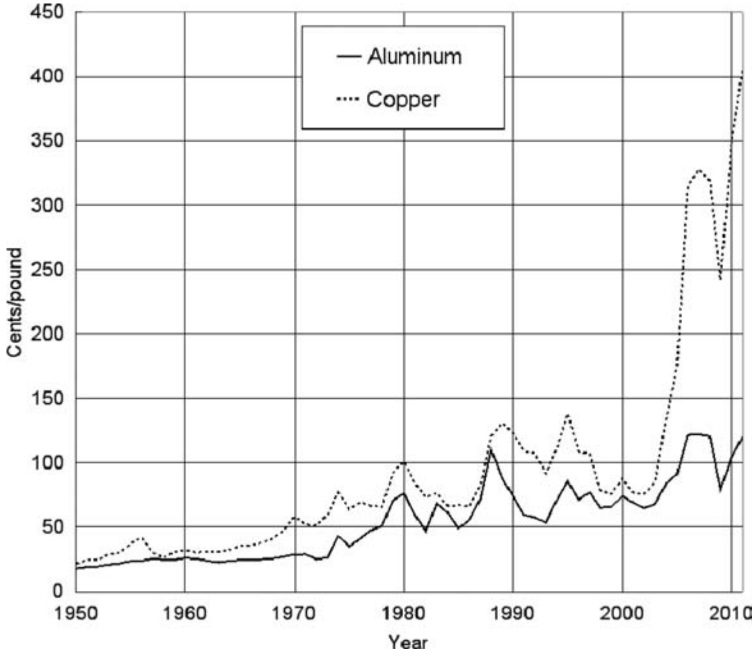


Figure 2.2. Price performance of copper and aluminum over the period 1950–2011.

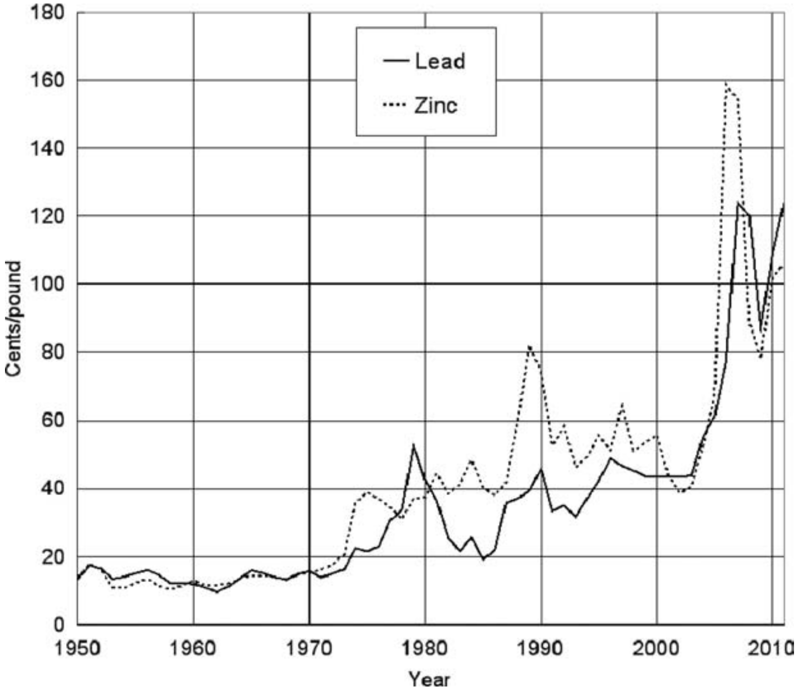


Figure 2.3. Price performance of lead and zinc over the period 1950–2011.

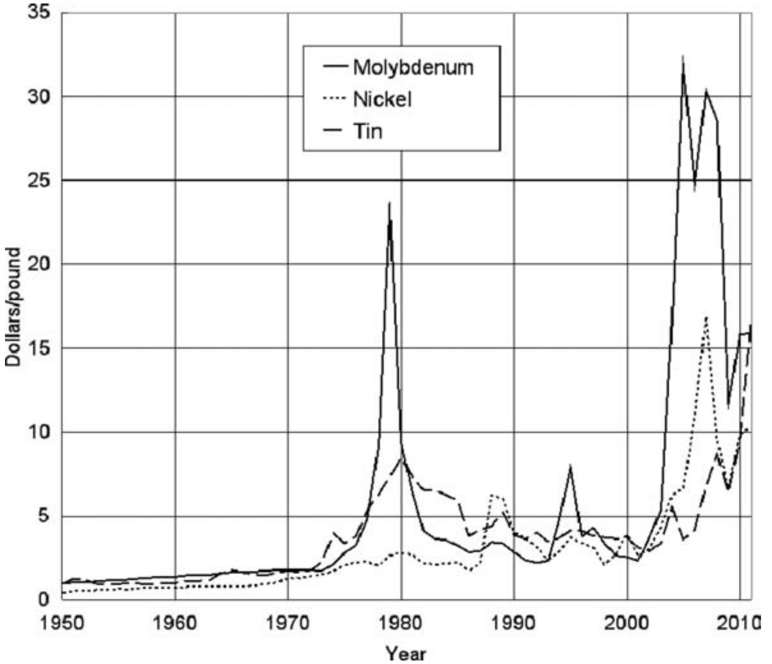


Figure 2.4. Price performance of molybdenum, nickel and tin over the period 1950–2011.

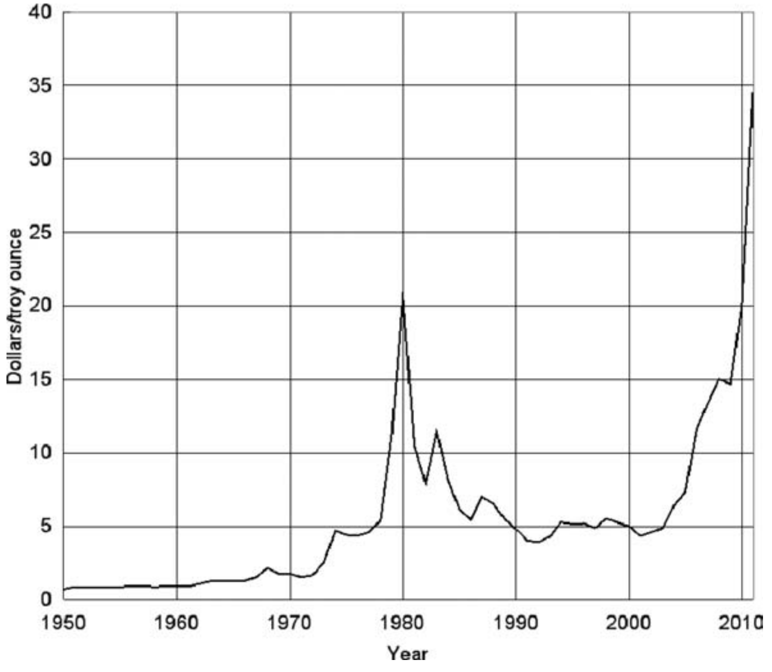


Figure 2.5. Price performance of silver over the period 1950–2011.

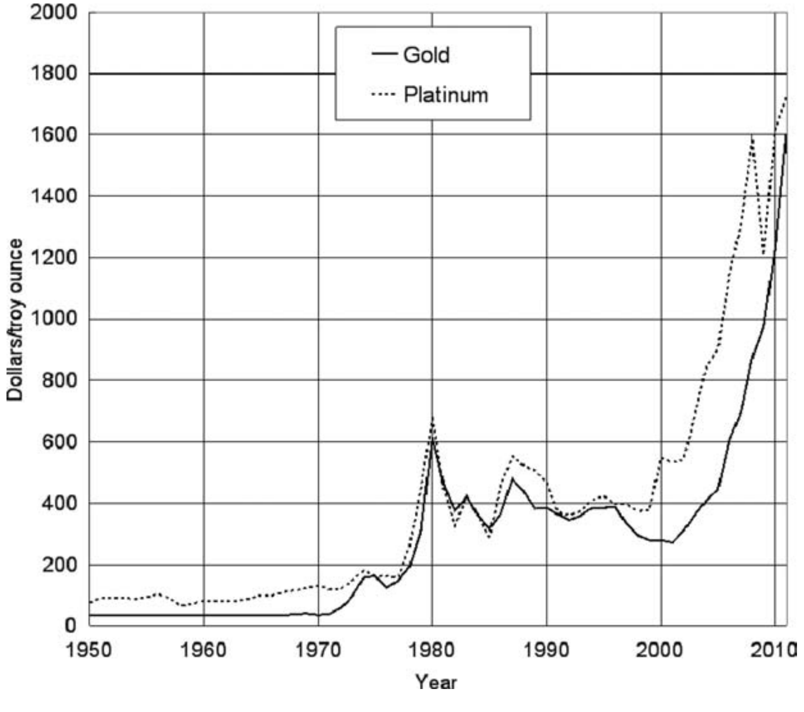


Figure 2.6. Price performance of gold and platinum over the period 1950–2011.

Table 2.17. Annual price of iron ore fines (USGS, 2012b).

Year	$/ton	Year	$/ton	Year	$/ton
1900	2.35	1938	2.56	1976	22.60
1901	1.68	1939	2.99	1977	25.00
1902	1.82	1940	2.52	1978	27.70
1903	1.88	1941	2.65	1979	30.80
1904	1.55	1942	2.61	1980	34.50
1905	1.75	1943	2.62	1981	37.50
1906	2.09	1944	2.69	1982	38.70
1907	2.53	1945	2.72	1983	46.30
1908	2.25	1946	3.01	1984	39.90
1909	2.13	1947	3.44	1985	38.60
1910	2.46	1948	3.88	1986	34.20
1911	1.98	1949	4.46	1987	29.60
1912	1.95	1950	4.92	1988	28.30
1913	2.12	1951	5.40	1989	31.30
1914	1.76	1952	6.21	1990	30.90
1915	1.83	1953	6.81	1991	30.10
1916	2.40	1954	6.76	1992	28.60
1917	3.12	1955	7.21	1993	25.80
1918	3.46	1956	7.68	1994	25.20
1919	3.20	1957	8.10	1995	27.70
1920	4.14	1958	8.27	1996	28.90
1921	3.00	1959	8.48	1997	29.90
1922	3.32	1960	8.35	1998	31.20
1923	3.44	1961	9.13	1999	26.80
1924	2.83	1962	8.82	2000	25.80
1925	2.57	1963	9.22	2001	24.50
1926	2.52	1964	9.46	2002	25.90
1927	2.50	1965	9.25	2003	31.00
1928	2.45	1966	9.50	2004	36.80
1929	2.65	1967	9.64	2005	43.80
1930	2.47	1968	9.78	2006	53.80
1931	2.36	1969	10.20	2007	59.40
1932	1.41	1970	10.40	2008	74.70
1933	3.52	1971	10.90	2009	93.20
1934	2.64	1972	12.10	2010	100.90
1935	2.66	1973	12.80	2011	120.00
1936	2.64	1974	15.50		
1937	2.82	1975	19.40		

a 'typical' curve one can see that it is cyclic and the cycles have different amplitudes. One could try to fit a function describing the behavior quite closely over a given time period using a type of regression analysis which is commonly available on computers as part of a statistical software package.

The general objective is to determine an equation of the form

$$y = a_0 + a_1x + a_2x^2 + a_3x^3 + a_4x^4 + \cdots + a_mx^m \tag{2.7}$$

where a_i are coefficients; y is the price in year $x; x$ is the year relative to the initial year $(x = 0)$.

Table 2.18. Monthly spot price for China import iron ore fines 62% FE (CFR Tianjin port), US Dollars per metric ton, (IMF, 2012).

Month	$/metric ton	Month	$/metric ton	Month	$/metric ton
Jan-00	12.45	Jan-08	60.80	Sep-10	140.63
...	...	...	...	Oct-10	148.48
Dec-00	12.45	Nov-08	60.80	Nov-10	160.55
Jan-01	12.99	Dec-08	69.98	Dec-10	168.53
...	...	Jan-09	72.51	Jan-11	179.63
Dec-01	12.99	Feb-09	75.59	Feb-11	187.18
Jan-02	12.68	Mar-09	64.07	Mar-11	169.36
...	...	Apr-09	59.78	Apr-11	179.26
Dec-02	12.68	May-09	62.69	May-11	177.10
Jan-03	13.82	Jun-09	71.66	Jun-11	170.88
...	...	Jul-09	83.95	Jul-11	172.98
Dec-03	13.82	Aug-09	97.67	Aug-11	177.45
Jan-04	16.39	Sep-09	80.71	Sep-11	177.23
...	...	Oct-09	86.79	Oct-11	150.43
Dec-04	16.39	Nov-09	99.26	Nov-11	135.54
Jan-05	28.11	Dec-09	105.25	Dec-11	136.46
...	...	Jan-10	125.91	Jan-12	140.35
Dec-05	28.11	Feb-10	127.62	Feb-12	140.40
Jan-06	33.45	Mar-10	139.77	Mar-12	144.66
...	...	Apr-10	172.47	Apr-12	147.65
Dec-06	33.45	May-10	161.35	May-12	136.27
Jan-07	36.63	Jun-10	143.63	Jun-12	134.62
...	...	Jul-10	126.36	Jul-12	127.94
Dec-07	36.63	Aug-10	145.34		

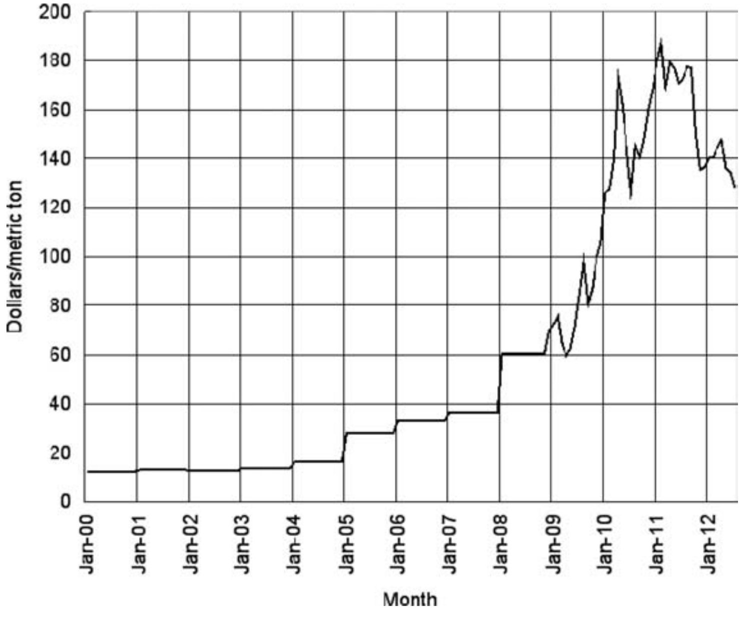


Figure 2.7. Monthly average price Jan-2000 to July-2012 for China import iron ore fines (62% Fe spot, CFR Tianjin port), US Dollars per metric ton (IMF, 2012).

Table 2.19. Annual steel production (in thousand tonnes) for the top twenty steel producing countries for years 2000 to 2011 ranked by year 2011 production, (World Steel Association, 2012).

Rank	Country	Year 2000	2001	2002	2003	2004	2005
1	China	128,500	151,634	182,249	222,336	272,798	355,790
2	Japan	106,444	102,866	107,745	110,511	112,718	112,471
3	United States	101,803	90,104	91,587	93,677	99,681	94,897
4	India	26,924	27,291	28,814	31,779	32,626	45,780
5	Russia	59,136	58,970	59,777	61,450	65,583	66,146
6	South Korea	43,107	43,852	45,390	46,310	47,521	47,820
7	Germany	46,376	44,803	45,015	44,809	46,374	44,524
8	Ukraine	31,767	33,108	34,050	36,932	38,738	38,641
9	Brazil	27,865	26,717	29,604	31,147	32,909	31,610
10	Turkey	14,325	14,981	16,467	18,298	20,478	20,965
11	Italy	26,759	26,545	26,066	27,058	28,604	29,350
12	Taiwan	16,896	17,261	18,230	18,832	19,599	18,942
13	Mexico	15,631	13,300	14,010	15,159	16,737	16,195
14	France	20,954	19,343	20,258	19,758	20,770	19,481
15	Spain	15,874	16,504	16,408	16,286	17,621	17,826
16	Canada	16,595	15,276	16,002	15,929	16,305	15,327
17	Iran	6,600	6,916	7,321	7,869	8,682	9,404
18	United Kingdom	15,155	13,543	11,667	13,268	13,766	13,239
19	Poland	10,498	8,809	8,368	9,107	10,593	8,336
20	Belgium	11,636	10,762	11,343	11,114	11,698	10,420

Rank	Country	Year 2006	2007	2008	2009	2010	2011
1	China	421,024	489,712	512,339	577,070	637,400	683,265
2	Japan	116,226	120,203	118,739	87,534	109,599	107,595
3	United States	98,557	98,102	91,350	58,196	80,495	86,247
4	India	49,450	53,468	57,791	63,527	68,321	72,200
5	Russia	70,830	72,387	68,510	60,011	66,942	68,743
6	South Korea	48,455	51,517	53,625	48,572	58,914	68,471
7	Germany	47,224	48,550	45,833	32,670	43,830	44,288
8	Ukraine	40,891	42,830	37,279	29,855	33,432	35,332
9	Brazil	30,901	33,782	33,716	26,506	32,928	35,162
10	Turkey	23,315	25,754	26,806	25,304	29,143	34,103
11	Italy	31,624	31,553	30,590	19,848	25,750	28,662
12	Taiwan	20,000	20,903	19,882	15,873	19,755	22,660
13	Mexico	16,447	17,573	17,209	14,132	16,870	18,145
14	France	19,852	19,250	17,879	12,840	15,414	15,777
15	Spain	18,391	18,999	18,640	14,358	16,343	15,591
16	Canada	15,493	15,572	14,845	9,286	13,013	13,090
17	Iran	9,789	10,051	9,964	10,908	11,995	13,040
18	United Kingdom	13,871	14,317	13,521	10,079	9,709	9,481
19	Poland	10,008	10,632	9,728	7,128	7,993	8,794
20	Belgium	11,631	10,692	10,673	5,635	7,973	8,114

If one has 10 pairs of data (price, year), then the maximum power of the polynomial which could be fitted is $m = 9$. As the power is increased, the actual behavior of the data could be more and more closely represented. Unfortunately while this is a good procedure for interpolation, that is, defining values for points within the range of the data, the equation

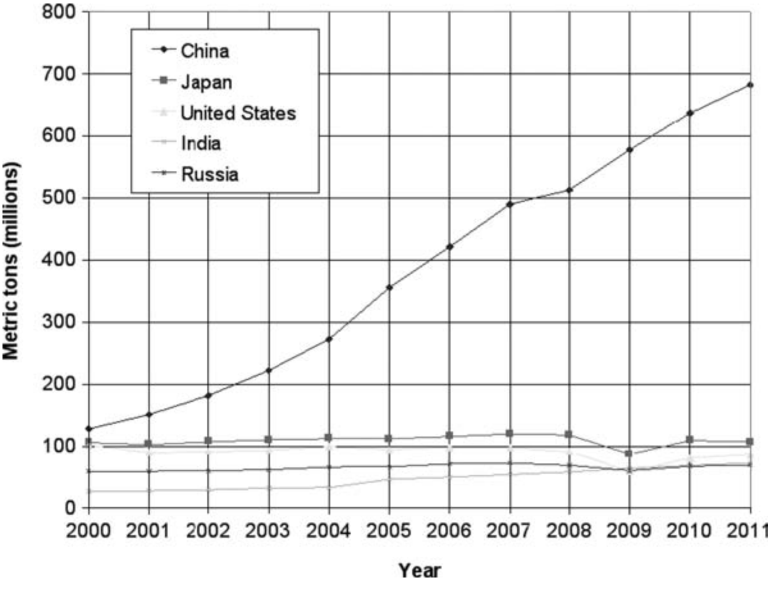


Figure 2.8. Annual steel production for the top five steel producing countries for years 2000 to 2011, (World Steel Association, 2012).

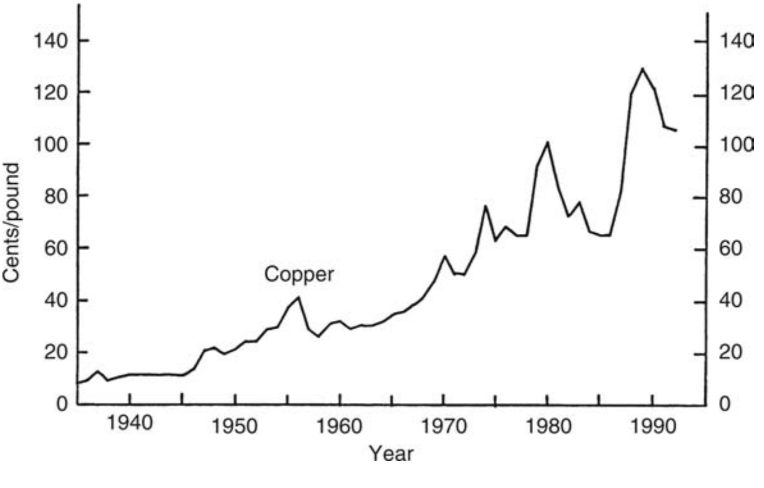


Figure 2.9. Average copper price by year over the time period 1935 to 1992 (Noble, 1979; *E/MJ*, 1992).

cannot be used for determining values beyond the endpoints (extrapolation). This however is what is desired, that of projecting the historical data past the end points into the future. It can be easily demonstrated that some of the terms of power 2 and higher can vary wildly both in sign and magnitude over only one year. Thus such a general power series representation is not of interest. There are some other possibilities however based upon the fitting of the first two terms of a power series. The simplest representation

$$y = a_0 + a_1 x \tag{2.8}$$

Table 2.20. Average annual copper prices 1935-92 (*E/MJ*, 1935–1992).

Calendar year	Relative year	Domestic copper (¢/lb)	Calendar year	Relative year	Domestic copper (¢/lb)
1935	0	8.649	1965	30	35.017
	1	9.474		31	36.170
	2	13.167		32	38.226
	3	10.000		33	41.847
	4	10.965		34	47.534
1940	5	11.296	1970	35	57.700
	6	11.797		36	51.433
	7	11.775		37	50.617
	8	11.775		38	58.852
	9	11.775		39	76.649
1945	10	11.775	1975	40	63.535
	11	13.820		41	68.824
	12	20.958		42	65.808
	13	22.038		43	65.510
	14	19.202		44	92.234
1950	15	21.235	1980	45	101.416
	16	24.200		46	83.744
	17	24.200		47	72.909
	18	28.798		48	77.861
	19	29.694		49	66.757
1955	20	37.491	1985	50	65.566
	21	41.818		51	64.652
	22	29.576		52	81.097
	23	25.764		53	119.106
	24	31.182		54	129.534
1960	25	32.053	1990	55	121.764
	26	29.921		56	107.927
	27	30.600		57	106.023
	28	30.600			
	29	31.960			

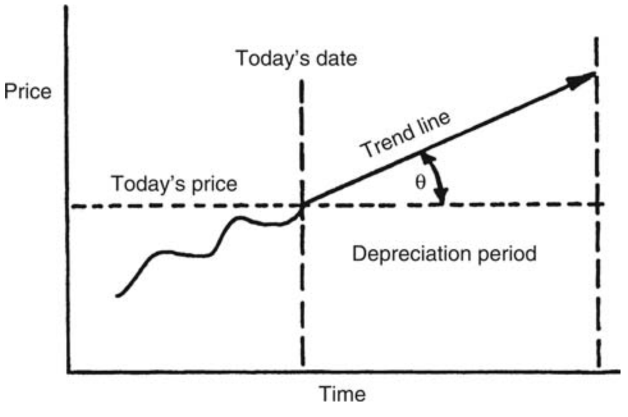


Figure 2.10. Diagrammatic representation of the price projection process.

represents a straight line with intercept a_0 ($x = 0$) and slope a_1. For this to apply the data should plot as a straight line on rectangular graph paper. Figure 2.9 shows such a plot for the copper data over the time period 1935–1992. The average trend over the period 1935 through 1970 might be fitted by such a straight line but then there is a rapid change in the rate of price growth. In examining the average trend it appears as if some type of non-linear function is required. The first approach by the engineer might be to try an exponential function such as

$$y = a\mathrm{e}^{bx} \tag{2.9}$$

Taking natural logs of both sides one finds that

$$\ln y = \ln a + bx \ln \mathrm{e}$$

Since the natural log of e is 1, then

$$\ln y = \ln a + bx \tag{2.10}$$

Letting

$$\begin{aligned} y^1 &= \ln y \\ a^1 &= \ln a \end{aligned}$$

Equation (2.10) becomes

$$y^1 = a^1 + bx \tag{2.11}$$

A straight line should now result when the natural log of the price is plotted versus the year. Such a plot, easily made using semi-log paper, is shown in Figure 2.11. A straight line can be made to fit the data quite well. In 1977, Noble (1979) fitted an equation of the form

$$y = a\mathrm{e}^{bx}$$

to the data in Table 2.20 for the period 1935 to 1976. For the least squares approach employed, the constants a and b are given by

$$b = \frac{\sum (x_i \ln y_i) - \frac{1}{n}\sum x_i \sum \ln y_i}{\sum x_i^2 - \frac{1}{n}\left(\sum x_i\right)^2} \tag{2.12}$$

$$a = \exp\left(\frac{\sum \ln y_i}{n} - b\frac{\sum x_i}{n}\right) \tag{2.13}$$

For the period of 1935 to 1976

$$n = 42$$

$$\begin{aligned} \sum (x_i \ln y_i) &= 3085.521 \\ \sum x_i &= 861 \\ \sum \ln y_i &= 136.039 \end{aligned}$$

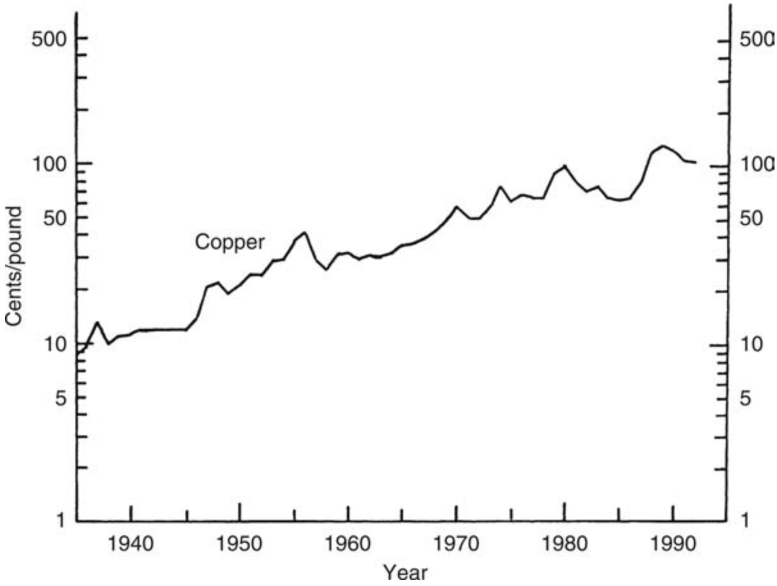


Figure 2.11. Logarithmic plot of copper price versus time for the time period 1935 to 1992.

$$\sum x_i^2 = 23{,}821$$

$$\sum (\ln y_i)^2 = 456.0775$$

Substituting the appropriate values into Equations (2.12) and (2.13) one finds that

$$b = \frac{3085.521 - \frac{1}{42}861 + 136.039}{23821 - \frac{1}{42}(861)^2} = 0.04809$$

$$a = \exp\left(\frac{136.039}{42} - \frac{0.04809}{42}861\right) = 9.5185$$

Hence the price predictor equation becomes

$$y = 9.5185e^{0.04809x} \tag{2.14}$$

The correlation coefficient r may be calculated using

$$r = \frac{S_{xy}}{\sqrt{S_{xx}S_{yy}}} \tag{2.15}$$

where

$$S_{xx} = n\sum x_i^2 - \left(\sum x_i\right)^2 = 259{,}161 \tag{2.16}$$

$$S_{yy} = n\sum (\ln y_i)^2 - \left(\sum \ln y_i\right)^2 = 648.65 \tag{2.17}$$

$$S_{xy} = n\sum (x_i \ln y_i) - \left(\sum x_i\right)\left(\sum \ln y_i\right) = 12{,}462.30 \tag{2.18}$$

In this case

$$r = 0.961$$

This high correlation coefficient indicates a strong relationship between price and time. Confidence limits (CL) for the estimates may be calculated using

$$\text{CL}(y) = a\text{e}^{bx \pm c} \tag{2.19}$$

The constant c is given by

$$c = t_{\alpha/2}(1 - r^2)^{1/2}\left[S_y^2 + \frac{(n - \bar{x})^2}{(n-2)S_x^2}\right]^{1/2} \tag{2.20}$$

where:

α is the probability (expressed as a decimal) of y being outside the confidence limits;
$t_{\alpha/2}$ is the value read from a Student 't' table for a cumulative probability (P) of $1 - \alpha/2$ and $n - 2$ degrees of freedom;
S_y^2 is the population variance of $\ln y$ $(=(\text{SD}_y)^2)$;
S_x^2 is the population variance of x $(=(\text{SD}_x)^2)$;
$\bar{x}$ is the mean (arithmetic value) of x;
(SD_y) is the standard deviation of $\ln y$;
(SD_x) is the standard deviation of x;
r is the correlation coefficient.

The required mean values, variances and standard deviations are given by

$$\overline{\ln y} = \frac{\sum \ln y_i}{n} = 3.2390 \tag{2.21}$$

$$S_y^2 = \frac{\sum (\overline{\ln y} - \ln y_i)^2}{n-1} = 0.3767 \tag{2.22}$$

$$S_y = \text{SD}_y = \sqrt{S_y^2} = 0.6137 \tag{2.23}$$

$$\bar{x} = \frac{\sum x_i}{n} = 20.5 \tag{2.24}$$

$$S_x^2 = \frac{\sum (\bar{x} - x_i)^2}{n-1} = 150.5 \tag{2.25}$$

$$S_x = \text{SD}_x = \sqrt{S_x^2} = 12.2678 \tag{2.26}$$

Substitution of these values into Equation (2.20) yields

$$c = t_{\alpha/2}(0.2765)\left[0.3767 + \frac{(x - 20.5)^2}{6020}\right]^{1/2} \tag{2.27}$$

If the probability of y being outside of the confidence limits is $\alpha = 0.20$ (80% probability that the price is within the upper and lower limits), then

$$P = 1 - 0.20/2 = 0.90$$

$$\text{DF} = 42 - 2 = 40$$

Table 2.21. Predicted and actual copper prices (Noble, 1979).

X	Year	Predicted average price (¢/lb)	Predicted range (¢/lb)		Actual price (¢/lb)
			Low	High	
42	1977	71.74	56.28	91.44	65.81
43	1978	75.27	58.94	96.13	65.51
44	1979	78.98	61.72	101.07	92.33
45	1980	82.87	64.63	106.27	101.42
46	1981	86.96	67.66	111.75	83.74
47	1982	91.24	70.84	117.51	72.91
48	1983	95.73	74.16	123.59	77.86
49	1984	100.45	77.63	129.98	66.76
50	1985	105.40	81.26	136.71	65.57
51	1986	110.59	85.05	143.80	66.10
52	1987	116.04	89.01	151.27	82.50
53	1988	121.76	93.16	159.14	120.51
Average		94.75	73.36	122.39	80.09

The value of $t_{\alpha/2}$ as read from the Student 't' table is

$$t_{\alpha/2} = 1.303$$

Equation (2.27) becomes

$$c = 0.3603\left[0.3767 + \frac{(x-20.5)^2}{6020}\right]^{1/2} \tag{2.28}$$

The average trend price value for 1976 ($x = 41$) is

$$y = 68.37 \text{ ¢/lb}$$

This happens to be very close to the actual 1976 price of 68.82 ¢/lb. The predicted average price and price ranges for the time period 1977–1988 ($x = 42$ to 53) are given in Table 2.21.

The actual average prices for the same years are also given. As can be seen for the time period 1982–1987, the actual value was considerably lower than the predicted lower limit. This corresponded to a very difficult time for producers. The conclusion is that it is very difficult to predict future price trends.

2.3.4 *Econometric models*

A commodity model is a quantitative representation of a commodity market or industry (Labys, 1977). The behavioral relationships included reflect supply and demand aspects of price determination, as well as other related economic, political and social phenomena.

There are a number of different methodologies applied to modelling mineral markets and industries. Each concentrates on different aspects of explaining history, analyzing policy and forecasting. The methodologies chosen for a model depend on the particular economic behavior of interest. It could be price determination, reserve and supply effects, or other aspects.

The market model is the most basic type of micro economic structure and the one from which other commodity methodologies have developed. It includes factors such as:

– Commodity demand, supply and prices;
– Prices of substitute commodities;
– Price lags;
– Commodity inventories;
– Income or activity level;
– Technical factors;
– Geological factors;
– Policy factors influencing the supply.

Market models, which balance supply and demand to produce an equilibrium price, are commonly used in the mineral business for: (a) historical explanation, (b) policy analysis decision making, and (c) prediction.

They are also used to simulate the possible effects of stockpiles and/or supply restrictions over time.

2.3.5 *Net smelter return*

For base metals such as copper, lead, and zinc, prices are not quoted for concentrates rather the refined metal price is given. The payment received by the company from the smelter for their concentrates (called the net smelter return (NSR)) (Lewis et al., 1978; Huss, 1984; Werner & Janakiraman, 1980) depends upon many factors besides metal price. The process by which the net smelter return is calculated is the subject of this section.

Assume that a mill produces a copper concentrate containing G percent of copper metal. The amount of contained metal in one ton of concentrate is

$$\mathrm{CM} = \frac{G}{100}2000 \tag{2.29}$$

where CM is the contained metal (lbs), G is the concentrate grade (% metal), and the ratio lbs/ton is 2000.

Most smelters and refiners pay for the contained metal based upon prices published in sources such as *Metals Week*. If the current market price ($/lb) is P then the contained copper value is

$$\mathrm{CV} = \frac{G}{100}2000P \tag{2.30}$$

where CV is the contained copper value ($/ton), P is the current market price ($/lb).

It is never possible for smelting and refining operations to recover one hundred percent of the contained metal. Some metal is lost in the slag, for example. To account for these losses, the smelter only pays for a portion of the metal content in the concentrate. The deductions may take one of three forms:

(1) Percentage deduction. The smelter pays only for a percentage (C) of the contained metal.

(2) Unit deduction. The concentrate grade is reduced by a certain fixed amount called the unit deduction. For minerals whose grade is expressed in percent one unit is one percentage point. For minerals whose grade is expressed in troy ounces, one unit is one troy ounce.

(3) A combination of percentage and unit deductions.

The 'effective' concentrate grade (G_e) is thus

$$G_e = \frac{G}{100}(G - u) \tag{2.31}$$

where G_e is the effective concentrate grade (%), u is the fixed unit deduction (%), C is the credited percentage of the metal content (%).

The payable (accountable) metal content in one ton of concentrates is

$$M_e = \frac{C}{100}\frac{G-u}{100}2000 \tag{2.32}$$

where M_e is the payable metal content (lbs).

Smelters sometimes pay only a certain percentage of the current market price. The factor relating the price paid to the market price is called the price factor. If 100% of the market price is paid then the price factor is 1.00. The gross value of one ton of concentrates is thus

$$GV = M_e P f \tag{2.33}$$

where f is the price factor, GV is the gross value ($/ton of concentrate).

To obtain the basic smelter return (BSR) the charges incurred during treatment, refining and selling must be taken into account. The basic equation is that given below

$$\mathrm{BSR} = M_e(Pf - r) - T \tag{2.34}$$

where r is the refining and selling cost ($/pound of payable metal), T is the treatment charge ($/ton of concentrate).

Often there are other metals/elements in the concentrate. Their presence can be advantageous in the sense that a by-product credit (Y) is received or deleterious resulting in a penalty charge (X).

The net smelter return (NSR) is expressed by

$$\mathrm{NSR} = M_e(Pf - r) - T - X + Y \tag{2.35}$$

where X is the penalty charge due to excessive amounts of certain elements in the concentrate ($/ton of concentrate) and Y is the credit for valuable by-products recovered from the concentrate ($/ton of concentrate).

Letting

$$P_e = Pf - r \tag{2.36}$$

where P_e is the effective metal price (after price reductions and refining charges) the NSR expression can be simplified to

$$\mathrm{NSR} = M_e P_e - T - X + Y \tag{2.37}$$

Long-term refining and treatment agreements generally contain cost and price escalation provisions. Escalation of refining charges (e_1) can be grouped into five distinct forms:

(1) No escalation.

(2) Predictive escalation. A specified rate of increase for each year of the contract based upon predictions of cost/price changes.

(3) Cost-indexed escalation. Escalation based upon published cost indices (e.g. wages, fuel and energy).

(4) Price based cost escalation. If the metal price increases above a certain level, the refining cost is increased. This allows the refinery to share in the gain. On the other hand if the price decreases below a certain level, the refining cost may or may not be decreased.

(5) Some combination of (2), (3) and (4).

Escalation of treatment charges (e_2) is generally either by predictive (No. 2) or cost-indexed (No. 3) means. Including escalation, the general NSR equation can be written as

$$\text{NSR} = M_e(Pf - r \pm e_1) - (T \pm e_2) - X + Y \tag{2.38}$$

Smelter contracts must cover all aspects of the sale and purchase of the concentrates from the moment that they leave the mine until final payment is made. Table 2.22 lists the elements of smelter contract and the questions to be addressed. Although the terms of an existing smelter contract are binding upon the contracting parties, supplementary agreements are usually made in respect to problems as they arise.

The net value of the concentrate to the mine is called the 'at-mine-revenue' or AMR. It is the net smelter return (NSR) minus the realization cost (R)

$$\text{AMR} = \text{NSR} - R \tag{2.39}$$

The realization cost covers such items as:

(a) Freight.
(b) Insurance.
(c) Sales agents' commissions.
(d) Representation at the smelter during weighing and sampling.

O'Hara (1980) presented the following formula for freight cost F in \$ Canadian (1979) per ton of concentrates

$$F = 0.17T_m^{0.9} + \$0.26R_m^{0.7} + \$0.80D_o \tag{2.40}$$

where F is the freight cost (\$/ton); T_m are miles by road (truck); R_m are miles by railroad; D_o are days of loading, ocean travel and unloading on a 15,000-ton freighter.

In 1989 US\$, the formula becomes

$$F = 0.26T_m^{0.9} + 0.39R_m^{0.7} + \$1.20D_o \tag{2.41}$$

The relationship between the AMR and the gross value of the metal contained in the concentrate (CV) is called the percent payment (PP)

$$\text{PP} = 100 \times \frac{\text{AMR}}{\text{CV}} \tag{2.42}$$

For base-metal concentrates the percent payment can vary from as little as 50% to more than 95%.

Smelter terms are, therefore, a significant factor in the estimation of potential revenue from any new mining venture.

To illustrate the concepts, an example using the model smelter schedule for copper concentrates (Table 2.23) will be presented. It will be assumed that the copper concentrate contains 30% copper and 30 tr oz of silver per ton. It also contains 2% lead. All other

Table 2.22. The major elements of a smelter contract and the questions addressed (Werner & Janakiraman, 1980).

1. *The parties to the contract.* Who is the seller and who is the buyer?
2. *The material and its quality.* What is the nature of the material; for example, ore or concentrate? What are its assay characteristics and on what basis (wet or dry) are any assays made and reported?
3. *The quantity and duration of the contract.* What are the amounts involved? Are the shipments sent wet or dry? What is the size of a lot? What is the duration of the contract and how are the shipments to be spread out over the period in question?
4. *What is the mode of delivery.* What is the mode of delivery (in bulk or otherwise)? In what manner is the shipment to be made and how is it to be spread over the life of the contract? Is it to be delivered by truck, rail or ship and on what basis; for example, C.I.F. or F.O.B.? Which of the parties to the contract pays the transportation charges?
5. *The price to be paid.* How much of the metal contained in the shipment is to be paid for? Is there any minimum deduction from the reported content of the concentrate? What is the nature of the prices; for example, simple or weighted average? Which particular price or prices are to be used, for example; New York or London Metal Exchange basis, and in what publications are these prices published? What is quotational period involved? How is the price derived averaged over what quotational period?

   In what current is the price to be paid? If it is an average price, what is the basis for the averaging? What forms of weight measurement are to be used: for example, troy, avoirdupois or metric? What premiums, if any, are to be offered? What are the provisions if the quotation ceases to be published or no longer reflects the full value of the metal?
6. *The smelter's charges*. What is the smelter's base charge for treatment? On what grounds might it be increased or decreased, and if so, how might this be done – in relation to the price of metal or to some other change such as a variation in the cost of labour at the smelter? What are the penalties or premiums for excess moisture and for exceeding minimum or maximum limits in content of other elements of mineral constituents?

   What are the associated refining charges, if any?
7. *The quotational period.* What is the quotational period (for example, the calendar month following the month during which the vessel reports to customs at a designated port) and how is this specified?
8. *Settlement.* Whose weights and samples are to be used and what represents a smelter lot?
9. *Mode of payment*. How much is to be paid and when (for example, 80% of the estimated value of each smelter lot shall be paid for at the latest two weeks after the arrival of the last car in each of the lot at buyers' plant)?

   Where, in what form and how are the payments to be made and what are the provisions for the making of the final payment?
10. *Freight allowances*. Which party pays duties, import taxes, etc.?
11. *Insurance*. Which party pays for insurance and to what point? For example, 'alongside wharf'.
12. *Other conditions*. What are the provisions concerning events beyond the control of the buyer or seller? When may 'force majeure' be invoked? What are the conditions under which the contract may be assigned to a third party by the buyer or the seller? What, if any, are the provisions in regard to the presence of specific minor minerals or metals in the concentrate?
13. *Payment conversion rates.* What price quotations and currency conversion rates are to be used and in what publication are they quoted? What are the provisions for the adoption of new bases if those used cease to be published?
14. *Arbitration.* Where are any problems to be arbitrated? How and under what jurisdiction?
15. *Weighing, sampling and moisture determination*. Who does this, where, and who pays for it? Does the seller have the right to be represented at the procedure at his own expense? How are the samples taken to be distributed among buyer, seller and umpire?

Table 2.22. (Continued).

16. *Assaying.* What are the procedures for assaying? What is an acceptable assay for settlement purposes and what are the splitting limits? Which party pays for umpire assaying?
17. *Definitions of terms.* What are the agreed upon definitions of terms? For example: 'Short ton equivalent to 2000 pounds avoirdupois.' Do dollars and cents refer to, for example, Canadian or United States currency?
18. *Insurance*. What insurance coverage is required and which party is to pay for it?
19. *Title.* At what point shall title pass from seller to buyer?
20. *Risk of loss.* At what point shall risk of loss pass from seller to buyer?
21. *Arbitration and jurisdiction*. What are the procedures for settling any disputes that may arise between the parties and which country's law are to govern the parties?

Table 2.23. Model 1989 smelter schedule for copper concentrates (Western Mine Engineering, 1989).

Payments	copper	Pay for 95% to 98% of the copper content at market value, minimum deduction of 1.0 unit per dry ton for copper concentrates grading below 30%. Unit deductions and treatment charges may be higher for concentrates above 30% copper.
	gold	Deduct 0.02 to 0.03 troy ounce per dry ton and pay for 90% to 95% of the remaining gold content at market value.
	silver	Deduct 1.0 troy ounce per dry ton and pay for 95% of the remaining silver content at market value.
Deductions	treatment charge	\$65 to \$100 per dry ton ore or concentrate. Concentrates sold on the spot market are commanding a treatment charge of about \$90 to \$100 per short ton.
	refining charges	\$5.00 to \$6.00 per ounce of accountable gold. \$0.30 to \$0.40 per ounce of accountable silver. \$0.075 to \$0.105 per pound of accountable copper, March, 1989 spot concentrate deals are carrying refining charges of about \$0.11 per pound, long term refining charges are usually \$0.09 to \$0.095 per pound.
	Excessive amounts of some of the following elements may result in rejection	
Deleterious element assessments	lead	Allow 1.0 units free; charge for excess at up to \$10.00 per unit. (No penalty at some plants).
	zinc	Allow 1.0 to 3.0 units free; charge for excess at up to \$10.00 per unit. (No penalty at some plants).
	arsenic	Allow 0.0 units free; charge for excess at \$10.00 per unit.
	antimony	Allow 0.0 units free; charge for excess at \$10.00 per unit.
	bismuth	Allow 0.5 units free; charge for excess at \$20.00 per unit.
	nickel	Allow 0.3 units free; charge for excess at \$10.00 per unit. (No penalty at some plants).
	moisture	Allow 10.0 units free; charge for excess at \$1.00 per unit. (No penalty at some plants).
	alumina	Allow 3.0 units free; charge for excess at \$1.00 per unit.
	other	Possible charges for fluorine, chlorine, magnesium oxide, and mercury.

deleterious elements are below the allowable limits. Assumed prices are \$1/lb for copper, \$6/tr oz for silver and \$0.50/lb for lead. The payments, deductions and assessments are:

Payments

Copper: $C = 98\%$

$f = 1.0$

$u = 1\%$

Silver: $C = 95\%$
$f = 1.0$
$u = 1.0$ tr oz

Deductions
Copper: $T = \$75/\text{ton}$
$r = \$0.10/\text{lb}$
Silver: $r = \$0.35/\text{tr oz}$ of accountable silver

Assessments
Lead: 1 unit is free
additional units charged at $10.00 per unit.

The copper provides the major source of income. Using Equation (2.34) one finds that the basic smelter return is

$$
\begin{aligned}
\text{BSR} &= M_{\text{e}}(Pf - r) - T \\
&= \frac{C}{100}\frac{G-u}{100}2000(Pf - r) - T \\
&= \frac{98}{100}\frac{30-1}{100}2000(\$1 \times 1 - 0.10) - \$75 \\
&= \$436.56/\text{ton of concentrate}
\end{aligned}
$$

The penalty charge for excess lead is

$$
\begin{aligned}
X &= \text{is the (Units present} - \text{Units allowable)} \times \text{Charge/unit} \\
&= (2-1)\$10.00 = \$10.00
\end{aligned}
$$

The by-product credit for silver is

$$
\begin{aligned}
Y &= \left(\frac{C}{100}\right)(G-u)(Pf-r) \\
&= \left(\frac{95}{100}\right)(30.0-1.0)(\$6 \times 1.0 - 0.35) = \$155.66/\text{ton of concentrate}
\end{aligned}
$$

Hence the net smelter return is

$$\text{NSR} = \$436.56 - \$10.00 + \$155.66 = \$582.22/\text{ton of concentrate}$$

It will be assumed that the concentrates are shipped 500 miles by rail to the smelter. The transport cost (F) per ton of concentrate is

$$F = 0.39R_{\text{m}}^{0.7} = 0.39(500)^{0.7} = \$30.22$$

The at-mine-revenue becomes

$$\text{AMR} = \$582.22 - \$30.22 = \$552$$

Table 2.24. Model 2011 smelter schedule for copper concentrates (InfoMine USA, 2012).

Payments	copper	Pay for 95% to 98% of the copper content at market value. Minimum deduction of unit (1 % of a ton) per dry tonne for copper concentrates grading below 30%. Unit deductions and treatment charges may be higher for concentrates above 40% copper.
	gold	Deduct 0.03 to 0.05 troy ounces per dry tonne and pay for 90% to 95% of the remaining gold content at market value.
	silver	Deduct 1.0 troy ounce per dry tonne and pay for 95% of the remaining silver content at market value.
Deductions	treatment/ refining charges	Treatment charges in annual contracts for 2011 varied between \$56.00 to \$90.00 per tonne concentrate with refining charges of \$0.056 to 0.090 per pound copper.
	price participation	Prior to mid-2006, contracts often included price participation clauses. Most contracts settled after late 2006 either eliminated price participation clauses or have capped the price participation to \$0.04–\$0.10 per pound of copper at a basis of \$1.20 per pound copper or above. In 2011 none of the contracts reported a price participation clause.
	complex concentrates	Treatment charges for "complex" concentrates are typically \$15 to \$25 per tonne higher than charges for clean concentrates. In addition, copper concentrates grading over 40% may be charged up to \$10 per tonne more for treatment. This extra charge varies depending on the tightness of the concentrate market. Currently, no smelters are imposing this charge on high grade concentrates.
	refining charges	Range from \$6.00 to \$8.00 per ounce of payable gold and \$0.50 to \$0.75 per ounce of payable silver.
Penalties	deleterious element assessments	Copper concentrates containing excessive amounts of the following elements may be penalized or rejected: lead, zinc, arsenic, antimony, bismuth, nickel, alumina, fluorine, chlorine, magnesium oxide, mercury. Lead, zinc, and arsenic levels above 2% each often result in rejection. For fluxing ores, iron must be less than 3% and the alumina at low levels so that the available silica fluxing content remains high.
	moisture	High moisture content may also be penalized due to material handling difficulties.
Notes:	Spot treatment charges since 2008 have fluctuated widely from treatment charges between \$5 to \$120 per tonne and \$0.005 to \$1.20 per pound refining. Mid-year 2011 spot contracts were reported lower at \$42 to \$44 per tonne.	

The gross value of the metal contained in one ton of concentrates is

$$\begin{aligned} & \quad\;\; \text{(copper)} \qquad \text{(silver)} \qquad \text{(lead)} \\ \mathrm{GV} &= 2000\frac{30}{100}\$1 + 30 \times \$6 + 2000\frac{2}{100}\$0.50 \\ &= \$600 + \$180 + \$20 = \$800 \end{aligned}$$

The percent payment is

$$\mathrm{PP} = 100\frac{552}{800} = 69\%$$

Table 2.24 is an example of a current model smelter schedule for copper concentrates (Info Mine USA, 2012).

2.3.6 *Price-cost relationships*

Using the net smelter return formula it is possible to calculate the revenue per ton of concentrate. The revenue to the mine every year depends upon the tons of concentrate produced and the price. The costs to the mine on the other hand depend upon the amount of material mined and processed.

If one assumes that K tons of concentrate are produced every year, then the yearly revenue depends directly on the price received for the product.

A large capital investment is required at the start of the mining. As will be discussed later this must be recovered from the yearly profits. If the yearly profits are not as expected, then the payments cannot be made. Therefore it is important that the price projections or price forecasts be made covering at least the depreciation period (that period in which the investment is being recovered).

By examining the simplified net smelter return formula,

$$\mathrm{NSR} = \frac{C}{100} 20(G - u) \frac{P - r}{100} - T \tag{2.43}$$

where P is the price, r is the refining and selling cost, T is the treatment cost, G is the percent of metal, u is the fixed unit deduction (%), one can see that the revenue is equal to

$$k(P - r) - T - F$$

where k is a constant.

Assume for definiteness that

$$\begin{aligned} C &= 100\% \\ r &= 12¢ \\ T &= \$60 \\ u &= 1.3\% \\ G &= 28.5\% \\ P &= 90¢ \end{aligned}$$

Thus

$$\begin{aligned} \mathrm{NSR} &= 1 \times 20(28.5 - 1.3)\left(\frac{P - r}{100}\right) - T = \frac{544}{100}(P - r) - T \\ &= 5.44(90 - 12) - 60 = \$364.32 \end{aligned}$$

Assume that next year both the price and the costs increase by 5% but that C, G and u remain constant

$$\begin{aligned} P &= 90 \times 1.05 = 94.5¢ \\ r &= 12 \times 1.05 = 12.6¢ \\ T &= 60 \times 1.05 = \$63 \end{aligned}$$

Hence

$$\mathrm{NSR} = 5.44 \times 81.9 - 63 = \$382.54$$

The net present value would be

$$\mathrm{NPV} = \frac{382.54}{1.05} = \$364.32$$

If the price however *decreased* by 5% and the costs *increased* by 5%, then

$$NSR = 5.44(85.5 - 12.6) - 63 = \$333.58$$

If the price increased by 10% and the costs increased by 5%, then

$$NSR = 5.44(99 - 12.6) - 63 = \$407.02$$

The conclusion is that the net smelter return depends upon the relative changes of the price and the costs. If the prices and costs *escalate* at the same rate then the expected return remains intact. If however, there is a difference then the return may be significantly more or significantly less than expected. Obviously the problem area is if the costs are significantly more than expected or the price significantly less.

2.4 ESTIMATING COSTS

2.4.1 *Types of costs*

There are a number of different types of costs which are incurred in a mining operation (Pfleider & Weaton, 1968). There are also many ways in which they can be reported.

Three cost categories might be:
– Capital cost;
– Operating cost;
– General and administrative cost (G&A).

The capital cost in this case might refer to the investment required for the mine and mill plant. The operating costs would reflect drilling, blasting, etc. costs incurred on a per ton basis. The general and administrative cost might be a yearly charge. The G&A cost could include one or more of the following:
– Area supervision;
– Mine supervision;
– Employee benefits;
– Overtime premium;
– Mine office expense;
– Head office expense;
– Mine surveying;
– Pumping;
– Development drilling;
– Payroll taxes;
– State and local taxes;
– Insurance;
– Assaying;
– Mine plant depreciation.

The capital and G&A costs could be translated into a cost per ton basis just as the operating costs. The cost categories might then become:
– Ownership cost;
– Production cost;
– General and administrative costs.

The operating cost can be reported by the different unit operations:
– Drilling;
– Blasting;

– Loading;
– Hauling;
– Other.

The 'other category' could be broken down to include dozing, grading, road maintenance, dump maintenance, pumping, etc. Some mines include maintenance costs together with the operating costs. Others might include it under G&A. Material cost can be further broken down into components. For blasting this might mean:

– Explosive;
– Caps;
– Primers;
– Downlines.

The operating cost could just as easily be broken down for example into the categories:

– Labor;
– Materials, expenses and power (MEP);
– Other.

At a given operation, the labor expense may include only the direct labor (driller, and driller helper, for example). At another the indirect labor (supervision, repair, etc.) could be included as well.

There are certain costs which are regarded as 'fixed', or independent of the production level. Other costs are 'variable', depending directly on production level. Still other costs are somewhere in between.

Costs can be charged against the ore, against the waste, or against both.

For equipment the ownership cost is often broken down into depreciation and an average annual investment cost. The average annual investment cost may include for example taxes, insurance and interest (the cost of money).

The bottom line is that when discussing, calculating or presenting costs one must be very careful to define what is meant and included (or not included). This section attempts to present a number of ways in which costs of various types might be estimated.

2.4.2 *Costs from actual operations*

Sometimes it is possible to obtain actual costs from 'similar' operations. However great care must be exercised in using such costs since accounting practices vary widely. For many years the *Canadian Mining Journal* has published its 'Reference Manual and Buyers Guide'. A great deal of useful information is contained regarding both mine and mill. Table 2.25 contains information from the 1986 edition for the Similkameen Mine.

Similar information for eleven open pit operations of different types and sizes as extracted from the 1993 edition of the *Reference Manual* (Southam Mining Group, 1992), is included in Table 2.26. Since Similco Mines Ltd. is the successor of Similkameen Property described in Table 2.25, one can examine changes in the operation and in the costs with time. Information as complete as this is seldom publicly available.

The 2004 edition of the CMJ Mining Source book (CMJ, 2004) has included the detailed cost information for the Huckleberry Mine given in Table 2.27. Operating and cost data for some Canadian open pit mines published in the 2010 Mining Sourcebook (CMJ, 2010) are presented in Table 2.28. The authors understand that the 2010 Mining Sourcebook is the last in a long series. This is unfortunate since it has been a very valuable resource. The authors are grateful to the Canadian Mining Journal for permission to include this valuable set of information.

Table 2.25. Cost information (Canadian $) for the Simikameen Mine, Newmont Mining Company Mines Limited (*CMI,* 1986).

1. *Location*: Princeton, British Columbia, Canada

2. *Pit geometry*
   (a) Pit size at surface: 3200′ × 1000′
   (b) Pit depth: 330′
   (c) Bench height: 40′
   (d) Bench face angle: 70°
   (e) Berm width: 40′
   (f) Road grade: 10%

3. *Capacity*
   (a) Mining: ore = 22,000 tpd
   Waste = 25,500 tpd
   ore and waste = 5,111,500 tpy (actual)
   (b) Milling: capacity = 20,000 tpd
   ore = 2,945,000 tpy (actual)
   mill heads = 0.43% Cu
   minerals recovered = Cu, Au, Ag
   recovery = 85.5%
   concentrate grade = 30% Cu
   principal processes: primary SAG, secondary cone crusher, ball milling, Cu flotation

4. *Pit equipment*
   (a) Ore and waste loading = 4 P & H 1900A shovels (10 yd^3)
   (b) Ore and waste haulage = 15 Lectra Haul M100 trucks
   (c) Other = 4 Cat D8K dozers
   1 Cat 14E grader
   3 Cat 824 r.t.d.
   1 Dart 600C f.e.l. (15 yd^3)
   2 Komatsu 705A graders

5. *Blasting in ore and waste*
   (a) Explosives 85% bulk Anfo
   15% packaged slurry
   (b) Powder factor (lb/ton) = 0.46
   (c) Loading factor (lb/yd^3) = 0.70

6. *Drilling in ore and waste*
   (a) Drills = 3 Bucyrus-Erie 60R
   (b) Hole diameter = $9\frac{7}{8}''$
   (c) Pattern (burden × spacing) = 18′ × 24′
   (d) Feet drilled/shift = 490′
   (e) Tons/foot = 26
   (f) Bit life = 6,000 ft
   (g) Rod life = 320,000 ft

7. *Power requirements*
   (a) Total (all motors) = 53,600 HP
   (b) Peak demand = 35,776 kVA
   (c) Annual mill demand = 255,282,172 kWh
   (d) Total annual demand = 266,573,172 kWh

(Continued)

Table 2.25. (Continued).

8. *Personnel*	
(a) Open pit	
Staff personnel	20
Equipment operators, labor	78
Mechanical, maintenance crew	44
Total open pit workforce	142
(b) Mineral processing plant	
Staff personnel	25
Operators (all classifications)	37
Repair and maintenance crew	45
Total mill workforce	107
(c) Surface plant	
Staff personnel	1
Mechanical and maintenance crew	13
Total surface plant workforce	14
(d) Other	
Office and clerical personnel	22
Warehouse	14
Total other	36
Total employees	299
9. *Mining costs for ore and waste ($/ton)*	
(a) Dozing and grading	0.05
(b) Drilling	0.07
(c) Blasting	0.13
(d) Loading	0.14
(e) Hauling	0.21
(f) Crushing	0.11
(g) Conveying	0.06
(h) Pumping	0.01
(i) Maintenance	0.10
(j) Supervision	0.02
(k) Other	0.02
Total	$0.92
10. *Milling costs ($/ton)*	
(a) Crushing	0.096
(b) Grinding	1.844
(c) Flotation	0.244
(d) Drying	0.075
(e) Assaying	0.015
(f) Conveying	0.091
(g) Power	0.921
(h) Tailings disposal	0.137
(i) Labor	0.437
(j) Supervision	0.094
Total	$3.954

Table 2.26. Operating and cost data from some Canadian open pit mines (Southam Mining Group, 1993).

1. Open pit mines included

Company, mine	Location	Minerals recovered	Pit size at surface Depth	Ore mined Waste removed	Bench height Slope	Berm road Grade
BHP Minerals Canada Ltd., Island Copper	Port Hardy, BC	Cu, Mo	7500′ × 4000′ 1300′	57,500 tpd ore 77,500 tpd waste	40′ 45°	25′ 10%
Equity Silver Mines Ltd.	Houston, BC	Ag, Au, Cu	1100 m × 500 m 240 m	10,000 mtpd ore 8000 mtpd waste	5 m 52°	8 m 12%
Hudson Bay Mining & Smelting Co. Ltd., Chisel Lake	Snow Lake, Man	Zn, Cu, Pb, Ag, Au	805′ × 200′ 200′	1315 tpd ore 656 tpd waste	20′ 90°	11′6″ 9%
Iron Ore Co. of Canada, Carol	Labrador City, Nfld	Fe ore	Avg. size, 5 pits: 1500 m × 500 m 90 m	106,000 mtpd ore 18,500 m^3 waste	13.7 m 40° and 60°	15 m 8%
Mines Selbaie, A1 zone	Joutel, Que	Cu, Zn, Ag, Au	1090 m × 875 m 80 m	6000 mtpd ore 19,000 mtpd waste	8–10 m 45–54°	8–10 m 9%
Placer Dome Inc., Dome	South Porcupine, Ont	Au, Ag	100′ × 500′ 130′	350,000 tpy ore 700,000 tpy waste	40°	20′ 10%
Similco Mines Ltd. No. 1	Princeton, BC	Cu, Au, Ag	1500′ × 1200′ 800′	Total 3 pits: 22,500 tpd ore 22,500 tpd waste	40′ 55°	40′ 8%
Similco Mines Ltd. No. 3	Princeton, BC	Cu, Au, Ag	4000′ × 2500′ 1200′	Total 3 pits: 22,500 tpd ore 22,500 tpd waste	40′ 55°	40′ 8%
Similco Mines Ltd., Virginia	Princeton, BC	Cu, Au, Ag	1350′ × 1200′ 440′	Total 3 pits: 22,500 tpd ore 22,500 tpd waste	40′ 55°	40′ 8%
Stratmin Graphite Inc.	Lac-des-Iles, Que	Graphite	650 m × 350 m 25 m	1000 mtpd ore 3500 mtpd waste	6 m 50°	4 m 10%
Williams Operating Corp., C zone	Hemlo, Ont	Rockfill and some Au	450 m × 350 m 36 m	330 mtpd ore 4500 mtpd waste	10 m 70°	8 m 10%

2. Deposit description

Company, mine	Proven and probable reserves	In situ grade	Ore type	Dimensions ($L \times W \times D$)	Host rock
BHP, Island Copper	95 million tonnes	0.355% Cu 0.017% Mo	Porphyry Cu	$5000' \times 2000' \times 1600'$	Andesite
Equity Silver Mines	6 million tonnes	72 g/t Ag 0.22% Cu 0.83 g/t Au	Disseminated and brecciated Sulphides	2.5 km × 500 m × 250 m	Pyroclastic
Hudson Bay, Chisel Lake	439,384 tonnes	0.056 g/t Au 1.23% g/t Ag 0.29% Cu 9.2% Zn 1.09% Pb	Mainly massive sulphides	$750' \times 100' \times 150'$	Altered felsic volcanics
Iron Ore Co., Carol	3 billion tonnes	39% Fe 19% magnetite	Specular hematite and magnetite in sedimentary iron formation	8 km × 250 m × 300 m	Quartzites and quartz-carbonate sandstones
Mines Selbaie	21.2 million tonnes	0.77% Cu 2.19% Zn 28.15 g/t Ag 0.47 g/t Au	Cu, Zn and pyrite lenses	1090 m × 875 m × 176 m	Welded acitic tuff, rhyodacitic breccia, massive pyrite and pyrite breccia
Placer Dome, Dome	9.3 million tonnes	0.143 Au	Hydrothermal quartz vein	$200' \times 1800'$ open at depth	Volcanic porphyry, conglomerate and ultramatic
Similco Mines	Pvn: 24.9 million tonnes Prb: 121.9 million tonnes stockpile: 13 million tonnes	0.45% Cu 0.40% Cu 0.25% Cu	Bornite, pyrite and chalcopyrite		Ore in adesitic volcanics with barren diorite gabbro intrusive
Stratmin Graphite	4 million tonnes	7.4% Cg	Graphite flakes	2700 m × 1700 m × 350 m	Marble and quartzite

(Continued)

Table 2.26. (Continued).

3. Pit equipment

Company, mine	Ore or waste	Shovels	Loaders	Trucks	Other
BHP, Island Copper	Ore and waste	3 P&H 2100BL, 15 yd^3 2 Marion 191M, 15 yd^3		16 Euclid R170 3 Unit Rig Mark 36, 170-t 2 Euclid R190	4 Cat D10N dozers 1 Cat D9L dozer 3 Cat 824 dozers 5 Cat 16G graders 1 Cat 988 loader 1 Hitachi UH20 backhoe
Equity Silver Mines	Ore and waste	3 P&H 1600, 7 yd^3	1 Cat 992C, 10 m^3	2 Wabco, 80-t 5 Cat 777B, 80-t 3 Cat 773, 45-t	2 Cat D8L dozers 2 Cat 14G graders 2 Cat 824 dozers 1 Cat D6K dozer 4 Cat 16G graders 2 Cat 824C dozers 2 Cat 235 backhoes
Hudson Bay, Chisel Lake	Ore and waste	1 Komatsu PC640, 4.6 yd^3	1 Cat 988B, 7 yd^3	3 Euclid, 35-t	1 Cat D8N dozer 1 Champion 780A grader
Iron Ore Co., Carol	Ore and waste	5 B-E 295bII, 14 m^3 2 P&H 2300, 14 m^3 3 Kubota 280, 9 m^3	1 Letourneau L1100, 11.5 m^3 1 Cat 992C, 7.5 m^3 2 Cat 938B, 6 m^3	22 Titan T2200, 180-t 8 Terex 33-15, 154-t	5 Cat 16G graders 4 Komatsu D375A dozers 2 Cat D10N dozers 6 Cat 834B dozers
Mines Selbaie, A1 zone	Ore Waste	 1 Hitachi VH801, 11 yd^3	1 Cat 992C, 11 yd^3 1 Cat 992C, 11 yd^3	3 Cat 777, 77-t 5 Cat 777, 77-t	3 Cat D8L dozers 2 Cat 16G graders 1 Cat 824 dozer 2 water trucks 1 dewatering truck 1 Anfo truck 2 Cat 235 shovels
Similco Mines	Ore and waste	4 P&H 1900A 1 P&H 1900A1	1 Cat 992 1 Terex 7271, 7 yd^3	4 Cat 785, 120-t 9 Unit Rig, 120-t	2 Cat D8 dozers 2 Cat 16G graders 2 Cat 824 dozers 1 Cat D9N dozer
Stratmin Graphite	Ore and waste	1 Cat 235C	2 Cat 980C, 3.8 m^3 1 Cat 988B, 5.5 m^3	4 Cat 769C, 35-t 2 Cat D25C, 25-t	1 Cat 14G grader
Williams Operating Corp.	Ore and waste	1 P&H 1900A1	1 Cat 992	4 Cat 777	1 Cat 14G grader 1 Cat D7 dozer 1 Cat D9N dozer

4. Drilling equipment and practices

Company, mine	Ore or waste	Drills	Hole diameter Pattern	Feet per shift	Tons per foot	Feet per bit	Feet per shank	Feet per rod
BHP, Island Copper	Ore and waste	2 Bucyrus-Erie 60R2 2 Bucyrus-Erie 60RIII	$9\frac{7}{8}''$ $25' \times 25'$	828	44	12,600	654,000	292,900
Equity Silver Mines	Ore and waste	3 Bucyrus-Erie 40R	230 mm 5 m × 5 m	625	22	37,400	215,225	269,030
Hudson Bay, Chisel Lake	Ore	1 GD SCH3500BU 1 with HPR 1H hammer	$4\frac{1}{2}''$ $7' \times 8'$	443	7	69	836	5297
	Waste	1 Copco ROC 812HCSO 2 with 1238ME hammer	$4\frac{1}{2}''$ $8' \times 10'$	280	5.8	423	871	7405
Iron Ore Co., Carol	Ore and waste	4 Bucyrus-Erie 49RH 3 Gardner Denver 120	381 mm Ore: 8 m × 8 m Waste: 8.5 m × 8.5 m	Ore: 260 Waste: 22	Ore: 65.5 Waste: 55.5	1660		
Mines Selbaire, A1 zone	Ore and waste	2 Driltech D40KII 1 Driltech D60KII 1 Copco ROC 712H (secd)	200 mm Ore: 5.1 m × 5.8 m Waste: var	435	Ore: 70 Waste: 77	1090	14,925	13,355
Placer Dome, Dome	Ore		$4''$ $10' \times 10'$		8			
Similco Mines	Ore and waste	3 Bucyrus-Erie 60R	$9\frac{7}{8}''$	540	30	4500		200,000
Stratmin Graphite	Ore and waste	2 Atlas Copco 812HC5001	$5''$ 4 m × 4 m	340	13.6	13,000	3000	6000
Williams Operating Corp.	Waste	1 Gardner Denver 100	$10\frac{5}{8}''$ 5.5 m × 6.5 m	328	23	1804		

(Continued)

Table 2.26. (Continued).

5. Blasting practices

Company, mine	Ore			Waste		
	Explosives	Loading factor	Powder factor	Explosives	Loading factor	Powder factor
BHP, Island Copper	100% emulsion	0.86 lb/yd^3	0.38 lb/ton	Magnafrac	0.86 lb/yd^3	0.38 lb/ton
Equity Silver Mines	60% Anfo 40% slurry	0.53 kg/m^3	0.18 kg/t	60% Anfo 40% slurry		
Hudson Bay, Chisel Lake	Dry holes: Amex Wet holes: Magnafrac	2.4 lb/yd^3	0.53 lb/ton	Dry holes: Amex Wet holes: Magnafrac	1.66 lb/yd^3	0.63 lb/ton
Iron Ore Co., Carol	Magnafrac B9000	1.71 kg/m^3	0.44 kg/t	Magnafrac B9000	1.65 kg/m^3	0.43 kg/t
Mines Selbaie, A1 zone	67% Anfo (column charge) 33% slurry (bottom charge)	0.80–0.86 kg/m^3	0.23–0.32 kg/t	67% Anfo (column charge) 33% slurry (bottom charge)	0.75–0.80 kg/m^3	0.26–0.28 kg/t
Placer Dome, Dome	90% Anfo 10 slurry	1.125 lb/yd^3	0.5 lb/ton	Amex & Detagel	1.125 lb/yd^3	0.5 lb/ton
Similco Mines	95% Fragmax 5% emulsion		0.59 lb/ton	Fragmax NBL 1019		0.59 lb/ton
Stratmin Graphite	90% Anfo 10% slurry	1.35 lb/yd^3	0.06 lb/ton	90% Anfo 10% slurry	1.12 lb/yd^3	0.5 lb/ton
Williams Operating Corp.	95% Anfo 5% packaged			95% Anfo 5% packaged	0.40 kg/m^3	

6. Primary crushing

Company, mine	Crusher site	Transport and distance to crusher	Crusher	Crusher setting	Throughput capacity per hour	Transport and distance to mill
BHP, Island Copper	In pit	170-t trucks, 2000′	A-C gyratory 54″ × 72″	6″	4500 tph	54″ conveyor, 4100′ 54″ conveyor, 800′
	At mill	170-t trucks, 11,200′	A-C gyratory 54″ × 72″	6″	4000 tph	
Equity Silver Mines	At mill	80-t trucks, 1200 m	A-C gyratory 1.1 m × 1.65 m	175 mm	1500 mtph	1.2 m conveyor
Hudson Bay, Chisel Lake	Adjacent to pit	35-t trucks, 2000′	Kue Ken jaw 36″ × 43″	6″	300 tph	Bottom dump rock wagons, 9 miles
Mines Selbaie, A1 zone	Between pit and mill	77-t trucks, 1050 m	A-C gyratory 42″ × 65″	6″	1200 mtph	5 conveyors, 375 m
Placer Dome, Dome	At pit edge	Cat 988 loader, 200′	Piedmont portable jaw 48″ × 60″	6″	250 tph	30″ conveyor, 1000′
Similco Mines		Trucks, 3500′ to 6600′	A-C gyratory 54″ × 72″	8″	120 tph	40″ conveyor, 6600′
Stratmin Graphite	At mill	35-t trucks, 7000 m	915 mm × 1220 mm	150 mm	150 mtph	915 mm conveyor, 150 m
Williams Operating Corp.	Outside pit	4 Cat 777 trucks	A-C gyratory 42″ × 65″	175 mm	100 mtph	1066 mm conveyor (stockpile), 450 m

(Continued)

Table 2.26. (Continued).

7. Mineral processing

Company, mill	Location	Daily throughput	Products	Mill heads	Recovery	Concentrate grade	Principal processes
BHP Minerals Canada Ltd., Island Copper	Port Hardy, BC	57,500 tpd	Cu conc Mo conc	0.39% Cu 0.016% Mo	84% Cu 65% Mo	24.0% Cu 45% Mo	Primary SAG, secondary ball milling, flotation, filtering and drying
Equity Silver Mines Ltd.	Houston, BC	9000 mtpd	Cu-Ag conc Ag-Au dore	0.22% Cu 86 g/t Ag 0.88 g/t Au	69% Cu 62% Ag 59% Au	11.1% Cu 3927 g/t Ag 21.9 g/t Au	Rod and ball milling, flotation, filtering, drying and CIL circuit
Hudson Bay Mining and Smelting Co. Ltd., Flin Flon	Flin Flon, Man	7000 tpd	Cu conc Zn conc	1.8% Cu 6.3% Zn 0.06 g/t Au	94.3% Cu 92.3% Zn 83% Au	21.0% Cu 51.5% Zn	Rod and ball milling, roughing and cleaning for both Cu and Zn conc., thickening and filtering
Mines Selbaie	Joutel, Que	A1 ore: 5800 mtpd	Cu conc Zn conc	0.67% Cu 2.28% Zn	87% Cu 82% Zn	24% Cu 56% Zn	SAG and ball milling, differential flotation and pressure filtering
		A2 & B ore: 1650 mtpd	Cu conc Zn conc	2.93% Cu 0.77% Zn	96% Cu 40% Zn	27% Cu 55% Zn	Rod and ball milling, differential flotation and pressure filtering
Placer Dome Inc., Dome	south Porcupine, Ont	4000 tpd	Au	0.125 g/t Au	95.5%		Crushing, rod and ball milling, jig concentrating, NaCN leaching and CIP
Similco Mines Ltd.	Princeton, BC	25,000 tpd	Cu conc	0.50% Cu	80%	28.5% Cu	SAG, ball milling, Cu flotation, thickening, filtering and drying
Stratmin Graphite Inc.	Lac-des-Iles, Que	1000 mtpd	Natural flake graphite	6.2% Cg	95%	96% Cg	Mechanical processes
Williams Operating Corp.	Hemlo, Ont	6000 mtpd	Au bullion	7.77 g/t	95%	873.7 fine Au	SAG, leaching, CIP, carbon stripping, E/W and refining

8. Personnel numbers and distribution

Company, operation	Underground					Open pit				Mineral processing				Surface plant			Other		
	Staff personnel	Stopping, productions miners	Haulage, hoisting crew	Development, maintenance crew	Total underground workforce	Staff personnel	Equipment operators, labor	Mechanical and maintenance crew	Total open pit workforce	Staff personnel	Operators (all classifications)	Repair and maintenance crew	Total mill workforce	Staff personnel	Mechanical and maintenance crew	Total surface plant workforce	Office and clerical personnel	Others	Total employees
BHP, Island Copper						47	155	135	337	41	60	83	184				34	17[1]	572
Equity Silver Mines Ltd.						12	37		49	15	33		48	8	42	50	12		159
Hudson Bay, Chisel Lake						2	11	2	15										
Iron Ore Co., Carol						80	247	235	562										
Mines Selbaie	12	31	46	43	132	6	58	29	93	15	57	21	93	47	99	146	42	32	538
Placer Dome Inc., Dome	18	54	63	41	176				Note 2	10	17		27	16	70	86	53		342
Similco Mines Ltd.						22	75	42	139	25	43	49	117	1	12	13	25		294
Stratmin Graphite						5	22	6	33	4	21	10	35	3	5	8	12		88
Williams Operating Corp.	25	120	125	50	320	2	13	7	18	19	19	10	48	24	130	124	80		619

[1]Warehouse and safety personnel. [2]Employees of contractor.

(Continued)

Table 2.26. (Continued).

9. Mine operating costs (Canadian $/ton)

Company, mine	Dozing and grading	Drilling	Blasting	Loading	Haulage	Crushing	Conveying	Pumping	Maintenance	Labour	Power	Other	Total
BHP, Island Copper		0.040	0.088	0.345		0.177[1]		0.020	0.044			0.240[2]	0.993
Equity Silver Mines*		0.161	0.161	0.230	0.440			0.033			0.050	0.116	1.189
Hudson Bay, Chisel Lake													
waste		1.640	0.480		0.870[3]	0.050		0.125	1.360		0.840	0.550[4]	5.915
ore		0.950[5]			0.820[3]	0.510		0.125	1.360		0.840	0.550[4]	5.155
Iron Ore Co., Carol													1.887
Mines Selbaie*													
A zone ore	0.137	0.270	0.285	0.090	0.232	0.070			0.955	0.052		0.668[6]	2.759
overburden	0.137			0.090	0.232				0.488			0.097	1.044
waste	0.137	0.270	0.285	0.090	0.232				0.615			0.097	1.726
Placer Dome, Dome													
ore													2.900
waste													3.100
Similco Mines	0.077	0.063	0.118	0.144	0.328	0.090	0.090			0.082		0.015	1.007
Stratmin Graphite*	0.163	0.254	0.308	0.336	0.336			0.082				0.390	1.869
Williams Operating Corp.*	0.073	0.327	0.218	0.308	0.354	0.744					0.136	0.227	2.387

*Amounts reported in metric units have been converted to imperial equivalents. [1]Includes conveying. [2]Includes reclamation $0.055, mine shipping expense $0.096, and other $0.089. [3]Includes loading. [4]Supervision. [5]Includes blasting. [6]Includes engineering and geology.

10. Processing costs (Canadian $/ton)

Company, mill	Crushing	Grinding	Reagents	Flotation	Leaching	Dewatering	Tailings disposal	Assaying	Power	Supervision and labor	Maintenance	Other	Total
BHP Island Copper			0.160						0.740	0.430	0.180	0.694[7]	2.200
Equity Silver Mines*	0.299	1.887	0.064	0.318	0.916	0.191	0.027	0.127				0.490	4.318
Placer Dome, Dome	1.220	1.690			1.340							0.960	5.210
Williams Operating Corp.*		1.660			0.680		0.218	0.308	1.642		1.769	0.889	7.167

*Amounts reported in metric units have been converted to imperial equivalents. [1]Includes grinding balls $0.480, mill liners $0.160, fuel and lubricants $0.008, and operating supplies $0.046.

Table 2.27. Cost information (Canadian $) for the Huckleberry Mine (*CMJ*, 2004).

1. *Location:* Houston, British Columbia, Canada

2. *Pit geometry*
   (a) Pit size at surface: 1050 m × 600 m
   (b) Pit depth: 320 m
   (c) Bench height: 12 m
   (d) Bench face angle: 70°
   (e) Berm width: 10 m
   (f) Road grade: 10%
   (g) Slope angle: 52°

3. *Capacity*
   (a) Mining: ore = 21,000 mtpd
   waste = 44,000 mtpd
   ore and waste = 19,034,000 tonnes (actual)
   (b) Milling: capacity = 20,333 mtpd
   (c) ore = 7,422,000 tonnes (actual)
   mill heads = 0.534% Cu, 0.014% Mo
   minerals recovered = Cu, Mo
   recovery = 88.38% Cu, 47.54% Mo
   concentrate grade = 27.19% Cu, 48.73% Mo
   principal processes: SAG & ball milling, bulk float,
   regrinding & dewatering, Moly float, Cu-Mo
   separation, float and regrinding.

4. *Pit equipment*
   (a) Ore and waste loading = 1 P&H 1900AL shovel
   1 P&H 2100BL shovel
   1 Cat 992C FEL
   1 Cat 416
   (b) Ore and waste haulage = 5 Cat 777C trucks
   4 Cat 785B trucks
   1 Cat 777B truck
   (c) Other = 2 Cat D9N dozers
   2 Cat D8N dozers
   1 Cat D10N dozer
   1 Cat 824 RTD
   2 Cat 16G graders
   1 Hitachi excavator
   1 Cat 769 water truck

5. *Blasting in ore and waste*
   (a) Emulsion
   (b) Powder factor (kg/t) = 0.20
   (c) Loading factor (kg/m^3) = 0.54

6. *Drilling in ore and waste*
   (a) Drills = 2 Bucyrus-Erie 60R
   (b) Hole diameter = 251 mm
   (c) Pattern (burden × spacing) = 7.5 m × 7.5 m to 9.3 m × 9.3 m offset & wall control
   (d) Drilling per shift = 140 m
   (e) Tons/m = 177
   (f) Bit life = 2227 m
   (g) Rod life = NA

(Continued)

Table 2.27. (Contined).

7. *Power requirements*	
(a) Total (all motors) = 190,900,000 kWh	
(b) Peak demand = 30 kVA	
(c) Annual mill demand = 190,500,000 kWh	
(d) Total annual demand = 195,300,000 kWh	
8. *Personnel*	
(a) Open pit	
Staff personnel	18
Equipment operators, labor	68
Mechanical, maintenance crew	34
Total open pit workforce	120
(b) Mineral processing plant	
Staff personnel	12
Operators (all classifications)	33
Repair and maintenance crew	12
Total mill workforce	68
(c) Other	
Office and clerical personnel	14
Others	9
Total other	23
Total employees	211
9. *Mining costs for ore and waste* ($/tonne)	
(a) Dozing and grading	0.158
(b) Drilling	0.053
(c) Blasting	0.111
(d) Loading	0.221
(e) Hauling	0.210
(f) Crushing	incl
(g) Conveying	incl
(h) Pumping	0.024
(i) Maintenance	incl
(j) Supervision and labor	incl
(k) Power	incl
(l) Other	0.135
Total	$1.123
10. *Milling costs* ($/tonne)	
(a) Crushing	0.269
(b) Grinding	1.940
(c) Reagents	0.009
(d) Flotation	0.545
(e) Dewatering	0.122
(f) Tailings disposal	0.056
(g) Assaying	0.047
(h) Conveying	0.091
(i) Power	incl
(j) Supervision & labor	incl
(k) Maintenance	incl
(l) Other	0.359
Total	$3.342

Table 2.28. Operating and cost data from some Canadian open pit mines (CMJ, 2010).

1. Open pit mines included

Company, Mine	Location	Minerals recovered	Pit size (L × W × D)	Ore mined	Waste Removed
ArcelorMittal, Mt-Wright	Fermont, QC	Fe			
Barrick, Williams	Hemlo, ON	Au	1100 × 600 × 230 m	5,000 t/d	18,000 t/d
BHP Diamonds, Ekati	Lac des Gras, NT	Diamonds		7,000 t/d	115,000 t/d
Highland Valley Copper	Logan Lake, BC	Bornite, Chalcopyrite, Molybdenite	Lornex: 2100 × 700 × 540 m Valley: 1400 × 1000 × 700 m	124,300 t/d	40,800 t/d
Huckelberry Mines	Houston, BC	Cu, Au, Ag, Mo	1050 × 600 × 320 m	19,500 t/d	20,100 t/d
Imperial Metals, Mount Polley	Likely, BC	Cu, Au, Ag	Wight pit: 800 × 380 × 200 m Bell pit: 650 × 400 × 170 m Springer pit: 1100 × 800 × 400 m SE pit: 300 × 250 × 100 m C2 pit: 620 × 300 × 130 m	20,000 t/d	76,000 t/d
Inmet, Troilus	Chibougamau, QC	Au, Cu	1100 × 500 m 1300 × 275 m	20,000 t/d	35,000 t/d
Iron Ore, Carol	Labrador City, NL	Hematite & magnetite	910 × 3000 × 200 m	110,000 t/d	11,000 m^3/d
Northgate, Kemess	Smithers, BC	Au, Cu, Ag	1800 × 900 × 390 m	49,700 t/d	52,000 t/d
Thompson Creek, Endako	Endako, BC	Mo	2000 × 760 × 260 m	28,500 t/d	28,500 t/d max
Xstrata Nickel, Spoon	Nunavik, QC	Ni, Cu	200 × 145 × 40 m	750 t/d	2,300 t/d

2. Deposit description

Company, Mine	Proven & probable reserves	In-situ grade	Ore Type
ArcelorMittal, Mt-Wright			
Barrick, Williams	13.2 million tonnes	1.80 g/t Au	Replacement type late hydrothermal mineralization along fracture systems mostly subparallel to foliation
BHP Diamonds, Ekati		1.90 carat/tonne	Kimberlite
Highland Valley Copper	494 million tonnes	0.381% Cu 0.0073% Mo	Lornex: Sulphide in fracture filling Valley: Sulphide
Huckelberry Mines	19.4 million tonnes	0.529% Cu 0.015% Mo 0.059 g/t Au 2.982 g/t Ag	Quartz & sericite veins Porphyry Cu-Mo stockworks & disseminations of Chalcopyrite & molybdenite
Imperial Metals, Mount Polley	55.6 million tonnes	0.357% Cu 0.298 g/t Au 0.661 g/t Ag	Alkalic Cu-Au porphyry
Inmet, Troilus	27.6 million tonnes	0.83 g/t Au 0.08% Cu	
Iron Ore, Carol	5,300 million tonnes	38.6% Fe	Iron formation
Northgate, Kemess	34.2 million tonnes	0.17% Cu 0.41 g/t Au	Porphyry Cu-Au
Thompson Creek, Endako		0.074% Mo	Porphyry style veined quartz molybdenum stockwork
Xstrata Nickel, Spoon	157,250 tonnes	3.16% Ni	

(Continued)

Table 2.28. (Continued).

3. Pit equipment

Company, Mine	Ore, waste or other	Shovels	Loaders	Trucks
ArcelorMittal, Mt-Wright		1 P&H 2300 4 Bucyrus-Erie BII 1 Bucyrus-Erie HR 1 Demag	4 Letourneau L1800	10 Cat 793, 240-t 20 Cat 789,190-t
Barrick, Williams	Ore & waste		2 Cat 992G	6 Cat 777, 80-t
BHP Diamonds, Ekati	Kimberlite & other	1 Cat 531 excavator 3 Komatsu PC 1800 excavator	1 Cat 992D, 12,000 t/d	10 Cat 777D, 90-t
	Waste & other	2 Demag 655SP 3 Komatsu PC 1800 excavator	1 Cat 994C, 21,400 t/d 1 Cat 994C, 12,000 t/d	13 Cat 793C, 218-t 3 Cat 789C, 180-t
Highland Valley Copper	Ore & waste	2 P&H 2800XP/B, 31.3 m^3 2 P&H 2800XP/A, 31.3 m^3 1 Bucyrus-Erie 295,16.8 m^3 2 Bucyrus 495 42 m^3	2 Letourneau L1850, 21.4 m^3 1 Letourneau L1400, 21.4 m^3	17 Cat 789, 172-t 38 Cat 793, 225-t
Huckleberry Mines	Ore & waste	1 P&H 1900AL 2 P8H 2100BL	1 Cat 992C 1 Cat 416	5 Cat 777C 4 Cat 785B 1 Cat 777B
Imperial Metals, Mount Polley	Ore & waste	1 P8H 2100 electric, 13.75 m^3 3 P8H 2100 electric, 12.5 m^3	1 Cat 992C, 9 m^3 1 Hitachi 1100, 3.5 m^3 1 Komatsu 1800, 12 m^3	2 Cat 777B, 86-t 3 Cat 777C, 86-t 2 Cat 777D, 90-t 11 Cat 785C, 140-t 1 Cat 785D, 140-t
Inmet, Troilus	Ore & waste	2 Demag H285 hyd, 16.8 m^3	1 Letourneau L1100, 16.8 m^3	9 Cat 7858,150-t
Iron Ore, Carol	Ore & waste	2 P&H 2800XPB, 32 m^3 4 Bucyrus-Erie 295BII, 18 m^3	1 Letourneau 1800,15.3 m^3 1 Letourneau 1100,11.5 m^3	15 Komatsu 830E, 227-t
Northgate, Kemess	Ore & waste	1 P&H 2800 XPB, 35.1 m^3 1 P&H 2300 XPB, 25.2 m^3 1 Hitachi EX5500 hyd, 26.8 m^3	1 Letourneau L1400, 21.4 m^3	15 Euclid 260, 240-t
Thompson Creek, Endako	Ore & waste	1 P&H 2100XP, 14.5 m^3 1 Bucyrus-Erie 290B, 14.5 m^3 1 P&H 280DD, 33.6 m^3 1 P8H 4100BL, 9.9 m^3	1 Cat 980C 1 Cat 690D 1 Cat 996D 1 Cat 992	9 Euclid R190 2 Unit Rig M120
Xstrata Nickel, Spoon	Ore & waste	1 Hitachi EX450, 3.8 m^3 1 Hitachi EX1100 1 John Deer 60C CL	2 Cat 988, 11.4-t 1 Cat 980, 4- 6 m^3 1 Cat 966, 3.4 - 4.2 m^3	8 Cat 769, 35-t 4 Mac CV713, 30-t

4. Drilling & Blasting

Company, Mine	Blasthole drills	Hole diameter	Hole pattern	Explosives		
				Ore/waste	Type	Loading factor
BHP Diamonds, Ekati	3 Tamrock DK90 diesel 2 Ingersoll-Rand DM45HP 1 Driltech D45S	265 mm 265 mm 265 mm	6.5 × 6.5 m 6.5 × 6.5 m 6.5 × 6.5 m	Kimberlite Waste	Anfo-bulk emulsion Anfo-bulk emulsion	
Highland Valley Copper	4 Bucyrus-Erie 49R1/111	Prod'n: 310 mm Walls. 270 mm		Ore Waste		0.51 kg/t 0.47 kg/t
Huckleberry Mines	2 Bucyrus-Erie 60 R	251 mm	7.5 × 7.5 m to 9.3 × 9.3 m	Ore	Emulsion	048 kg/ m^3
Imperial Metals, Mount Polley	1 Bucyrus-Erie 45R 2 Bucyrus-Erie 60R 1 Driltech C40K DDH 1 Pit Viper 351	250 mm 250 mm 165 mm 310 mm	7.4 × 8.5 m 7. × 8.5 m 6 × 6 m 7.9 × 9.1 m	Ore Waste Ore	SuperAn (same as ore) Apex Gold	0.56 kg/t 0.47 kg/t
Inmet, Troilus	3 Ingersoll-Rand DML60	170 & 190 mm	5.5 × 5.5 m	Ore & waste	Emulsion	
Iron Ore, Carol	1 Gardner Denver 120 4 Bucyrus-Erie 49R	380 mm	7.8 × 7.8 m & 8 × 8 m			
Northgate, Kemess	1 P8H 100 rotary	311 mm		Ore Waste	BLX emulsion & Anfo (same as ore)	0.55 kg/ m^3
	1 Ingersoll-Rand Pit Viper	270 & 310 mm		Ore	BLX heavy Anfo & emulsion	0.55 kg/ m^3
	3 Atlas Copco CM789D 1 Air Trak	115 &130 mm 51 mm		Ore	BLX heavy Anfo & emulsion	
Thompson Creek, Endako	Marion 4 Gardner Denver 100 Gardner Denver 120	315 mm	7.9 × 9.1 m	Ore & waste	Fragmax & Fragmite	0.72 kg/ m^3
Xstrata Nickel, Spoon	(owned by contractor)	140 mm	4 × 4 m	Ore Waste	Emulsion Emulsion	1.17 kg/ m^3 1.13 kg/ m^3

(Continued)

Table 2.28. (Continued).

5. Primary Crushing

Company, Mine	Crusher site	Crusher	Crusher setting	Transport & distance from crusher to mill
ArcelorMittal, Mt-Wright		2 gyratory		
Barrick, Williams	Outside pit	A-C gyratory	170 mm	Crusher to stockpile: Conveyor, 450 m Stockpile to mill: Truck, 450 m
BHP Diamonds, Ekati	At plant	Fuller 4270 gyratory	200 mm	Conveyor, 192 m
Highland Valley Copper	Lornex: Pit rim Valley: In pit	1 A-C gyratory 2 A-C gyratory semi-mobile Krupp carrier	152 mm 152 mm	Trucks, 2500 m Trucks, 850 m
Huckelberry Mines	At mill	Allis-Chalmers gyratory	150 mm	Conveyor, 280 m
Imperial Metals, Mount Polley	At mill	Allis-Chalmers	260 mm	1.5 to 4 km
Inmet, Troilus	At mill	Allis-Chalmers gyratory	150 mm	250 m
Iron Ore, Carol	At mill	2 Allis-Chalmers gyratory	150 mm	1800 m
Northgate, Kemess	At mill	1 Metso gyratory	150–200 mm	Trucks, 3790 m
Thompson Creek, Endako	In pit	Kobello Allis-Chalmers	205 mm	Conveyor, 792 m

6. Mineral Processing

Company, Mine	Daily throughput	Products	Mill heads	Recovery	Concentrate grade	Principal processes
ArcelorMittal, Mt-Wright						Autogenous grinding, spiral gravity concentration & load out
Barrick, Williams	9085 tonnes	Au bullion (84% Au & 10% Ag)	4.03 g/t Au	94.2%		SAG, ball & vertical milling with gravity extraction, thickening, leaching & CIP, carbon stripping, electrowinning & refining
BHP Diamonds, Ekati	12,700 tonnes	Diamonds		100%	1 carat/t	Crushing, scrubbing, screening heavy media separation & x-ray recovery
Highland Valley Copper	116,694 tonnes	Cu conc	0.373% Cu	87.88% Cu	41.6% Cu	3 SAG lines with 2 closed-circuit ball mills each, 2 autogenous lines with 1 closed-circuit ball mill & pebble crusher each, bulk float of Cu-Mo conc, diff Cu-Mo float, vacuum disc filtration & rotary gas-fired drying of Cu conc.
		Mo conc	0.0068% Mo	62.7% Mo	51.8% Mo	Ferric chloride leach of Mo float conc, pressure filtering & drying
Huckelberry Mines	19,822 tonnes	Cu conc	0.555% Cu	88% Cu	27.38% Cu	SAG & ball milling, bulk float, regrinding & dewatering
		Mo conc	0.014% Mo	29% Mo	44.95% Mo	Moly float, Cu-Mo separation, float & regrinding.
Imperial Metals, Mount Polley	20,000 tonnes	Cu-Au-Ag conc	0.57% Cu 0.345 g/t Au 2.543 g/t Ag	≈76% all metals	24.67% Cu 14.39 g/t Au 106.71 g/t Ag	Conventional grinding & bulk flotation

(Continued)

Table 2.28. (Continued).

6. Mineral Processing (Continued)

Company, Mine	Daily throughput	Products	Mill heads	Recovery	Concentrate grade	Principal processes
Inmet, Troilus	17,800 tonnes	Au doré Cu conc	0.85 g/t Au 0.052% Cu	82.4% Au 86.8% Cu	170 g/t Au 16% Cu	Crushing with pre-crush plant. SAG followed by 2 ball mills in series. 4 Knelson gravity machines for the first U/F BB circuit, 1 Flash cell & 1 Falcon concentrator in 2nd U/F BB circuit, plus bulk float.
Iron Ore, Carol	49,200 tonnes (finished product)	Fe conic	39% Fe 12% magnetite	73%	3.5–4.5% SiO_2	Gravity separation & magnetic recovery.
	34,300 tonnes (finished product)	Fe pellets	var SiO_2 to meet prod requirem'ts	98%	(variable)	Conc regrinding, filtering. Balling & induration process.
Northgate, Kemess	52,000 tonnes	Cu-AU bulk conc	0.17% Cu 0.52 g/t Au	84% Cu 69% An	20% Cu 53 g/t Au	Crushing, SAG & ball milling, float & dewatering.
Thompson Creek, Endako	28,500 tonnes	MoS_2	0.112 % MoS_2	78%	90% MoS_2	Mill: Prim. secd & tert crushing, prim milling, rougher-scavenger float, regrind milling, cleaner float & tails.
		MoO_3				Roaster Dewatering, leaching & roasting to MoO_3. Column cell cleaner flotation.
Xstrata Nickel, Spoon	84,000 tonnes	Ni conc	2.5% Ni	86% Ni	17% Ni	Grinding, float, dewatering & drying.
Xstrata Nickel, Strathcona	8600 tonnes	Ni conc	1.4% Ni	82% Ni	9.5–11 5% Ni 2.0–4.5% Cu	2-stage crushing, 2-stage grinding & Cu-Ni separation.
		Cu conc	1.2% Cu	95% Cu	31% Cu 0.5% Ni max	

7. Personnel numbers and distribution

Company, Mine	Underground Employees	Open Pit Employees	Mineral Processing		Engineering	Geology	Environment	Health & safety	Office & clerical	Other	Total workers on site
			Employees	Contractors							
Barrick, Williams	203	53	55					29	147	487	
BHP Diamonds, Ekati	430	645	189		25	6	34	20	81	358	1788
Highland Valley Copper		496	420						132	1048	
Huckelberry Mines		115	71	7	1	1	1		13		209
Imperial Metals, Mount Polley		98	11	4	3	5	4	3	5	28	386
Northgate, Kemess		200	116		8	3	3	7	18	16	371
Thompson Creek, Endako	455	113	101					17	24	243	

(Continued)

Table 2.28. (Continued).

8. Estimated operating costs.

Company, Operation	Minerals recovered	Unit cost	Underground mining cost/t	Open pit mining cost/t	Mineral proces. cost/t	G & A	Total operating cost/t	source	year
Abacus, Afton-Ajax	Cu-Au			$1.50	$3.76	$0.90		43-101	2009
Anaconda Mining, Pine Cove	Au			$5.92	$1.57	$1.47		43-101	2005
Apollo Gold, Black Fox	Au	$387/oz Au	$53.39 (ore)	$2.13 (ore)	$31.17		$75.40 (ore)	FS	2008
Atlantic Gold, Touquoy	Au			$5.52 (ore)	$9.44		$16.37	Eng & cost est	2007
Detour Gold, Detour Lake	Au			$1.75 (ore) $1.50 (waste)	$8.25		$12.00	FS	2008
Fortune Minerals, Sue-Dianne	Cu-Ag			$2.59 (ore & waste)	$14.00			43-101	2008
Getty Cooper, Getty North & South	Cu-Mo			$1.60 ore & waste	$12.94	$0.76/t	$19.47	PFS	2009
Highland Valley, Highland Valley	Cu			$2.82 (ore)	$4.02		$7.30	43-101	2007
Imperial Metals, Mount Polley	Cu-Au			$2.40	$3.61		$4.24	43-101	2004
Inmet, Troilus	Cu-Au						$13.00	Presentation	2008
International Wayside, OR	Au		$17.50–$18.00	$4.85 (ore) $3.37 (waste)	$17.53			PEA	2007
Marathon PGM, Marathon	PGM-Cu			$7.37	$6.02	$1.11	$14.50	FS	2009
Mustang Minerals, Mayville	Ni-Cu			$2.50 (ore)	$10.00	$5.00	$17.50	43-101	2006
Northgate, Kemess	Cu-Au			$2.32	$4.10		$13.18	Presentation	2009
Northgate, Young-Davidson	Au	US$333/oz Au	$20.74 (ore)	$3.09	$10.20	$2.65		PFS	2009

8. Estimated operating costs. (Continued)

Company, Operation	Minerals recovered	Unit cost	Underground mining cost/t	Open pit mining cost/t	Mineral proces. cost/t	G & A	Total operating cost/t	source	year
Osisko, Canadian Malartic	Au	$319/oz Au		$1.52	$3.27	$0.60/t		FS	2008
Rainy River, Rainy River	Au		$50.00	$1.00	$7.00	$1.25/t pit $7.50/t UG		43-101	2009
Shore Gold, Orion South	Diamonds			$1.60	$3.29	$1.65		43-101	2009
Shore Gold, Star	Diamonds			$6.29 ore	$3.29	$1.65/t	$15.88	PFS	2009
Starfield, Ferguson Lake	Ni-Cu		$49.98 (ore)	$2.88 (moved)	$18.89	$7.89	$84.54		2008
Taseko, Gibraltar	Cu-Mo			$3.41 ore	$2.24		$6.15	43-101	2007
Taseko, Prosperity	Cu-Au			$2.27	$3.55		$6.26	FS	2007
Terrane, Mount Milligan	Cu-Au	US$0.17/lb Cu US$51/oz Au LOM		$2.35	$3.89	$0.57	$6.96	FS	2009
Thompson Creek, Endako	Cu-Mo			$0.69 + Var. haulage	$4.45			43-101	2007
Victory Nickel, Minago	Ni	US$3.09/lb Ni	$5.12	$8.86	$9.87	$0.87	$40.96	PEA	2006
Western Copper, Camacks	Cu-Au	US$0.98/lb Ni		$9.98	$7.64		$19.22 (LOM)	FS	2007
Western Copper, Casino	Cu-Au			$3.24	$6.01	$0.47	$9.72	PFS	2008
Western Troy, MacLeod Lake	Cu-Mo	US$470/oz Ni		$7.06	$9.39		$157.90	PEA	2008

Notes:
In section 1, only open pit metal mines are included. Mines producing coal or oil sands are not included.
For sections 2 through 7, only data from the mines listed in section 1 are included.
Section 8, includes all open pit metal mines. Many were not listed in section 1. Some also have underground operations.
The estimated costs in section 8 are not as detailed as those costs included in section 9, of older versions of the Sourcebook.

2.4.3 *Escalation of older costs*

Publications from years past often contain valuable cost information. Is there some simple technique for updating so that these costs could be applied for estimating even today? The answer is a qualified yes. The qualification will be discussed later in this section. The procedure involves the escalation of costs through the application of various published indexes. Table 2.29 is an example of the:

- Construction cost;
- Building cost;
- Skilled labor;
- Common labor;
- Materials.

Table 2.29. ENR cost indices (Engineering News Record (ENR), 2012).

Year	Construction Cost	Building Cost	Skilled Labor	Common Labor	Materials
1976	2499	1425	2136	4700	1055
1977	2577	1545	2264	4977	1159
1978	2776	1674	2405	5303	1289
1979	3003	1819	2564	5676	1427
1980	3237	1941	2767	6168	1488
1981	3535	2097	3025	6802	1527
1982	3825	2234	3358	7545	1548
1983	4066	2384	3591	8020	1651
1984	4146	2417	3721	8269	1621
1985	4182	2425	3778	8396	1617
1986	4295	2483	3867	8616	1634
1987	4406	2541	3986	8869	1659
1988	4519	2598	4085	9120	1694
1989	4615	2634	4174	9381	1693
1990	4732	2702	4310	9646	1720
1991	4835	2751	4457	9935	1709
1992	4985	2834	4580	10243	1761
1993	5210	2996	4703	10525	1953
1994	5408	3111	4818	10856	2068
1995	5471	3112	4943	11146	1993
1996	5620	3203	5085	11444	2046
1997	5826	3364	5229	11697	2226
1998	5920	3391	5374	12024	2179
1999	6059	3456	5537	12383	2184
2000	6221	3539	5740	12790	2195
2001	6343	3574	5965	13242	2113
2002	6538	3623	6208	13871	2044
2003	6694	3693	6496	14386	1981
2004	7115	3984	6747	14978	2296
2005	7446	4205	7035	15555	2476
2006	7751	4369	7274	16164	2595
2007	7966	4485	7604	16756	2580
2008	8310	4691	7902	17415	2730
2009	8570	4769	8197	18190	2674
2010	8799	4883	8483	18776	2684
2011	9070	5058	8712	19257	2826

indices published weekly in the *Engineering News Record* (ENR). The average yearly values are given except where noted. To illustrate the application of the index system, assume that the cost of the mine maintenance building was \$100,000 in June of 1978. The estimated cost of the same building in June of 1989 would be

$$\text{Cost June 1989} = \text{Cost June 1978} \times \frac{\text{Building cost index (June 1989)}}{\text{Building cost index (June 1978)}}$$

In this case

$$\text{Cost June 1989} = 100{,}000 \times \frac{2626}{1664} = 100{,}000 \times 1.58 = \$158{,}000$$

The escalation factor of 1.58 is the ratio of the index values for the years involved. In a similar way one can compute the escalation factors for the other ENR indexes over this period. They are summarized below:

$$\text{Construction cost factor} = \frac{4568}{2754} = 1.67$$

$$\text{Building cost factor} = \frac{2626}{1664} = 1.58$$

$$\text{Skilled labor factor} = \frac{4166}{2376} = 1.75$$

$$\text{Common labor factor} = \frac{9336}{5241} = 1.78$$

$$\text{Materials factor} = \frac{1686}{1229} = 1.37$$

Other indexes are also available. Table 2.30 gives the average hourly earnings for mining production/non supervisory workers as published by the U.S. Bureau of Labor Statistics (BLS) over the period 1964 through 2002. These values can also serve as a labor cost escalator. For the period 1978 to 1989 the factor would be

$$\text{Mining hourly wage factor} = \frac{13.25}{7.67} = 1.73$$

Table 2.31 gives average hourly earnings broken down by industry. Contained with the publication *Statistical Abstract of the United States* are values for the producer price index for construction machinery and equipment. The values for the time period 1978 to 1991 are given in Table 2.32. The resulting factor for the 1978 to 2004 time period is

$$\text{Construction machinery and equipment factor} = \frac{117.2}{67.7} = 1.73$$

Considering the five ENR indices plus the two from the BLS, an average escalation factor of 1.71 is selected. The average cost inflation rate r over this 11 year period is computed by

$$(1 + r)^{11} = 1.71$$

Hence

$$r = 0.050$$

The rate is 5%/year.

Table 2.30. Average hourly wage of production workers broken down by industry using the SIC base (BLS, 2004).

Year	Average hourly earnings ($/hr)							
	Mining*	Metal mining	Iron ores	Copper ores	Coal mining	Bitum. Coal/ lignite	Nonmetallic minerals*	Crushed/ broken stone
1964	$2.81	2.96	3.13	3.04	3.26	3.30	2.48	2.41
1965	2.92	3.06	3.16	3.15	3.45	3.49	2.57	2.47
1966	3.05	3.17	3.28	3.22	3.63	3.66	2.70	2.61
1967	3.19	3.24	3.30	3.26	3.73	3.76	2.85	2.72
1968	3.35	3.42	3.47	3.44	3.83	3.86	3.04	2.93
1969	3.60	3.64	3.70	3.65	4.20	4.24	3.27	3.21
1970	3.85	3.88	3.90	3.93	4.54	4.58	3.47	3.39
1971	4.06	4.12	4.19	4.16	4.78	4.83	3.70	3.64
1972	4.44	4.56	4.60	4.64	5.27	5.31	3.95	3.94
1973	4.75	4.84	4.88	4.90	5.70	5.75	4.22	4.21
1974	5.23	5.44	5.53	5.54	6.22	6.26	4.50	4.49
1975	5.95	6.13	6.29	6.36	7.21	7.24	4.95	4.84
1976	6.46	6.76	7.04	7.04	7.74	7.77	5.36	5.20
1977	6.94	7.28	7.49	7.49	8.25	8.27	5.81	5.67
1978	7.67	8.23	8.48	8.46	9.51	9.55	6.33	6.14
1979	8.49	9.27	9.57	9.53	10.28	10.31	6.90	6.62
1980	9.17	10.26	10.95	10.61	10.86	10.90	7.52	7.16
1981	10.04	11.55	12.16	11.83	11.91	11.95	8.28	7.93
1982	10.77	12.31	12.97	12.53	12.69	12.73	8.90	8.53
1983	11.28	12.58	12.39	13.10	13.73	13.78	9.31	8.70
1984	11.63	13.05	12.74	13.56	14.82	14.87	9.87	9.26
1985	11.98	13.38	13.01	13.62	15.24	15.3	10.18	9.59
1986	12.46	13.19	13.88	12.29	15.40	15.46	10.38	9.80
1987	12.54	12.94	14.36	11.42	15.76	15.81	10.60	10.00
1988	12.80	13.24	14.19	11.62	16.06	16.26	10.94	10.37
1989	13.25	13.58	14.24	11.80	16.26	16.39	11.25	10.69
1990	13.69	14.05	14.59	12.48	16.71	16.85	11.58	11.07
1991	14.21	14.87	16.36	13.36	17.06	17.21	11.93	11.24
1992	14.54	15.26	16.52	13.83	17.15	17.29	12.26	11.55
1993	14.60	15.29	16.67	14.03	17.27	17.46	12.70	12.01
1994	14.88	16.08	17.87	14.31	17.76	17.97	13.11	12.47
1995	15.30	16.77	18.49	14.93	18.45	18.70	13.39	12.66
1996	15.62	17.35	18.70	15.72	18.74	19.03	13.75	13.15
1997	16.15	17.82	18.85	16.32	19.01	19.30	14.19	13.56
1998	16.91	18.24	19.90	16.53	19.17	19.43	14.67	14.03
1999	17.05	18.26	20.43	16.18	19.15	19.33	14.97	14.47
2000	17.22	18.60	21.48	15.65	19.09	19.20	15.28	14.83
2001	17.56	18.74	21.63	15.77	18.94	19.05	15.62	15.02
2002	17.77	18.81	21.93	16.02	19.64	19.77	15.99	15.49

*Except fuels
BLS, 2005. Employment, Hours, and Earnings from the Current Employment Statistics Survey (National) SIC.
Series Reports EEU10000006, EEU10100006, EEU10101006, EEU10102006, EEU10120006, EEU10122006, EEU10140006, EEU10142006
http://www.bls.gov/data/

It was indicated earlier that such escalation has to be done with some care. A major reason for this is the change in labor productivity which has occurred over time.

Productivity is a very important aspect of cost estimation. It deals with the rate at which a certain task can be accomplished. If for example the daily production for a one shift per

Table 2.31. Average hourly earnings of production workers using the NAICS base (BLS, 2012a).

Year	Average hourly earnings ($/hr)				
	Mining*	Coal Mining	Metal Ore Mining	Nonmetallic Mineral Mining and Quarryings	Bituminous coal/Lignite surface mining
1990	15.47	18.42	15.19	11.81	11.82
1991	15.96	18.79	16.15	12.15	12.16
1992	16.11	18.86	16.50	12.50	12.52
1993	16.08	18.96	16.53	12.95	12.97
1994	16.67	19.52	17.40	13.36	13.37
1995	17.15	20.32	18.17	13.64	13.66
1996	17.51	20.70	18.77	14.05	14.07
1997	17.85	20.97	19.29	14.50	14.51
1998	18.08	21.04	19.77	14.98	14.99
1999	18.04	20.90	19.81	15.30	15.31
2000	18.07	20.74	20.07	15.64	15.65
2001	18.22	20.36	19.97	16.09	16.09
2002	18.61	20.57	20.53	16.55	16.56
2003	19.14	20.85	21.91	17.11	17.14
2004	19.85	21.57	22.91	17.74	17.74
2005	20.18	22.05	22.66	18.08	
2006	20.58	22.08	22.40	18.72	
2007	20.77	21.96	23.42	18.81	
2008	22.03	23.27	25.99	19.11	
2009	23.39	26.13	25.95	19.31	
2010	24.63	28.23	27.06		
2011	25.39	28.45			

*Except oil and gas.

Column NAICS Code Description
(col. A) 212 Mining (except Oil and Gas)
(col. B) 2121 Coal Mining
(col. C) 2122 Metal Ore Mining
(col. D) 2123 Nonmetallic Mineral Mining and Quarrying
(col. E) 212111 Bituminous Coal and Lignite Surface Mining

Columns A and B can be retrieved using the BLS series reports CEU1021200008 and CEU1021210008. http://www.bls.gov/data/

Columns C, D, and E could not be retrieved using the series report IDs.

Data for columns C and D were found using the website http://www.econstats.com/blsftp/ceu102121y.htm, and http://www.econstats.com/blsftp/ceu1021230y.htm

day mining operation is 20,000 tons with 100 employees, then one way of expressing the productivity is

$$\text{Productivity} = \frac{20{,}000}{100} = 200\,\text{tons/manshift}$$

Assume that the payroll is $10,000/day or $100/manshift. The labor cost would be $0.50/ton.

Table 2.32. Producer price indexes for construction machinery and equipment (BLS, 2012b).

Year	Index (1982 = 100)	Year	Index (1982 = 100)
1978	67.7	1995	136.7
1979	74.5	1996	139.8
1980	84.2	1997	142.2
1981	93.3	1998	145.2
1982	100.0	1999	147.2
1983	102.3	2000	148.6
1984	103.8	2001	149.1
1985	105.4	2002	151.1
1986	106.7	2003	153.2
1987	108.9	2004	158.5
1988	111.8	2005	168.3
1989	117.2	2006	175.4
1990	121.6	2007	179.6
1991	125.2	2008	185.3
1992	128.7	2009	191.0
1993	132.0	2010	191.4
1994	133.7	2011	197.4

If through some type of change, the daily production could be raised to 30,000 tons, with the same employees, then the productivity would be

$$\text{Productivity} = \frac{30{,}000}{100} = 300\,\text{tons/manshift}$$

and the labor component of the cost would drop to \$0.333/ton. If this productivity has come about through the purchase of new, larger equipment then the decrease in unit labor cost will be accompanied by an increase in other costs (ownership, etc.).

A copper mining example will be used to demonstrate the effect of productivity changes on cost escalation.

In 1909, the use of steam shovels was just beginning in the Utah Copper Company Bingham Canyon Mine of Kennecott (Anonymous, 1909a,b; Finlay, 1908; Jackling, 1909). The following data are available from that time.

1.	Direct ore mining cost	= 15.39 ¢/ton
2.	General mining expense (includes fixed change per ton to retire prepaid stripping)	= 9.83 ¢/ton
	Total mining cost	= 25.22 ¢/ton
3.	Average stripping cost	= 31.43 ¢/yd^3
4.	Direct milling cost	= 47 ¢/ton
5.	General milling expense	= 5.16 ¢/ton
	Total milling cost	= 52.16 ¢/ton

6. Ore grade	= 36 lbs/ton (1.8% Cu)
Milling rate	≅ 7000 tpd
Recovery	≅ 70%
Production rate	= 63,000,000 lbs Cu/year
7. Total average cost (mining, milling, smelting)	= 8.125 ¢/lb Cu
8. Labor wages	= \$2/day
9. Copper price	≅ 12.7 ¢/lb

The index values in 1913 (the closest available year to 1910) were 100. In January 1993 the index values were as follows:

Category	Index	Ratio = $\frac{\text{index 1993}}{\text{index 1913}}$
Skilled labor	4650	47
Common labor	10,395	104
Materials	1806	18
Building cost	2886	29
Construction cost	5070	51

In 1992 the average mining wages were about \$15.00/hour or \$120/day. The labor cost ratio (LCR) of 1992 to 1910 is

$$\text{LCR} = \frac{\$120}{\$2} = 60$$

This is similar to the index values for skilled labor. The copper price ratio (CPR) for the same period is about

$$\text{CPR} = \frac{100}{12.7} = 7.9$$

Due to major changes in productivity over these intervening 80 years, the price and overall cost increase has been much less than would be expected due to labor costs alone. Therefore, when using productivity factors, one must bear in mine the effect of productivity changes over the intervening time.

Table 2.33 gives productivity figures for the mining of iron ore, copper ore, crushed and broken stone and non metallic minerals over the period 1967 through 1990. One can see the major productivity increase which has taken place over the period 1978 through 1990 in iron and copper mining. Table 2.34 provides productivity figures for the period 1987 through 2000. Here a major productivity increase in the gold mining sector is shown. In June 2003, the Standard Industrial Classification system (SIC) was replaced by the North American Industry Classification System (NAICS). Table 2.35 provides the NAICS based labor productivity figures for 1987 through 2010.

It is often very difficult to interpret productivity figures since it makes a big difference as to who has or has not been included.

2.4.4 *The original O'Hara cost estimator*

In 1980 O'Hara (1980) published what has become a classic paper 'Quick guides to the evaluation of orebodies'. He has since produced an updated version which is the subject of

Table 2.33. Productivity (relative output per hour) for mine production workers (BLS, 1992a) (1982 = 100).

Year	Iron mining (crude ore)	Copper mining (crude ore)	Crushed and broken stone	Non-metallic minerals except fuels
1967	85.3	58.7	78.5	87.2
1968	92.7	65.4	85.5	94.8
1969	96.7	73.1	87.3	97.2
1970	99.0	79.7	88.2	100.5
1971	97.7	81.4	86.0	99.7
1972	107.0	86.3	90.8	103.7
1973	112.2	86.4	99.9	108.9
1974	106.9	80.5	97.8	104.8
1975	111.6	81.9	97.2	101.5
1976	112.5	93.2	99.5	107.7
1977	99.1	94.0	106.2	112.0
1978	114.4	103.1	115.1	117.3
1979	121.6	102.5	113.6	115.0
1980	123.6	93.4	107.6	108.1
1981	131.5	95.8	102.8	106.1
1982	100.0	100.0	100.0	100.0
1983	136.8	121.0	109.9	110.1
1984	169.4	130.1	111.6	117.7
1985	182.3	153.9	109.7	120.0
1986	192.2	181.9	110.0	120.8
1987	243.0	179.1	125.7	127.8
1988	260.8	190.4	126.9	130.5
1989	251.2	187.7	123.6	131.8
1990	229.7	182.8	125.6	134.8

Table 2.34. Productivity (relative output per hour) for mine production workers (SIC code) (1987 = 100).

Year	Iron ores	Copper ores	Gold Ores	Crushed and broken stone	Non-metallic minerals*
1987	100	100	100	100	100
1988	103	109.2	99	101.3	101
1989	98.4	106.6	108.9	98.7	99.6
1990	88.5	102.7	119.4	102.2	101.4
1991	85	100.5	118.2	99.8	98.5
1992	83.3	115.2	130.1	105	103
1993	86.9	118.1	144.7	103.6	100.8
1994	85	126	146	108.7	104.4
1995	94.8	117.2	131.9	105.4	104.5
1996	90.7	116.5	128.6	107.2	104.3
1997	89.1	118.9	146.6	112.6	107.3
1998	93	118.3	176.2	110.2	108.6
1999	89.2	110	186.8	105	108.6
2000	103.2	122.6	229.3	101.9	103.3

*Except fuels.
http://ftp.bls.gov/pub/special.requests/opt/dipts/oaehhiin.txt

Table 2.35. Labor productivity output per hour (NAICS Code, 2002 = 100) (BLS, 2012c).

Year	Mining*	Metal Ore Mining	Nonmetallic mineral mining & quarrying
1987	62.3	50.5	84.3
1988	66.5	53.3	85.5
1989	69.0	53.3	87.1
1990	71.4	56.7	89.1
1991	72.1	59.1	86.4
1992	78.2	65.5	92.6
1993	80.8	73.5	89.2
1994	83.9	75.0	93.4
1995	84.6	70.7	93.4
1996	86.5	68.6	93.4
1997	90.2	72.1	96.0
1998	94.3	79.6	97.6
1999	95.6	79.7	98.2
2000	95.3	85.7	92.1
2001	98.5	93.8	96.5
2002	100.0	100.0	100.0
2003	102.8	103.3	104.3
2004	104.9	101.5	109.4
2005	104.3	97.2	115.1
2006	101.1	90.8	116.7
2007	94.4	77.0	103.9
2008	94.9	77.1	105.1
2009	92.2	85.5	97.3
2010	93.3	88.4	97.4

*Except oil and gas.

the next section. However, one of his original curves (Fig. 2.12), which relates mine/mill capital cost C to daily milling rate T_p will be used to demonstrate cost escalation procedures.

The mill generally has a much larger capital cost per daily ton of ore, and hence dominates the curve. It was assumed that the mining operations run only 5 days/week, but that the mill is operated continuously 7 days per week. Thus, the daily ore tonnage mined and crushed T_o will be 40% higher than the milling rate T:

$$T_o = \text{Ore mining rate} = \frac{7}{5}T = 1.4T$$

The combined mine/mill capital cost expressed in mid-1978 Canadian dollars is

$$C = \$400{,}000T^{0.6} \tag{2.44}$$

This must first be converted to U.S. dollars and then escalated to 1989 U.S. dollars. In mid-1978, one Canadian dollar had a value of 0.877 U.S. dollars. The approximate escalation factor from mid-1978 using ENR indices is 1.71. Combining the escalation factor and the exchange rate factor yields an overall multiplying factor of 1.50. Applying this, the expected capital cost in U.S. dollars for mid-1989 is

$$C = \$600{,}000T^{0.6} \tag{2.45}$$

These values are reflected by the right hand axis in Figure 2.12. The interested student is encouraged to escalate these costs to the present time.

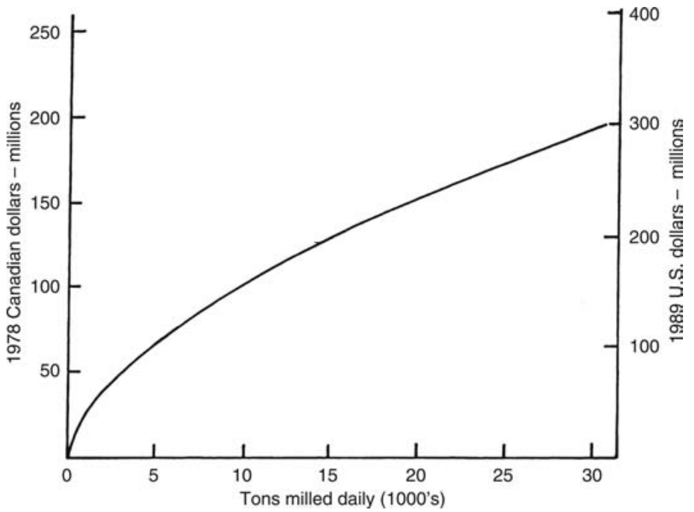


Figure 2.12. Mine/mill project capital cost as a function of milling rate (O'Hara, 1980).

2.4.5 *The updated O'Hara cost estimator*

Introduction

Included in the 2nd edition of the *SME Mining Engineers Handbook* (Hartman, 1992), is an updated chapter on 'Costs & Cost Estimation', prepared by O'Hara and Suboleski (1992). They cover the costs associated with both open pit and underground mining. This section presents material extracted from their paper. The presentation, however, is organized somewhat differently from theirs. All of the costs are expressed in U.S. dollars appropriate for the third quarter of 1988.

Pits may vary greatly in shape, size, and pit slope, especially in mountainous areas or where the ore and/or waste rock varies greatly in competence. The typical open pit mine in North America produces about 43,000 tpd (39 kt/day) of ore and waste from a pit depth of about 400 to 500 ft (120 to 150 m), with an oval shaped periphery 2200 ft (670 m) wide and 4700 ft (1430 m) long. Pit benches are typically 40 ft (12 m) high, and overall pit slope (excluding roads) is about 57° in pits with competent rock, and 44° in pits with oxidized or altered rock, with in-pit haulage road gradients averaging 9%.

The formulas given for equipment sizing, preproduction stripping, and maintenance facilities presume that the shape and type of open pit is similar, except in daily tonnage, to the 'typical' open pit.

Daily tonnage

the most important factor affecting costs is the size of the mine, primary crusher, and processing plant as expressed in terms of the tons of material handled per day of operation. To simplify the discussion the following terms will be introduced:

T = tons of ore milled/day
T_o = tons of ore mined/day
T_w = tons of waste mined/day
T_c = tons of ore passing the primary crusher/day
$T_p = T_o + T_w$ = total material mined/day

In this estimator it is assumed that the mill operates three 8-hour shifts per day and 7 days/week irregardless of the shifts worked by the open pit. Many open pit mines operate 7 days/week, but others may operate only 5. In the case of a 5 day/week mining operation.

$$T = \frac{5}{7}T_o = 0.71T_o \tag{2.46}$$

The cost guides in this section are based upon this assumption that the mill capacity is 71% of the daily mined ore tonnage.

The crushing plant may operate 5, 6, or 7 days/week, depending on the mine schedule and whether or not there is adequate fine ore storage capacity to keep the mill supplied with ore when the crusher is shut down for repairs or regular maintenance.

It is assumed that the crushing plant has the same daily capacity as the mine, but will work 6 days/week to ensure that the mill will be supplied with crushed ore if the fine ore bins have insufficient capacity to keep the mill supplied with ore during the two-day mine shutdown.

Personnel numbers

It may seem somewhat unusual to begin the cost discussion with personnel, but their productivity is extremely important to the profitability of an operation and their compensation is a major cost item.

The number of mine personnel N_{op} required in open pit mines using shovels and trucks for loading and hauling the ore may be estimated from the following formulas:

$$N_{op} = \begin{cases} 0.034T_p^{0.8} & \text{for hard rock} \\ 0.024T_p^{0.8} & \text{for competent soft rock} \end{cases} \tag{2.47}$$

The number personnel N_{ml} required to operate mills treating T tons of low-grade ore may be estimated from the following formulas:

$$N_{ml} = \begin{cases} 5.90T^{0.3} & \text{for cyanidation of precious metal ores} \\ 5.70T^{0.3} & \text{for flotation of low-grade base metal ores} \\ 7.20T^{0.3} & \text{for gravity concentration of iron ores} \end{cases} \tag{2.48}$$

The mill crew size (which includes those involved in crushing and/or grinding as well as beneficiation) as a function of process type and mill rate is shown in Figure 2.13.

The number of service personnel N_{sv} required for open pits mining low grade ore may be estimated as a percentage of the total mine and mill personnel as shown below:

$$N_{sv} = 25.4\% \text{ of } (N_{op} + N_{ml}) \tag{2.49}$$

The number of administrative and technical personnel N_{at} required for a mining and milling plant may be estimated as a percentage of the total required for mining, milling, and services:

$$N_{at} = 11\% \text{ of } (N_{op} + N_{ml} + N_{sv}) \tag{2.50}$$

It should be noted that the formulas do not include the personnel required for smelters, refineries, mine townsite services, concentrate transport, or offsite head offices, since these services may not be required for many mine projects. Whenever these services can be

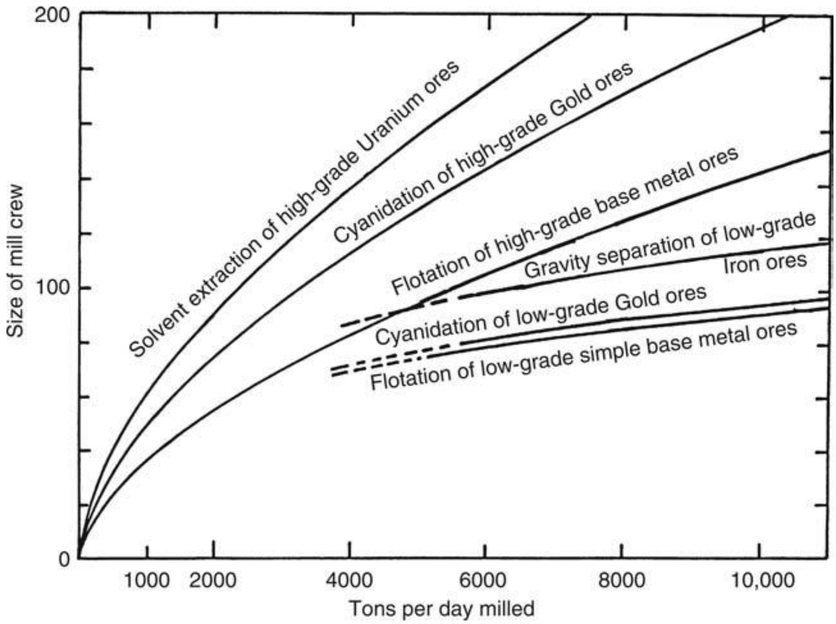


Figure 2.13. Mill crew size versus mill process and size (O'Hara & Suboleski, 1992).

financially justified for the mine project circumstances, the additional personnel should be estimated separately.

Mine associated capital costs

Mine site clearing. Prior to beginning construction, the mine/mill site must be first cleared of trees, plants and topsoil. The soil overburden should be stripped to the limits of the ultimate pit and stockpiled.

The average soil thickness can be found from drilling logs or ultrasonic techniques. By multiplying the average thickness times the pit area, the volume is determined. As an aid to tonnage calculations, an acre of moist soil averaging 10 ft in thickness contains about 23,000 tons of material. For the pit, the required area A_p in acres is

$$A_p = 0.0173T_p^{0.9} \tag{2.51}$$

The clearing costs depend upon the topography, the type of cover, and the total area. They are expressed as

$$\text{Total clearing cost} = \begin{cases} \$1600A_p^{0.9} & \text{for 20\% slopes with light tree growth} \\ \$300A_p^{0.9} & \text{for flat land with shrubs and no trees} \\ \$2000A_p^{0.9} & \text{for 30\% slopes with heavy trees} \end{cases} \tag{2.52}$$

Clearing, initial stripping and access road costs are plotted as a function of T_p in Figure 2.14.

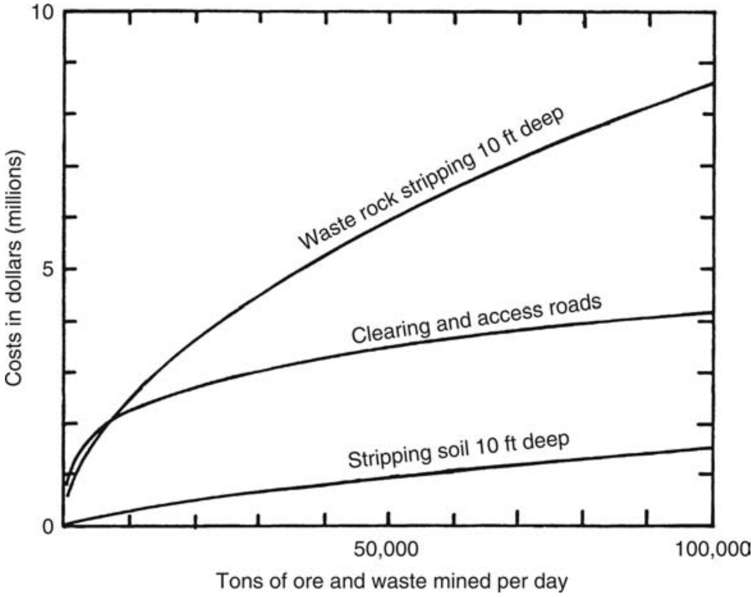


Figure 2.14. Clearing, stripping and road costs for open pit mines (O'Hara & Suboleski, 1992).

Pre-production waste stripping. The rock overburden above the ore must be stripped to expose a sufficient amount of ore to supply the planned daily ore tonnage for a period of four to six months. If insufficient ore has been exposed by the pre-production stripping of waste, it may become difficult to continue ore mining due to the close proximity of waste benches where blasting, loading, and haulage of waste is taking place.

The location and required area of the ore exposure is determined from ore body mapping. Once this has been done, the average thickness and area of the waste rock overlying this ore can be computed. Each acre of waste rock averaging 10 ft in thickness contains about 40,000 tons of waste.

Because of the inverted conical shape of the ultimate open pit, the waste/ore tonnage ratio at each horizontal bench decreases with each lower bench. Typically, the uppermost ore bench to be exposed has a waste/ore ratio of at least twice the waste/ore ratio of the ultimate pit. If T_s is the tons of soil, and T_{ws} is the tons of waste rock that must be stripped to expose an amount of ore to sustain four to six months ore production, then the estimated costs of waste stripping will be

$$\text{Soil stripping costs} = \$3.20T_s^{0.8} \text{ for soil not more than 20 ft deep} \tag{2.53a}$$

$$\text{Waste stripping costs} = \$340T_{ws}^{0.6} \text{ for rock requiring blasting, loading, and haulage} \tag{2.53b}$$

Mine equipment

(a) Drills. The size, hole diameter, and number of drills required depends on the tons of ore and waste to be drilled off daily.

Typically, drill hole sizes have standard diameters of 4, 5, 6(1/2), 7(7/8), 9, 9(7/8), 10(5/8), 12(1/4), 13(3/3), 15, and 17(1/2) inches (or 102, 125, 165, 200, 229, 250, 270, 310, 350, 380, and 445 mm). Thus drill selection will be limited to one of these sizes.

The tons of ore or waste that are drilled off per day by a drill with a hole diameter of d inches is:

$$\begin{aligned} \text{tons of medium drillable rock} &= 170d^2 \\ \text{tons of easily drillable rock} &= 230d^2 \\ \text{tons of hard drillable rock} &= 100d^2 \end{aligned} \tag{2.54}$$

For the rock defined as 'medium' drillable, the expected production rate is about 500 ft per shift.

The number of drills N_d should never be less than two. For tonnages up to 25,000 tpd, two drills of appropriate hole diameter should be chosen. Three drills should be adequate for up to 60,000 tpd and four or more drills will be required for daily tonnages over 60,000.

The cost of the drilling equipment is given by:

$$\text{Drilling equipment costs} = N_d \times \$20{,}000d^{1.8} \tag{2.55}$$

This formula includes a 25% allowance for drilling and blasting supplies and accessory equipment.

(b) Shovels. The optimum shovel size S expressed in cubic yards of nominal dipper capacity in relation to daily tonnage of ore and waste T_p to be loaded daily is

$$S = 0.145T_p^{0.4} \tag{2.56}$$

The number of shovels N_s with dipper size S that will be required to load a total of T_p tons of ore and waste daily will be

$$N_s = 0.011\frac{T_p^{0.8}}{S} \tag{2.57}$$

In practice, the size of shovel chosen will be one with a standard dipper size close to the size calculated by Equation (2.56). The calculated number of shovels N_s usually is not a whole number. It should be rounded down. The omitted fractional number expresses the need for either a smaller-sized shovel or a front-end loader for supplemental loading service. This smaller shovel or front end loader must, of course, be capable of loading trucks of a size appropriate to the shovels with dipper size S.

The total costs of the fleet of shovels supplemented by auxiliary bulldozers and front end loaders will be

$$\text{Loading equipment cost} = N_s \times \$510{,}000S^{0.8} \tag{2.58}$$

(c) Trucks. The optimum truck size t in tons that is well matched with shovels of bucket size S (cubic yards) is

$$\text{Truck size } t \text{ (tons)} = 9.0S^{1.1} \tag{2.59}$$

The total number of trucks N_t of t tons capacity required for the open pit truck fleet, plus an allowance for trucks under repair, is approximated by the following formula:

$$N_t \text{ (Number of trucks required)} = 0.25\frac{T_p^{0.8}}{t} \tag{2.60}$$

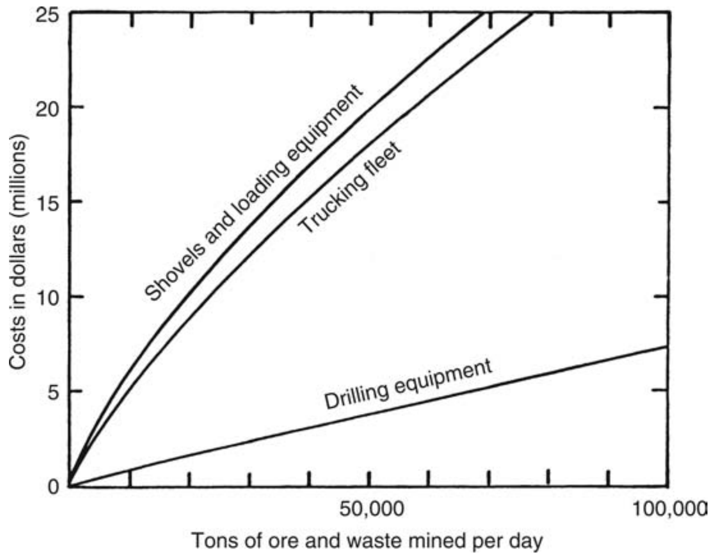


Figure 2.15. Costs for open pit equipment (O'Hara & Suboleski, 1992).

The formula for N_t determines the size of the truck fleet under the typical conditions where the average haulage distance and gradient outside the pit periphery is less than the haulage distance and gradient inside the pit periphery. If the waste dump and the ore dump by the primary crusher are well removed from the pit boundaries, or if the haulage road beyond the pit has a steep gradient, it may be necessary to increase the truck fleet size to allow for the longer trip time per load.

The cost of haulage equipment including the accessory road maintenance equipment is given by:

$$\text{Haulage equipment cost} = N_t \times \$20{,}000t^{0.9} \tag{2.61}$$

The capital costs for the production fleet are given in Figure 2.15.

Pit services

(a) Maintenance facilities. The size of maintenance facilities for repair and maintenance of open pit equipment depends primarily on the number and size of the mine haulage trucks, which in turn depends on the daily tonnage of ore and waste to be hauled. Repair and maintenance of the shovels and drills is normally performed on site by mobile repair vehicles.

The area in square feet required by the open pit maintenance shop (which should be located close to the open pit) is as follows:

$$\text{Area of open pit repair shot} = 360T_p^{0.4} \tag{2.62}$$

Thus the areas of repair shops required for open pit mines are:

Mine size, tpd	10,000	20,000	40,000	80,000
Repair shop area, ft^2	14,300	18,900	25,000	33,000

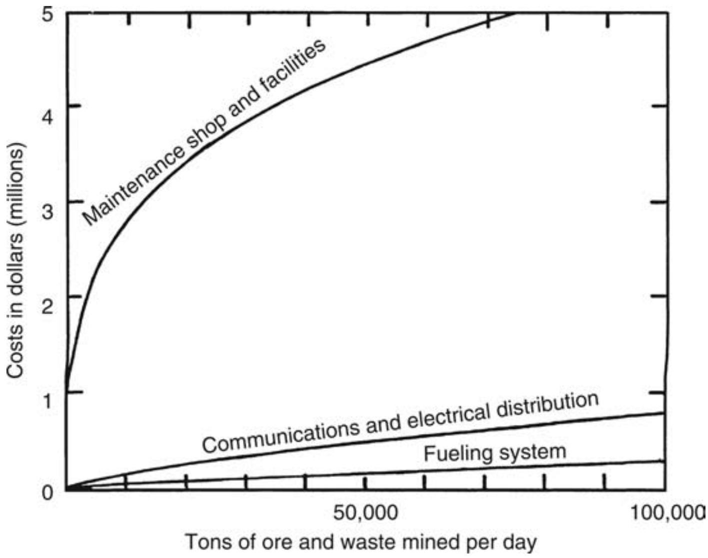


Figure 2.16. Costs for open pit services (O'Hara & Suboleski, 1992).

The cost of constructing and equipping the shop is expressed by

$$\text{Cost of pit maintenance facilities} = \$6000\,A^{0.6}t^{0.1} \tag{2.63}$$

(b) Communication and electrical distribution. This cost includes the installed costs for a surface telephone system with mobile and base radio units and one or more repeaters depending on the size of the mine. The electrical distribution includes the installed costs of primary substations, transmission lines, portable skid-mount transformers, and trailing cables, all of which depend on the size of the open pit mine as measured by the daily tons T_p of ore and waste mined.

$$\text{Cost of communications/electrical} = \$250\,T_p^{0.7} \tag{2.64}$$

(c) Fueling system. This cost includes the storage and services for diesel fuel, gasoline, lubricants, and coolants for the truck haulage fleet and mobile service vehicles

$$\text{Cost of refueling system} = \$28\,T_p^{0.7} \tag{2.65}$$

The open pit services costs are shown in Figure 2.16.

Mill associated capital costs

Mill site clearing and foundation preparation costs. The area A_c (in acres) to be cleared for the concentrator building, crusher building, substation, warehouse, and ancillary buildings is given by

$$A_c = 0.05T^{0.5} \tag{2.66}$$

In addition to this clearing, roads must be constructed from the nearest existing suitable road to provide access to the concentrator site, the hoisting plant, the proposed tailings basin, and the source of the water supply. Costs for clearing and access roads for the surface plant are estimated to be:

$$\text{Clearing costs} = \$2000\,A_c^{0.9} \text{ for lightly treed area with slopes of not more than 20\% gradient} \quad (2.67a)$$

$$\text{Access roads} = \$280{,}000 \text{ per mile for 30-ft (9-m) wide graveled road in mildly hilly region} \quad (2.67b)$$

The formulas should be modified ±30% for more adverse or more favorable slope and tree growth conditions.

Soil overburden must be stripped wherever buildings and facilities are to be sited. The cost of stripping soil overburden D_o feet deep over and area of A acres will be:

$$\text{Cost of soil stripping} = \$1000\,A^{0.8}D_o \quad (2.68)$$

After the soil overburden is removed and the underlying rock or basal strata is exposed, this rock or strata will require localized removal, probably by drilling and blasting, to establish sound foundation conditions over levelled areas for the plant buildings and plant equipment. If there are C_u cubic yards of rock requiring drilling, blasting, and haulage to a dump site, this mass excavation will cost:

$$\text{Cost of mass excavation} = \$200\,C_u^{0.7} \quad (2.69)$$

for excavations of up to 100,000 yd^3.

If the mass excavation is in rock that can be broken by ripping, the cost will be only 20% of that indicated.

When the mass excavation has been completed, detailed excavation to tailor the rock surface to the exact levels for pouring concrete foundations can be done. At the same time, suitable fill will be placed and compacted over level areas where deep trenches of soft soil have been removed. If there are C_d cubic yards of rock to be excavated by detailed excavation and F_c cubic yards of compacted fill to be placed, the cost will be:

$$\text{Excavated and fill compaction} = \$850\,C_d^{0.6} + \$75\,F_c^{0.7} \quad (2.70)$$

Concrete costs for the foundations of the concentrator building, fine ore bins, and concentrator equipment probably will cost between \$350 and \$900/yd^3, depending on whether the concrete pour is for a simple form with little reinforcing steel or for a complex form that is heavily reinforced. The concrete cost may be significantly higher per cubic yard if concrete is scheduled to be poured in winter months when the temperature is below 40°F (4.4°C) and heating of aggregate and water and heating of concrete forms is required for sound concrete.

It is difficult to estimate the shape and volume of concrete forms before these forms have been designed, and hence concrete costs related to concrete volume are unreliable for preliminary estimation. Assuming no difficulties

$$\text{Approximate concrete foundation costs} = \$30{,}000\,T^{0.5} \quad (2.71)$$

These different costs are shown in Figure 2.17 as a function of daily plant capacity.

Concentrator building. The costs of the concentrator building include all costs of constructing the building above the concrete foundations and enclosing the building, plus the cost of internal offices, laboratories, and changerooms. It does not include the cost of process

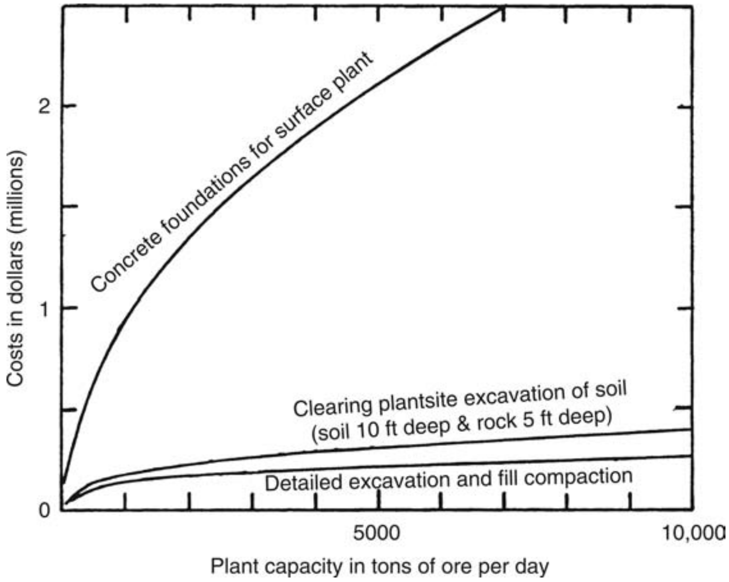


Figure 2.17. Costs for surface plant clearing, excavation and foundations (O'Hara & Suboleski, 1992).

equipment, piping, or electrical wiring, because these items are included in the costs of each functional area. The equipment in operating concentrators generates a substantial amount of heat and comfortable working conditions can be attained with little or no insulation, as long as the concentrator is located in a region with a mild climate. For flotation mills located in a mild climate

$$\text{Cost of building} = \$27{,}000\, T^{0.6} \tag{2.72}$$

A 'mild climate' is defined as a region where the degree-days are about 7000 (in °F) or 4000 (in °C) per year. Weather stations usually record the 'degree-days' (°F × D, or °C × D), which represents the average number of days times the degrees that the temperature is below 65°F or 18°C. In hot climates, where freezing temperatures are not experienced, the building costs may be reduced by only partially enclosing the building and by locating thickeners and other hydrometallurgical equipment outside the building. In cold climates, the additional cost of insulation, heating, and snow loading is likely to increase the building cost by about 10% for each increase of 1800 (°F × D) above 7000 or 1000 (°C × D) above 4000.

Primary crushing plant with gyratory crusher. Open pit mines generally place the primary crusher on the surface outside the pit, within convenient conveying distance to the coarse or stockpile and the fine ore crushing plant. Open pit trucks normally dump the ore onto a grizzly mounted over the gyratory crusher which discharges crushed ore to a conveyor. Because of the headroom required to operate and discharge the crushed ore from a gyratory crusher, a substantial excavation and volume of concrete is required for the primary crusher

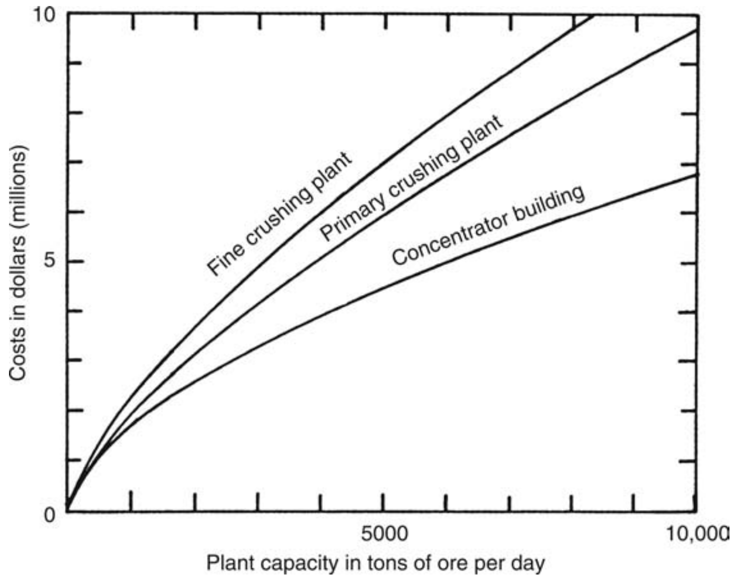


Figure 2.18. Costs for the concentrator building and crushing plant (O'Hara & Suboleski, 1992).

plant. The cost of the primary crusher depends on the size and capacity of the gyratory crusher selected for crushing T_c tons of ore daily:

$$\text{Cost of gyratory crusher} = \$63\, T_c^{0.9} \tag{2.73}$$

The cost of excavating and concreting the foundations for the primary crusher, installing the crusher, construction of the truck dump and grizzly, plus the coarse ore conveyor and feeder under the crusher is:

$$\text{Cost of primary crushing plant} = \$15{,}000\, T_c^{0.7} \tag{2.74}$$

The cost of the crusher itself is not included.

Fine ore crushing and conveyors. This cost includes the crushing plant building, installed equipment and conveyors.

$$\text{Cost of fine ore crushing plant} = \$18{,}000\, T^{0.7} \tag{2.75}$$

Note: The cost may be 12% higher if the conveyors must be enclosed and heated.

Grinding section and fine ore storage. The fine storage bins must have sufficient live capacity to provide mill feed for at least the number of days that the crushing plant is idle per week. The cost of the fine ore bins will be proportional to the weight of steel used in constructing these bins, and the weight of steel will be proportional to $T^{0.7}$.

The size and cost of the grinding mills depend on the tons of ore to be ground daily by each mill, but they also depend on the hardness of the ore as measured by the work index

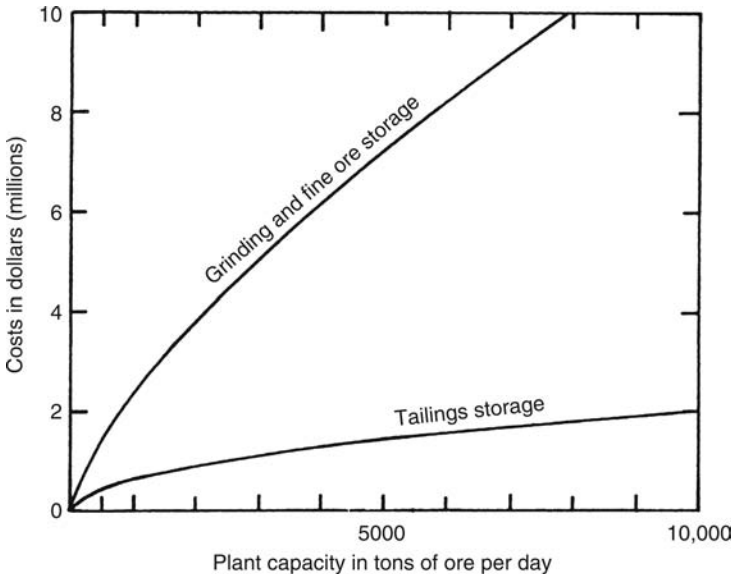


Figure 2.19. Costs for the grinding section, storage bins and tailings storage (O'Hara & Suboleski, 1992).

and the fineness of grind that is required to attain the desired concentration and recovery of valuable minerals.

$$\text{Cost of grinding and bins} = \begin{cases} \$18{,}700\,T^{0.7} \text{ for medium hard ore} \\ \text{with a work index of 15, ground} \\ \text{to 70\% passing 200 mesh} \\ \$12{,}500\,T^{0.7} \text{ for soft ores ground} \\ \text{to 55\% passing 200 mesh} \\ \$22{,}500\,T^{0.7} \text{ for hard ores with a} \\ \text{work index of higher than 17, ground} \\ \text{to 85\% passing 200 mesh} \end{cases} \tag{2.76}$$

These costs are plotted in Figure 2.19 as a function of plant capacity.

Processing and related sections. The capital costs in this section cover the purchase and installation of all equipment required to concentrate or extract valuable minerals from the slurried ground ore, and process the concentrates or extracted minerals into dried solids or impure metals that are directly salable as dry concentrates, ingots of precious metals, uranium yellowcake, or impure metallic gravity concentrates of alloy metals. These capital costs include equipment and tanks for thickening, filtering, precipitation, leaching, solvent extraction, etc., plus all process piping, electrical wiring, and process control.

Process costs for different types of ore by different methods are listed below:

1. High-grade gold ores leached by cyanidation, followed by zinc dust precipitation of gold by Merrill Crowe process, filtering, drying, and gold refining:

$$\text{Process capital costs} = \$60{,}200\,T^{0.5} \tag{2.77}$$

2. Low-grade ores, cyanide leaching, CIP (carbon-in-pulp) or CIL (carbon-in-leach) adsorption, refining:

$$\text{Process capital costs} = \$47{,}300\, T^{0.5} \tag{2.78}$$

3. High-grade gold ores with base metal sulfides; cyanide leaching, secondary flotation, carbon adsorption by CIP or CIL process, filtering, thickening, drying, and refining:

$$\text{Process capital costs} = \$103{,}200\, T^{0.5} \tag{2.79}$$

4. Simple low-grade base metal ores of copper with minor content of gold, which can be recovered as smelter credits. Flotation, thickening, filtering, and drying of auriferous copper concentrates:

$$\text{Process capital costs} = \$13{,}700\, T^{0.6} \tag{2.80}$$

5. Pyritic gold/silver ores where the precious metals are locked in the pyritic minerals. Differential flotation, selective roasting, recovery of deleterious materials, cyanidation, thickening, precipitation, filtering, and refining.

$$\text{Process capital costs} = \$180{,}000\, T^{0.5} \tag{2.81}$$

6. High-grade Cu/Pb ores, Cu/Zn ores, Pb/Zn ores, Cu/Ni ores. Recovery by differential flotation, thickening, filtering, and drying of separate concentrates:

$$\text{Process capital costs} = \$20{,}600\, T^{0.6} \tag{2.82}$$

7. Complex base metal ores containing at least three valuable metals, with recoverable minor amounts of precious metals; Cu/Zn/Pb ores, Pb/Zn/Ag ores, Cu/Pb/Ag ores, Cu/Zn/Au ores. Recovery by differential flotation, separate thickening, filtering, and drying of several concentrates and/or bulk concentrates.

$$\text{Process capital costs} = \$30{,}100\, T^{0.6} \tag{2.83}$$

8. Non-sulfide ores containing specialty metals such as columbium (niobium), tantalum, tungsten, and tin in minerals that do not respond to flotation, and which are separated by specialized gravity concentration methods:

$$\text{Process capital costs} = \$5000\, T^{0.7} \text{ to } \$13{,}000\, T^{0.7} \tag{2.84}$$

9. Uranium ores: acid leaching, countercurrent decantation, clarification, solvent extraction and yellowcake precipitation:

$$\text{Process capital costs} = \$150{,}000\, T^{0.5} \text{ to } \$200{,}000\, T^{0.5} \tag{2.85}$$

Figure 2.20 is a plot of these relationships.

Initial tailings storage. There are many aspects of tailings storage such as topography, distance from mill to tailings site, localized environmental concerns, etc., that could drastically alter the costs of tailings storage. If, however, all adverse aspects are absent, and a suitable tailings site is available within two miles of the mill, and the nature of the tailings does not have adverse environmental effects, the minimum cost of tailings storage may be:

$$\text{Minimum tailings storage cost} = \$20{,}000\, T^{0.5} \tag{2.86}$$

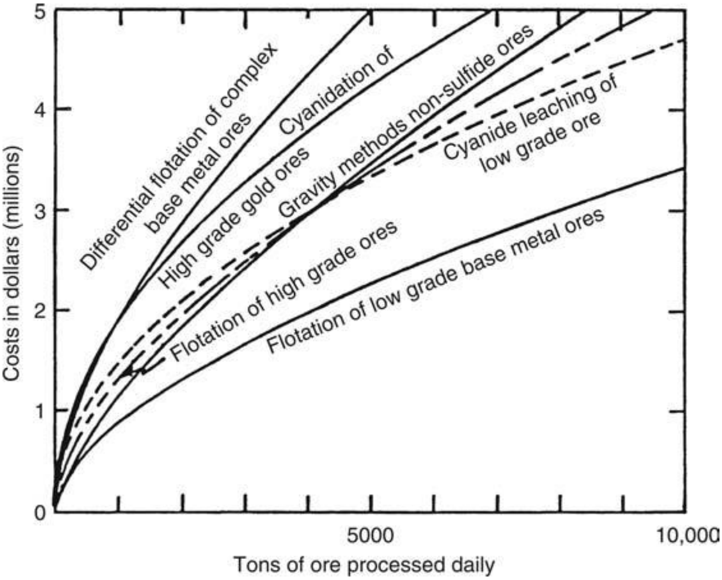


Figure 2.20. Processing section costs (O'Hara & Suboleski, 1992).

Very few mines have such favorable conditions, and if the area topography is steep or the environmental constraints are stringent, the tailings storage costs could be several times as high as the foregoing cost guide.

General plant capital cost
Water supply system. The cost of fresh water pumping plants, reclaim water plants, and provision for fire protection water supply, plus potable water supply, varies according the local topography and the proximity and nature of nearby sources of year-round supplies of water. If there is a suitable source of water within two miles of the mill, and the intervening topography is moderately level, the water supply system would cost:

$$\text{Cost of water supply system} = \$14{,}000\, T^{0.6} \tag{2.87}$$

The cost of the water supply system for the mine, mill, and plant (but excluding the mine water distribution system) will be much higher if the local topography is steep and rugged or if there are severe constraints on sources of fresh water.

Electrical substation and surface electrical distribution. The capital cost of electrical facilities for a mining/milling plant depends primarily on the size of the electrical peak load in kilowatts.

The peak load (PL) expressed in kilowatts per month and the average daily power consumption in kilowatt hours can be estimated from the following formulas:

$$\text{Peak load (PL)} = 78\, T^{0.6} \text{ for open pit mines milling } T \text{ tons of ore daily} \tag{2.88}$$

$$\text{Power Consumed} = 1400\, T^{0.6} \text{ for open pit mines with shovel and truck haulage to concentrator} \tag{2.89}$$

Typically, the concentrator and related facilities account for about 85% of the total power consumption for open pit mines and concentrators.

The cost of power supply depends on whether the power is generated by an existing electric utility or by a mine diesel-electric plant. Small mines in remote areas may be forced to generate their own electric power, because the cost of a lengthy transmission line from an existing utility may be too high due to the low peak load and low electric power consumption of a small mine.

If the mine is supplied with utility power, the cost of a utility substation with step-down transformers will be

$$\text{Cost of substation} = \$580(\text{PL})^{0.8} \tag{2.90}$$

The cost of installing low-voltage power distribution to the surface concentrator, crushing plant, and surface facilities, but excluding the distribution to the surface open pit is likely to be

$$\text{Cost of surface power distribution} = \$1150(\text{PL})^{0.8} \tag{2.91}$$

A diesel-electric generating plant may be required for a small mine in a remote area or by a larger mine supplied with utility power that may require a standby electric power plant for protection of vital equipment.

$$\text{Cost of diesel-electric plant} = \$6000(\text{PL})^{0.8} \tag{2.92}$$

General plant services. These costs include the costs of constructing, furnishing, and equipping the general administrative office, general warehouse, electrical and mechanical repair shop (for smaller mill equipment and services equipment), vehicle garages, changehouses, first aid and mine rescue stations, security stations plus general purpose vehicles, parking lots, and yard fencing.

The size of the buildings tends to depend on the number of employees served by each building. It is necessary to estimate the building size in square feet before estimating building cost, which will vary with the area of each type of building.

(a) Administrative office. The floor space per person tends to increase as the number of administrative and technical staff N_{at} becomes larger. This reflects the more complex records of accounting and technical staff and the consequent requirement of more space for computer facilities, mining plans, and reference file facilities.

$$\begin{aligned} A = \text{Office are required in ft}^2 &= 35\,N_{at}^{1.3} \\ \text{Cost of office} &= \$155\,A^{0.9} \end{aligned} \tag{2.93}$$

(b) Maintenance shop. Maintenance personnel N_{sv} will require about 85 ft^2/person for maintenance and repair of movable equipment from the mill and service departments.

$$\text{Cost of shop} = \$102(85\,N_{sv})^{0.9} \tag{2.94}$$

(c) Mine changehouse. The mine changehouse requires about 24 ft^2/person on the mine payroll and includes the first aid station and mine rescue facilities.

$$\text{Changehouse cost} = \$125(24\,N_{op})^{0.9} \tag{2.95}$$

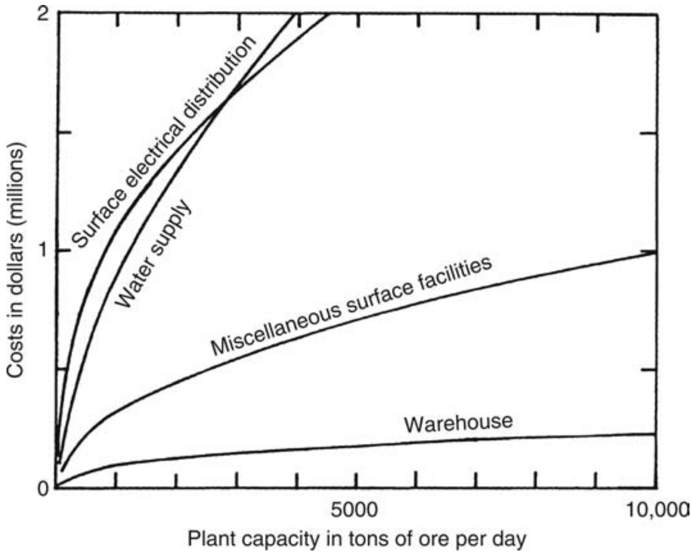


Figure 2.21. Costs of plant service facilities (O'Hara & Suboleski, 1992).

(d) Surface warehouse. This should accommodate all supplies and spare parts for the mine, mill, and service facilities that must be kept indoors. Bulky supplies such as rough lumber, structural steel, etc., can be stored outdoors in most climates.

$$\text{Surface warehouse cost} = \$5{,}750\,T^{0.4} \tag{2.96}$$

(e) Miscellaneous surface facilities. This includes general purpose vehicles and garages, security stations and fencing, parking lots, and miscellaneous services.

$$\text{Miscellaneous surface facilities} = \$10{,}000\,T^{0.5} \tag{2.97}$$

Those general plant capital costs dependent on plant capacity are shown in Figure 2.21.

Mine project overhead costs

In addition to the direct costs for specific facilities for a mine project, which may total many millions of dollars, there are substantial costs and expenses involved in project design, general site costs, supervision and administration, and provision of working capital. These overhead costs may be estimated as a function of the total direct costs D in dollars.

Engineering. This includes the costs of feasibility studies, environmental impact studies, design engineering, equipment specifications and procurement, and specialized consulting services:

$$\text{Engineering costs} = \$2.30\,D^{0.8} \tag{2.98}$$

General site costs. This includes construction camp costs, specialized construction equipment, and general construction site costs:

$$\text{General site costs} = \$0.310\,D^{0.9} \tag{2.99}$$

Project supervision. This includes project supervision, scheduling and budgeting, and construction management:

$$\text{Project supervision costs} = \$1.80\,D^{0.8} \tag{2.100}$$

Administration. This includes local office administration by corporate owner's representatives, accounting and payment of general contractor, legal costs, plus preproduction employment of key operating staff:

$$\text{Administration costs} = \$1.50\,D^{0.8} \tag{2.101}$$

Project overhead costs as a percentage of direct project costs tend to vary depending on the size and complexity of the project. The lower percentages of 4 to 6% would be typical for \$100 million projects and conventional technology, whereas the higher percentages of 8 to 11% would apply to smaller \$10 million project that are technically novel or complex.

Working capital. The allowance for working capital for a mining project should be sufficient to cover all operating costs plus purchase of the initial inventory of capital spares and parts until revenue is received from smelters or purchasers of metallic products. The time period elapsing before receipt of revenue sufficient to pay imminent operating costs will vary depending on the smelter terms or marketing terms, but the typical allowance is about 10 weeks after the concentrator is operating at full capacity.

Typical working capital allowance is equal to the operating costs for 10 weeks after commissioning of concentrator plus cost of purchasing initial inventory of capital spares and parts.

Whenever the mine or mill design is based on extensive usage of reconditioned used equipment, there is a higher frequency of equipment downtime that requires additional time allowance of working capital; this will decrease the apparent savings of used equipment.

The total overhead costs as a summation of the different components are shown in Figure 2.22.

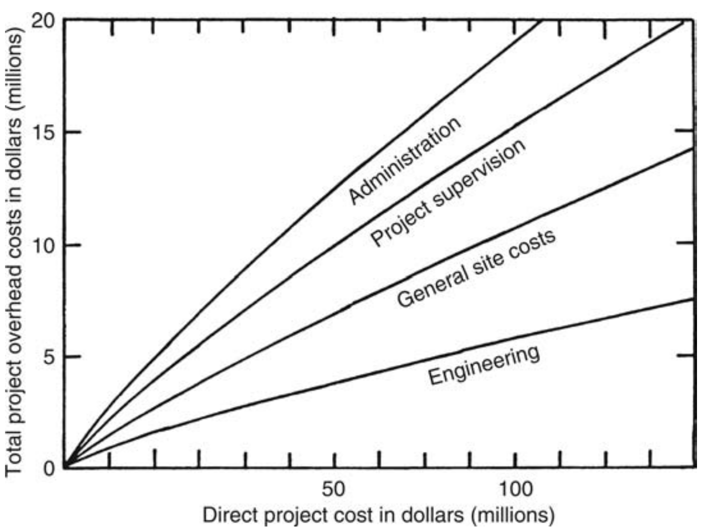


Figure 2.22. Project overhead versus direct costs (O'Hara & Suboleski, 1992).

Daily operating costs

Introduction. In this section the operating costs per day for each activity will be presented in the form

$$\text{Operating cost} = KT_I^x \tag{2.102}$$

The operating cost per ton can be derived from the given formula simply by dividing the operating cost per day by the tons mined (or processed) per day. If, for example,

$$\text{Operating cost per day} = 100\,T_I^{0.7} \tag{2.103}$$

the operating cost per ton is

$$\text{Operating cost per ton} = 100\frac{T_I^{0.7}}{T_I} = 100\,T_I^{20.3} \tag{2.104}$$

Pit operating costs. The operating costs of open pit mines depends on the size and numbers of drills, shovels, and trucks, which in turn is dependent on the tons per day of ore and waste. In most open pit mines mining low grade ore, there is little if any difference in the specific gravities, blasting characteristics, and drillabilities of ore or waste, and the haulage distance to the ore dump usually does not differ very much from the waste haulage distance. Consequently, the cost of mining a ton of ore will be virtually the same as the cost of mining a ton of waste.

The daily operating costs are:

$$\text{Drilling cost per day} = \$1.90\,T_p^{0.7} \tag{2.105}$$

$$\text{Blasting cost per day} = \$3.17\,T_p^{0.7} \tag{2.106}$$

$$\text{Loading cost per day} = \$2.67\,T_p^{0.7} \tag{2.107}$$

$$\text{Haulage cost per day} = \$18.07\,T_p^{0.7} \tag{2.108}$$

$$\text{General services cost per day} = \$6.65\,T_p^{0.7} \tag{2.109}$$

The open pit general services cost includes the cost of pit maintenance, road grading, waste dump grading, pumping, and open pit supervision, but it does not include the cost of primary crushing or electric power.

Concentrator operating costs. Although the gyratory crusher may be located at the edge of the open pit, the costs of operating it are grouped under milling costs (rather than as open pit operating costs) since it is the first stage of ore treatment.

The design of the milling flowsheet is usually optimized after extensive testwork on the types of processes tailored to the characteristics of the ore. At the preliminary feasibility stage however, the optimum processing requirements are not known with accuracy, and the costs of processing can only be approximately estimated.

The following cost guides are offered as rough estimates of crushing and concentrating costs per day.

(a) Primary crushing. This cost includes the cost of primary crushing, the cost of conveying the primary crushed ore to the coarse ore stockpile, plus operating costs of the coarse ore stockpile.

$$\text{Crushing costs per day} = \$7.90\,T^{0.6} \tag{2.110}$$

(b) Fine crushing and conveying. This includes fine crushing, conveying from coarse ore storage, and conveying to the fine ore bins.

$$\text{Fine crushing costs per day} = \$12.60\,T^{0.6} \tag{2.111}$$

(c) Grinding. This cost includes the fine ore bin storage and the rod mills, ball mills, and/or SAG (semiautogenous grinding) mills:

$$\text{Grinding section costs per day} = \$4.90\,T^{0.8} \tag{2.112}$$

(d) Process section. This includes the operating costs of all sections that involve concentration of ore by flotation or by gravity, leaching of metals from ore, thickening of slurries, ion exchange, precipitation, filtering, drying, and recovery of metallic concentrations, or deleterious materials that would otherwise penalize smelter revenue.

$$\begin{array}{l} \text{Processing costs} = \\ \text{per day} \end{array} \begin{cases} \$65\,T^{0.6} \text{ for cyanidation of gold/silver ores} & (2.113\text{a}) \\ \$54\,T^{0.6} \text{ for flotation of simple base metal ores} & (2.113\text{b}) \\ \$34 \text{ to } \$41\,T^{0.7} \text{ for complex base metal ores varying in complexity} & (2.113\text{c}) \\ \$65\,T^{0.7} \text{ for uranium ores by leaching, CCD, solvent extraction, and precipitation} & (2.113\text{d}) \\ \$45\,T^{0.7} \text{ for nonfloatable nonsulfide ores reponding to gravity separation} & (2.113\text{e}) \end{cases}$$

$$\text{Tailings costs per day} = \$0.92\,T^{0.8} \text{ for all concentrators} \tag{2.113f}$$

$$\text{Assaying costs per day} = \$1.27\,T^{0.8} \text{ for all concentrators} \tag{2.113g}$$

$$\begin{array}{l} \text{Supervision, maintanance,} \\ \text{and general costs per day} \end{array} = \$40.80\,T^{0.8} \text{ for all concentrators} \tag{2.113h}$$

Processing costs would be decreased to 55% of those shown by the foregoing formulas when low-grade ore, typically mined by open pit mining, is being treated by a concentrator that rejects tailings at an early stage.

Other operating costs

(a) Electrical power. Expressions for the peak load and daily power requirements for the open pit, crushing plant and concentrator, etc. have been given earlier. The power cost for open pit mines and plants processing T tons of ore per day is

$$\text{Cost of electric power} = \$145\,T^{0.56} \tag{2.114}$$

(b) Surface services. The daily cost of each person in the surface maintenance and general services departments is estimated to be $141 in wages and fringe benefits, plus an average cost of $16 in supplies consumed. If the number of maintenance and general services personnel is N_{sv}, then the daily costs of maintenance and general services departments is

$$\text{Services cost per day} = \$157\,N_{sv} \tag{2.115}$$

The daily costs of the administrative and technical staff, including supplies and services required by them, plus fixed costs for local property taxes and legal fees paid by administrative services, are proportional to the number of staff N_{at}.

Each staff person is estimated to cost on the average \$185 in salary per day, and to consume \$37.60 in supplies and services per day.

$$\text{Total cost per day for administrative and technical staff salaries and supplies} = \$222.60\, N_{at} \tag{2.116}$$

(c) Additional assistance in cost estimation. O'Hara and Suboleski (1992) suggest that the following sources/publications may be of assistance to those making cost estimates.

1. *General Construction Estimation Standards,* 6 volumes, revised annually, published by Richardson Engineering Services, Inc., P.O. Box 1055, San Marcos, CA 92069.

2. *Means Construction Costs*, revised annually, and published by Robert Snow Means Co., Inc., 100 Construction P1., Kingston, MA 02364.

3. *US Bureau of Mines Cost Estimating System Handbook,* 2 volumes, Information Circular 9142 (surface and underground mining), and Information Circular 9143 (mineral processing). Mining and milling costs are as of January 1984. The two volumes, IC 9143, are available from the Superintendent of Documents, U.S. Government Printing Office, Washington, DC 20402.

4. *Canadian Construction Costs: Yardsticks for Costing,* revised annually; available from Southam Business Publications, 1450 Don Mills Rd., Don Mills, ON, Canada, M3B 2X7.

5. *Mining and Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations,* Special Vol. 25, 1982; published by The Canadian Institute of Mining and Metallurgy, 1 Place Alexis Nihon, 1210-3400 de Maisonneuve Blvd. W., Montreal, PQ, Canada H3Z 3B8.

A recent addition to this list of useful publications is "CAPCOSTS: A Handbook for Estimating Mining and Mineral Processing Equipment Costs and Capital Expenditures and Aiding Mineral Project Evaluations" by Andrew L. Mular and Richard Powlin. Special Vol. 47, 1998; published by The Canadian Institute of Mining, Metallurgy and Petroleum, Xerox Tower, 1210-3400 de Maisonneure Blud. W., Montreal, PQ, Canada H3Z 3B8. Since the passing of T. A. O'Hara, no one, unfortunately, has continued to update his useful curves.

2.4.6 *Detailed cost calculations*

Before discussing some techniques for estimating mining costs, a more detailed cost examination will be presented. With this as a basis, the costs will be grouped in several ways to show the dependence on accounting practice. The overall process is as follows:

Step 1. Given the annual production requirements for ore and waste plus the operating schedule, determine the daily production rate.

Step 2. Select a basic equipment fleet.

Step 3. Calculate the expected production rate for each type of equipment. Calculate the number of machines required. Determine the amount of support equipment needed.

Step 4. Determine the number of production employees required. Determine the number of support employees.

Step 5. Calculate the owning and operating costs for the equipment.

Step 6. Calculate the other costs.

Step 7. Calculate the overall cost per ton.

This procedure will be demonstrated using an example presented by Cherrier (1968). Although the costs are old, the process remains the same.

The cross-sections through the molybdenum orebody used in this example will be presented in Chapter 3. The initial mine design has indicated a pit for which

- Rock type is granite porphyry,
- 32,300,000 tons of waste,
- 53,000,000 tons of ore,
- Stripping ratio SR equals 0.6:1, and
- Average ore grade is 0.28% MoS_2.

The waste will be hauled by trucks to a dump area. The ore will be hauled by trucks to one of two ore passes. The ore is crushed and then transported by underground conveyor to the mill. The step-by-step process will be developed.

Step 1: Daily production rate determination. It has been decided that the annual production rate will be

- 3,000,000 tons ore and
- 2,000,000 tons waste.

The mine will operate 2 shifts/day, 5 days/week, 52 weeks per year, with 10 holidays.

The ore and waste production per shift becomes:

- Ore: 6000 tons/shift,
- Waste: 4000 tons/shift.

Step 2: Selection of a consistent set of pit equipment. The major types of production equipment to be selected are:

- Drills,
- Shovels,
- Trucks.

The bench height has been chosen to be 30 ft. The basic equipment fleet selected consists of:

- 6 yd^3 electric shovels,
- 35 ton capacity rear dump trucks,
- Rotary drills capable of drilling $9\,{}^7/_8''$ diameter hole.

Step 3: Production capacity/Number of machines. Based upon a detailed examination of each unit operation, the following production rates were determined:

- Drills: 35 ft of hole per hour,
- Shovels (waste): 630 tons/hour,
- Shovels (ore): 750 tons/hour,
- Trucks (waste): 175 tons/hour,
- Trucks (ore): 280 tons/hour.

From this, of the number production units and required scheduling were determined for ore and waste:

- Ore
  1 drill (1 shift/day),
  1 shovel (2 shifts/day),
  3 trucks (2 shifts/day).

– Waste
  1 drill (1 shift/day),
  1 shovel (2 shifts/day),
  4 trucks (2 shifts/day).

The support equipment includes:
- 4 dozers (2 shifts/day),
- 2–5 yd^3 rubber tired front end loaders (2 shifts/day),
- 2 road graders (2 shifts/day),
- 1 water truck (2 shifts/day),
- 1 explosives truck (1 shift/day).

The reserve production equipment to be purchased is:
- 2–35 ton trucks,
- 1–5 yd^3 front end loader.

In case of shovel breakdown, a front end loader will substitute.

Step 4: Determine the number of production employees. A manpower scheduling chart is prepared such as is shown in Table 2.36. The overall numbers are summarized below.
- 1 assistant superintendent,
- 4 shift foreman,
- 4 shovel operators,
- 4 oilers,
- 14 truck drivers,
- 2 drillers,
- 2 driller helpers,
- 1 blaster,
- 1 blaster helper,
- 8 dozer operators,
- 4 loader operators,
- 4 grader operators,
- 2 water truck drivers,
- 4 truck spotters,
- 4 crusher operators,
- 4 conveyor operators,
- 10 laborers.

As can be seen, there are 73 production employees.

The crusher/conveyor part of the production system operates 7 days/week and 3 shifts per day. There is one crusher operator and one conveyor operator per shift. The ore passes contain enough storage capacity so that the mill can run 7 days/week even though the mine runs 5.

Step 5: Determine the number of other employees. There are four basic departments at the mine:
- Administration,
- Engineering,
- Mine,
- Maintenance.

The overall structure is shown in Figure 2.23. The administration and engineering departments work straight day shift. The maintenance department works two shifts/day, 5 days per week as shown in the manpower chart (Table 2.37).

Table 2.36. Production employees manpower estimate.

(a) Salaried manpower estimate

Classification	Shift requirements																					Manshifts per week				No. of req'd men
	Sun.			Mon.			Tue.			Wed.			Thur.			Fri.			Sat.							
	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	Total	
Assist. supt.				1			1			1			1			1						5			5	1
Shift				2			2			2			2			2						10				
foreman					2			2			2			2			2						10		20	4
																									Total	5

(b) Day pay manpower estimate

Classification	Shift requirements																					Manshifts per week				No. of req'd men
	Sun.			Mon.			Tue.			Wed.			Thur.			Fri.			Sat.							
	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	Total	
Shovel				2			2			2			2			2						10				
operator					2			2			2			2			2						10		20	4
Shovel				2			2			2			2			2						10				
oiler					2			2			2			2			2						10		20	4
Truck				7			7			7			7			7						35				
driver					7			7			7			7			7						35		70	14
Doser				4			4			4			4			4						20				
operator					4			4			4			4			4						20		40	8
Drill				2			2			2			2			2						10			10	2
operator																										
Drill				2			2			2			2			2						10			10	2
helper																										
Loader				2			2			2			2			2						10				
operator					2			2			2			2			2						10		20	4
Grader				2			2			2			2			2						10				
operator					2			2			2			2			2						10		20	4
Water truck				1			1			1			1			1						5				
driver					1			1			1			1			1						5		10	2
Truck				2			2			2			2			2						10				
spotters					2			2			2			2			2						10		20	4
Blaster				1			1			1			1			1						5			5	1
Blaster				1			1			1			1			1						5			5	1
helper																										
Crusher	1			1			1			1			1			1			1			7				
operator		1			1			1			1			1			1			1			7			
			1			1			1			1			1			1			1			7	21	4
Conveyer	1			1			1			1			1			1			1			7				
operator		1			1			1			1			1			1			1			7			
			1			1			1			1			1			1			1			7	21	4
Laborers				5			5			5			5			5						25				
					5			5			5			5			5						25		50	10
																									Total	68

ADMINISTRATION	ENGINEERING	MINE	MAINTENANCE
1 Office Manager	1 Chief Engineer	1 Assistant Supt.	1 Shop Foreman
1 Clerk	2 Engineers	4 Shovel Operators	2 Shift Foreman
10 General and	2 Surveyors	4 Oilers	1 Master Mech.
Administrative	4 Survey Helpers	14 Truck Drivers	6 Mechanics
Personnel	1 Geologist	2 Drillers	2 Electricians
—	2 Samplers	2 Drillers Helpers	2 Geasers
13*	3 Draftsmen	1 Blaster	2 Oilers
	1 Safety Engineer	1 Blaster Helper	2 Welders
	1 Industrial Eng.	8 Dozer Operators	4 Helpers
	—	4 Grader Operators	2 Janitors
	17	2 Water Truck Drivers	—
		4 Shift Foreman	24
		4 Truck Spotters	
		4 Crusher Oper's	
		4 Crusher Oper's	
		10 Laborers	
		—	
		73	

* Includes Mine Superintendent

Figure 2.23. Mine personnel requirements (Cherrier, 1968).

Step 6: Determine the payroll cost. Table 2.38 summarizes the basic annual wage figures for the various job classifications. Fringe benefits amounting to 25% are not included in this table.

Step 7: Determine the operating costs for the equipment. The total operating costs for the various unit operations include materials, supplies, power and labor. These are summarized in Table 2.39. The labor cost includes the 25% fringe benefits.

Note that there are three different units which have been used to express the costs ($/ft, $/hr and $/ton). Some conversion factors are required to obtain the desired common values of $/ton. These are:

– Drilling: 27.2 tons/ft,
– Loading (ore): 750 tons/hr,
– Loading (waste): 630 tons/hr,
– Hauling (ore): 750 tons/hr (3 trucks),
– Hauling (waste): 630 tons/hr (4 trucks).

To simplify the presentation, only two cost categories will be carried further. These are operating labor and MEP (material, expenses and power) which includes all of the rest. The ore-waste separation will be made. The results are summarized in Tables 2.40 and 2.41.

It is noted that the labor cost is strictly for operating labor and does not include repair labor nor does it include supervision.

Step 8: Determine the capital cost and the ownership cost for the equipment. The capital cost for the pit equipment is given in Table 2.42.

During the life of the mine, some of the equipment will have to be replaced. This is indicated in Table 2.43. As can be seen, the original equipment falls into three lifetime groups (5, 10 and 20 years). These are summarized below:

Life (yrs)	Total original cost ($)
5	861,000
10	396,000
20	1,401,000

Table 2.37. Maintenance employees manpower estimate.

(a) Salaried manpower estimate

Classification	Shift requirements																					Manshifts per week				No. of req'd men
	Sun.			Mon.			Tue.			Wed.			Thur.			Fri.			Sat.							
	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	Total	
Shop foreman				1			1			1			1			1						5			5	1
Shift				1			1			1			1			1						5				
foreman					1			1			1			1			1						5		10	2
Master mechanic				1			1			1			1			1						5			5	1
																									Total	4

(b) Day pay manpower estimate

Classification	Shift requirements																					Manshifts per week				No. of req'd men
	Sun.			Mon.			Tue.			Wed.			Thur.			Fri.			Sat.							
	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	A	B	C	Total	
Mechanic				3			3			3			3			3						15				
					3			3			3			3			3						15		30	6
Electrician				1			1			1			1			1						5				
					1			1			1			1			1						5		10	2
Machinist				1			1			1			1			1						5				
					1			1			1			1			1						5		10	2
Greaser				1			1			1			1			1						5				
					1			1			1			1			1						5		10	2
Oiler				1			1			1			1			1						5				
					1			1			1			1			1						5		10	2
Welder				1			1			1			1			1						5				
					1			1			1			1			1						5		10	2
Helper				2			2			2			2			2						10				
					2			2			2			2			2						10		20	4
Janitor				1			1			1			1			1						5				
					1			1			1			1			1						5		10	2
																									Total	22

The equipment ownership cost consists of two parts:

(1) Depreciation,

(2) Average annual investment cost.

Straight line depreciation with zero salvage value is assumed. Thus the average equipment depreciation per year is \$281,850. The average annual investment (AAI) is calculated using the following formula.

$$AAI = \frac{n+1}{2n}CC \tag{2.117}$$

where n is the life (yrs) and CC is the capital cost (\$).

Table 2.38. Annual payroll (Cherrier, 1968).

Classification	Number required	Unit annual age	Total annual age
Administration			
Mine superintendent	1	$25,000	$25,000
Office manager	1	13,000	13,000
Clerk	1	7,000	7,000
Pro-rated G&A personnel (average salary $8,000)	10	8,000	80,000
Total	13		$125,000
Engineering			
Chief engineer	1	$16,000	$16,000
Mining engineers	2	9,500	19,000
Safety engineer	1	8,000	8,000
Industrial engineer	1	8,500	8,500
Geologist	1	7,500	7,500
Surveyors	2	7,500	15,000
Surveyor helpers	4	5,500	22,000
Draftsmen	3	6,000	18,000
Samplers	2	5,500	11,000
Total	17		$125,000
Mine			
Ass't mine superintendent	1	$18,000	$18,000
Shift foreman	2	11,000	22,000
Shovel operators	4	8,500	34,000
Shovel oilers	4	6,000	24,000
Truck drivers	14	7,000	84,000
Drillers	3	8,000	24,000
Driller helpers	3	6,500	19,500
Blaster	1	8,500	8,500
Blaster helper	1	6,500	6,500
Dozer operators	8	8,000	64,000
Loader operators	6	8,000	48,000
Grader operators	4	8,000	32,000
Water truck driver	2	7,000	14,000
Crusher operators	4	7,500	30,000
Conveyor operators	4	7,500	30,000
Truck spotters	4	5,500	22,000
Laborers	10	5,000	50,000
Total	73		$526,500
Maintenance			
Shop foreman	1	$14,000	$14,000
Shift foreman	2	10,500	21,000
Master mechanic	1	11,000	11,000
Mechanics	6	8,500	51,000
Electricians	2	8,000	16,000
Machinist	2	7,000	14,000
Greaser	2	6,000	12,000
Oiler	2	6,000	12,000
Welder	2	6,500	13,000
Helpers	4	5,500	22,000
Janitor	2	5,000	10,000
Total	26		$196,000

Table 2.39. Mine operating costs (Cherrier, 1968).

Unit operation	Cost category		$/ft	$/hr	$/ton
Drilling	Bits		0.100		
	Maintenance, lub, repairs		0.200		
	Supplies		0.015		
	Fuel		0.250		
	Labor		0.133		
Blasting	Ammonium nitrate				0.0125
	Fuel oil				0.0010
	Primers				0.0020
	Primacord, caps, fuse				0.0030
	Labor				0.0030
Loading	Repairs, maintenance, supplies	cost		7.20	
	Power	per		1.65	
	Lubrication	shovel		0.10	
	Labor			8.75	
Hauling	Tires cost	cost		1.65	
	Tire repairs	per		0.25	
	Repairs, maintenance	truck		2.70	
	Fuel			0.96	
	Oil, grease			0.40	
	Labor			3.60	
Auxilliary equipment	Tracked dozer	materials			0.015
	Rubber tired dozer	supplies			0.013
	Front end loader	repairs			0.010
	Graders	fuel			0.020
	Water truck				0.005
	Labor				0.040
Secondary drilling and blasting	MEP				0.005
	Labor				0.005
Crushing	MEP				0.020
	Labor				0.010
Conveying	MEP				0.018
	Labor				0.013
Snow removal	MEP				0.005
	Labor				0.005

Table 2.40. Summary of direct operating expenses for ore (Cherrier, 1968).

Unit operation	Cost ($/ton)		Total
	MEP	Labor	
Drilling	0.021	0.005	0.026
Blasting	0.019	0.003	0.022
Loading	0.012	0.012	0.024
Hauling	0.024	0.014	0.038
Auxilliary	0.063	0.040	0.103
Secondary D&B	0.005	0.005	0.010
Crushing	0.020	0.010	0.030
Conveying	0.018	0.013	0.031
Snow removal	0.005	0.005	0.010
Total	0.187	0.107	0.294

Table 2.41. Summary of direct operating expenses for waste (Cherrier, 1968).

Unit operation	Cost ($/ton)		Total
	MEP	Labor	
Drilling	0.021	0.005	0.026
Blasting	0.019	0.003	0.022
Loading	0.014	0.018	0.032
Hauling	0.038	0.029	0.067
Auxilliary	0.063	0.040	0.103
Secondary D&B	0.005	0.005	0.010
Snow removal	0.005	0.005	0.010
Total	0.165	0.105	0.270

Table 2.42. Capital costs and annual depreciation for the mining equipment (Cherrier, 1968).

Operation and Item	No.	Unit cost	Total cost	Life (yrs)	Annual deprec.
Drilling					
Bucyrus Erie 40-R Rotary drill	2	$75,000	$150,000	10	$15,000
Blasting					
Powder truck	1	8,000	8,000	10	800
Excavating					
P&H shovel model 1400	2	288,000	576,000	20	28,800
Haulage and Transporation					
Haulpak 35 ton trucks rear dump	9	58,000	522,000	5	104,400
Crusher	2	150,000	300,000	20	15,000
Conveyor	1	395,000	395,000	20	19,750
Chloride spray system	2	15,000	30,000	20	1,500
Pit maintenance					
Bulldozers	3	50,000	150,000	5	30,000
Rubber tired dozers	3	40,000	120,000	5	24,000
Front end loaders	2	45,000	90,000	10	9,000
Road graders	3	25,000	75,000	10	7,500
Sprinkler truck	2	7,000	14,000	10	1,400
Fuel truck	1	9,000	9,000	10	900
Mobile maintenance	1	15,000	15,000	10	1,500
Miscellaneous					
Pick up trucks	4	3,000	12,000	5	2,400
9 passenger trucks	4	4,000	16,000	5	3,200
Truck crane	1	35,000	35,000	10	3,500
Radio system	1	7,000	7,000	5	1,400
Power	1	100,000	100,000	20	5,000
Contingencies (20%)(170,000)			34,000	5	6,800
Total			$2,658,000		$281,850

The depreciation categories of 5, 10 and 20 years will be used. Thus the AAI for $n =$ 5 yrs is

$$\text{AAI (5 yrs)} = \frac{6}{10} \times \$861{,}000 = \$516{,}600$$

Table 2.43. Equipment replacement schedule and cost (Cherrier, 1968).

Item	No.	Year of replacement	Total cost
Haulpak truck	9	6th, 11th, 16th	$522,000
Bulldozers	3	6th, 11th, 16th	150,000
Rubber tired dozers	3	6th, 11th, 16th	120,000
Pick up trucks	4	6th, 11th, 16th	12,000
9 Passenger trucks	4	6th, 11th, 16th	16,000
Radio system	1	6th, 11th, 16th	7,000
Contingencies			34,000
Total			$861,000
Bucyrus Erie drill	2	11th	$150,000
Powder truck	1	11th	8,000
Front end loader	2	11th	90,000
Road grader	3	11th	75,000
Sprinkler truck	2	11th	14,000
Fuel truck	1	11th	9,000
Mobile maintenance	1	11th	15,000
Truck crane	1	11th	35,000
Total			$396,000

The total AAI is

$$\text{AAI} = \$1{,}469{,}930$$

To obtain the average annual investment cost AAIC a percent P (expressed as a ratio) is applied. Included in P are interest, taxes and insurance.

$$\text{AAIC} = P \times \text{AAI} \tag{2.118}$$

In this case 10 percent will be used. Hence

$$\text{AAIC} = 0.10 \times \$1{,}469{,}930 = \$146{,}993$$

The average annual equipment ownership cost becomes

$$\begin{aligned} \text{Ownership cost} &= \text{Depreciation} + \text{AAIC} \\ &= \$281{,}850 + \$146{,}993 \\ &= \$428{,}843 \end{aligned} \tag{2.119}$$

Step 9: Calculation of other capital expenditures (mine). The other capital expenditures at the mine include those for the required mine buildings and the costs associated with the mine development period.

The following list includes all required mine buildings, and their required capital outlays.

Office building (Admin. and eng.)	
Building (brick)	$180,000
Equipment	
Office (desks, files, typewriters, etc.)	40,000
Engineering (Calculators, survey equip. etc.)	25,000
Total	$245,000

Change house	
Building (brick)	$30,000
Equipment (lockers, showers, fixtures, etc.)	$5,000
Total	$35,000
Warehouse	
Building (sheet metal)	$40,000
Fuel station	
Building (sheet metal)	$6,000
Equipment (tanks, pumps)	$10,000
Total	$16,000
Maintenance shops	
Building (sheet metal)	$105,000
Equipment (electrical, cranes, tools, etc.)	$130,000
Total	$235,000
Powder hopper	
Purchase and erection	$12,000
Total for all buildings	$583,000
Contingencies (10%)	$58,300
Total estimated capital expenditures for buildings	$641,300

This will be depreciated over the 20 year mine life. The development expenses are as follows:

Schedule of capital expenditures	
1969	
Development	
Extended exploration	$205,000
Access road and site preparation	200,000
Preliminary stripping	400,000
Tunnel excavation	537,000
Equipment	
Surface installations	641,000
Equipment	1,232,000
Total	$3,215,000
1970	
Development	
Preliminary stripping	$675,000
Ore pass excavation	415,000
Equipment	
Crusher	300,000
Total	$1,390,000

1971	
Equipment	
Mill	12,000,000
Equipment	1,126,000
Total	$13,126,000
Development drilling	205,000
Access road	200,000
Development stripping	1,075,000
Tunnel excavation	537,000
Ore pass excavation	415,000
Total	$2,432,000

Both the buildings and the development costs are considered as sunk. They will be recovered against the ore produced.

Step 10: Calculation of milling costs. In this case the open pit ore tonnage (averaged over 7 days) of 8,600 tpd will go to a mill having a capacity of 34,400 tpd. The additional 25,800 tpd will be supplied by an underground mine. Since the total mill investment is $48,000,000 the share for the open pit operation is $12,000,000. This will be depreciated over the 20 year mine life. The ownership cost (depreciation plus average annual investment) per year is

$$\begin{aligned}\text{Ownership cost} &= \$600{,}000 + \$630{,}000\\ &= \$1{,}230{,}000/\text{year}\end{aligned}$$

The primary crushing will be done at the mine, therefore the first step in milling will be the secondary crushing. Operating costs for milling are estimated to be the following (includes administration and overhead):

Labor	$0.13/ton
Material	0.39/ton
Power	0.26/ton
Total	$0.78/ton

Step 11: Expression of the mining costs. There are a variety of ways by which the mining costs can be expressed. A series of cases will be presented to illustrate this.

Case 1. Direct operating costs. The simplest way is to examine the direct operating mining costs for ore and waste. These are:

Ore

$$\begin{aligned}\text{MEP} &= \$0.187/\text{ton}\\ \text{Labor} &= \$0.107/\text{ton}\\ \text{Total} &= \$0.297/\text{ton}\end{aligned}$$

Waste

$$\begin{aligned}\text{MEP} &= \$0.165/\text{ton}\\ \text{Labor} &= \$0.105/\text{ton}\\ \text{Total} &= \$0.270/\text{ton}\end{aligned}$$

The weighted average costs are:

$$\begin{aligned} \text{MEP} &= \$0.178/\text{ton} \\ \text{Labor} &= \$0.106/\text{ton} \\ \text{Total} &= \$0.284/\text{ton} \end{aligned}$$

Using the average costs, the percent breakdown is:

$$\begin{aligned} \text{MEP} &= 63\% \\ \text{Labor} &= 37\% \end{aligned}$$

Case 2. Total operating cost. The labor costs used in Case 1 did not include all of the people involved under the mine category. The total labor expense is equal to \$530,500. Including fringes of 25%, this figure increases to \$663,125. The labor, MEP (ore), and MEP (waste) costs incurred in the mining of 5,000,000 tons are

$$\text{Labor (with fringes)} = \$663{,}125$$

$$\text{MEP (ore)} = 0.187 \times 3{,}000{,}000 = \$561{,}000$$

$$\text{MEP (waste)} = 0.165 \times 2{,}000{,}000 = \$330{,}000$$

The total MEP cost is \$891,000. The costs per ton of material moved are

$$\begin{aligned} \text{Labor} &= \$0.133/\text{ton} \\ \text{MEP} &= \$0.178/\text{ton} \\ \text{Total} &= \$0.311/\text{ton} \end{aligned}$$

In terms of percentages one finds

$$\begin{aligned} \text{Labor} &= 43\% \\ \text{MEP} &= 57\% \end{aligned}$$

Case 3. Direct operating cost plus maintenance. The basic annual maintenance labor cost is \$196,000. With the 25% fringe this becomes

$$\text{Maintenance labor} = \$245{,}000$$

The overall mine plus maintenance labor cost is therefore

$$\begin{aligned} \text{Labor (Mine + Maintenance)} &= \$663{,}125 + \$245{,}000 \\ &= \$908{,}125 \end{aligned}$$

The average operating cost per ton of material moved is

$$\begin{aligned} \text{Operating cost} &= \frac{\$891{,}000 + \$908{,}125}{5{,}000{,}000} \\ &= \$0.360/\text{ton} \end{aligned}$$

The percent distribution is now

$$\begin{aligned} \text{MEP} &= 49.8\% \\ \text{Labor} &= 50.2\% \end{aligned}$$

Case 4. All mine related costs. The costs for the engineering and administration departments can now be added. The annual wages (including fringes) are \$312,500. The associated MEP is \$150,000. Thus the average operating cost per ton of material moved is

$$\text{Operating cost} = \frac{\$1{,}799{,}125 + \$312{,}500 + \$150{,}000}{5{,}000{,}000}$$
$$= \frac{\$2{,}261{,}625}{5{,}000{,}000} = \$0.452/\text{ton}$$

The cost breakdown is:

$$\text{MEP} = \$1{,}041{,}000$$
$$\text{Labor} = 1{,}220{,}625$$

The percent distribution is:

$$\text{MEP} = 46\%$$
$$\text{Labor} = 54\%$$

Step 11: Productivity calculations. The productivity in terms of tons per manshift can now be calculated. It will be assumed that each employee works 250 shifts per year in producing the 5,000,000 tons of total material. The productivity will change depending on the number of departments included:

$$\text{Mine productivity} = \frac{5{,}000{,}000}{73 \times 250} = 274\,\text{t/ms}$$

$$(\text{Mine} + \text{Maintenance})\ \text{productivity} = \frac{5{,}000{,}000}{993250} = 202\,\text{t/ms}$$

$$(\text{Mine} + \text{Maintenance} + \text{Engineering} + \text{Administration})\ \text{productivity}$$
$$= \frac{5{,}000{,}000}{1293250} = 155\,\text{t/ms}$$

Step 12: Mine ownership costs/ton. As indicated earlier, in addition to the operating costs, there are a number of capital costs to be charged against the material moved.

These are:
- Equipment ownership costs;
- Development costs;
- Mine buildings.

The development and mine buildings will be amortized over the total amount of material moved. In this case

$$\text{Cost/ton} = \frac{\$641{,}3001\$2{,}432{,}000}{84,800,000} = \$0.036$$

The equipment ownership cost is

$$\text{Cost/ton} = \frac{\$428{,}843}{5{,}000{,}000} = \$0.086$$

Hence the ownership cost/ton is

$$\text{Ownership cost/ton} = \$0.036 + \$0.086 = \$0.122$$

Step 13: Total mining cost. The total mining cost is equal to the total operating cost plus the ownership cost. In this case it is

$$\text{Total mining cost} = \$0.452/\text{ton} + \$0.122/\text{ton} = \$0.574/\text{ton}$$

Note that the ownership cost here is about 27% of the operating cost and 21% of the total mining cost.

Step 14: Milling cost. As was indicated earlier, the mills operating costs per ton is

$$\begin{aligned} \text{Mill operating cost} &= \$0.78/\text{ton} \\ \text{Mill recovery} &= 90\% \end{aligned}$$

The mill ownership cost/year is the depreciation/year + AAIC/year.
In this case

$$\begin{aligned} \text{Depreciation/year} &= \$600,000 \\ \text{AAIC/year} &= \underline{\$630,000} \\ \text{Total} &= \$1,230,000 \end{aligned}$$

The ownership cost per ton milled is

$$\text{Ownership cost/ton} = \$0.41$$

This is about 53% of the mill operating cost and 34% of the total milling cost.

Step 15: Profitability estimate. The revenues are attributable to the ore and all the costs must now be charged against the ore as well.

Revenue per ton of ore

Average grade	$= 0.28\%$ MoS_2
Recovery	$= 90\%$
Mo contained	$= 60\%$ of recovered MoS_2
Price per lb of contained Mo	$= \$1.62$
Revenue $= 2000 \times 0.0028 \times 0.90 \times 0.60 \times 1.62$	$= \$4.90$/ton

Cost per ton of ore

Mining of ore	$= \$0.452$
Stripping of waste	$= 0.452 \times 0.6 = \$0.271$
Mine operating (Mining + Stripping)	$= \$0.72$
Mine overhead (27% of mine operating)	$= \$0.20$
Mill operating	$= \$0.78$
Milling overhead (53% of mill operating)	$= \$0.41$
Total cost	$= \$2.11$/ton
Profitability per ton	$= \$4.90 - \$2.11 = \$2.79$/ton

2.4.7 *Quick-and-dirty mining cost estimates*

After going through the detailed cost estimate, one might ask about ways in which ballpark estimates could be made. Even if existing operations are not willing to release cost data they sometimes will provide data concerning cost distributions, production rates (ore and waste), total personnel and stripping ratio. If so then the following formulae may be used for making preliminary cost estimates (Pfleider & Wheaton, 1968):

$$\text{Labor cost per ton} = \frac{\begin{array}{c}\text{Estimated average labor cost per man shift}\\ \text{(including fringe benefits)}\end{array}}{A} \tag{2.120}$$

$$\text{Total operating cost per ton} = \frac{\text{Labor cost per ton}}{B} \tag{2.121}$$

$$\text{Total mining cost per ton} = \text{Total operating cost}(1 + C) \tag{2.122}$$

where A is the estimated average tons per manshift (t/ms),

$$B = \frac{\text{Labor cost}}{\text{Total operating cost}}, \quad \text{as an estimated value}$$

$$C = \frac{\text{Ownership costs}}{\text{Total operating costs}}, \quad \text{as an estimated value}$$

The information contained in the previous section will be used in this example.

Given:

Productivity	= 155 t/ms
Operating cost percentage: Labor	= 54%
MEP	= 46%
Overhead cost	= 27% of operating cost
Stripping ratio	= 0.6:1
Estimate: base labor cost/hour	= \$4
Average fringe benefits	= 25%

Calculation:

$$\text{Labor cost/shift} = 8\text{ hr} \times \$4/\text{hr} \times 1.25 = \$40/\text{shift}$$

$$\text{Labor cost/ton} = \frac{\$40}{155} = \$0.26/\text{ton}$$

$$\text{Total operating cost/ton} = \frac{0.26}{0.54} = \$0.48/\text{ton}$$

$$\text{Total mining cost/ton} = 0.48(1 + 0.27) = \$0.61/\text{ton}$$

$$\begin{aligned}\text{Total mining cost/ton of ore} &= 0.61(1 + SR) \\ &= 0.61(1.6) = \$0.97/\text{ton ore}\end{aligned}$$

As can be seen, this compares well with the value of $0.92 obtained in the long calculation.

2.4.8 *Current equipment, supplies and labor costs*

Current prices for mining equipment and supplies can easily be obtained from the individual manufacturer or supplier. Lists of such suppliers are published yearly in special publications of the *Engineering/Mining Journal (E/MJ), Canadian Mining Journal* and *Coal Age.* Several companies publish guidebooks which are extremely useful for making preliminary cost estimates. The *Cost Reference Guide* published by Equipment Watch, a Penton Media brand. New York City, NY. (Equipment Watch, 2012), contains ownership and operating costs for a wide range of mining equipment. The table of contents is given below:

Section	Description
1	Aerial Lifts
2	Aggregate Equipment
3	Air Compressors
4	Air Tools
5	Asphalt Equipment
6	Buckets
7	Compactors
8	Concrete Equipment
9	Cranes
10	Crawler Tractors
11	Drilling Equipment
12	Electric Motors
13	Excavators
14	Forestry Equipment
15	Generator Sets
16	Graders
17	Heaters
18	Internal Combustion Counterbalanced Lift Tracks
19	Marine Equipment
20	Off-Highway Trucks
21	On-Highway Trucks
22	On-Highway/Off-Highway Trailers
23	Pile Drivers
24	Portable Water Towers
25	Pumps
26	Railroad Equipment
27	Road Maintenance Equipment
28	Rough Terrain Lift Trucks
29	Scrapers
30	Sectional Steel Scaffolding
31	Shop Tools
32	Shoring

Table 2.44. Specifications and costs associated with Bucyrus Erie crawler mounted rotary blasthole drills (Equipment Watch, 2012).

Equipment specifications

Model	Power	Maximum pulldown capacity	Maximum rotary hole size	Compressor CFM	CWT
39-R	Diesel	90,000 lbs	12-¼″	1645	1900
49-R111	Electric	120,000 lbs	16″	2600	3400
	Diesel	120,000 lbs	16″	2600	3400
59-R	Electric	140,000 lbs	17-½″	3450	4050

Hourly ownership and overhaul expenses

Model	Econ. Hours	Ownership			Overhaul		Total Ownership Cost ($)
		Depreciation ($)	CFC ($)	Overhead ($)	Labor ($)	Parts ($)	
39-R	18,000	87.21	17.69	42.24	70.61	50.69	268.44
49-R111 (E)	18,000	99.66	19.89	38.98	52.19	50.17	260.89
49-R111 (D)	18,000	99.66	20.22	39.66	70.61	57.93	288.08
59-R	18,000	118.35	23.62	56.36	52.19	59.58	310.10

Hourly field repair and fuel expenses

Model	Labor ($)	Parts ($)	Elec./ fuel ($)	Lube ($)	Tires ($)	GEC ($)	Total operating cost ($/hr)	Total hourly cost ($/hr)
39-R	131.85	88.34	–	11.03	–	8.83	240.05	508.49
49-R111 (E)	104.03	80.54	–	12.60	–	8.05	205.22	466.11
49-R111 (D)	131.85	100.97	–	12.60	–	10.10	255.52	543.60
59-R	104.03	95.64	–	9.56	–	14.96	224.19	534.29

Notes:
1. Rates do not include drill bits
2. Operating costs do not include the cost of fuel/electricity
3. Cost of money rate = 2.5%
4. Mechanics wage = $50.31/hr (including fringe benefits)
5. All models include water injection, cab heaters, second pipe rack, and automatic lubrication for upper and lower works.

33 Skid Steer Loaders
34 Straddle Carriers
35 Trenchers
36 Tunneling Equipment
37 Wheel Loaders
38 Wheel Tractors
39 Wrecking Balls

Table 2.44 extracted from the section on electric powered rotary blasthole drills provides an indication of the information provided. Operating costs for selected mining equipment are provided in Table 2.45. The costs are updated on a regular basis.

Table 2.45. Operating costs for selected mining equipment (Equipment Watch, 2012).

Equipment	Make	Model	Operating cost ($/hr)
Standard crawler dozer	Caterpillar	D8T	100
		D9T	135
		D10T	173
4-WD articulated wheel loader	Caterpillar	972H	67.36
		980H	74.71
		988H	115.69
		990H	155.06
		992K	206.75
	Komatsu	WA600-6	117.12
		WA700-3	155.17
		WA800-3	177.10
		WA900-3	186.30
		WA1200-3	358.37
	LeTourneau	L1350	357.06
		L1850	443.45
		L2350	515.37
Articulated frame grader	Caterpillar	12M	43.03
		14M	64.35
		16M	84.30
Crawler mounted rotary blasthole drills, diesel powered*	Atlas Copco	Pit Viper 271	324
		Pit Viper 275	323
		Pit Viper 351	451
		DMM3	384
		DMM2	317
		DML/HP	296
		DML/LP	249
		DML/SP	185
		DM25 SP	147
		DM30	144
		DM45/HP	174
		DM45/LP	157
	Bucyrus Erie	39-R	240
		49-RIII	256
	Drilltech	D245S	172
		D25KS	161
		D45KS	176
		D50KS	178
		D55SP	220
		D75KS	298
		D90KS	397
	Sandvik	D55SP	260
		Tamrock T2000	156
		Tamrock T4000	308
	Schramm	T450 GT	183
		T685 WS	226
Crawler mounted rotary blasthole drills, electric powered*	Atlas Copco	Pit Viper 351	251
	Bucyrus Erie	49-RIII	205
		59-R	224
	Drilltech	1190E	209

(Continued)

Table 2.45. (Continued).

Equipment	Make	Model	Operating cost ($/hr)
Mechanical drive rear dump trucks	Caterpillar	770	74.10
		772	82.61
		773F	102.88
		775F	105.73
		777F	153.59
	Hitachi	750-3	74.58
	Komatsu	HD605-7	111.08
	Terex	TR100	124.84
		TR70	90.99
		TR60	80.88
		TR45	68.17
Electric drive rear dump trucks	Hitachi	EH3000	204.23
		EH3500	235.21
		EH4000	296.61
		EH4500	322.65
	Komatsu	730E	220
		830E	270.48
	LeTourneau	T2240	249.20
	Unit Rig-Terex	MT3300	137.67
		MT4400	316.34

*Rates do not include drill bits
None of the operating costs include the cost of the operator
Electricity cost = $0.14 per KWh
Diesel cost = $3.79 /gallon
Mechanics wage = $50.31/hr (including fringe benefits)

Info Mine USA of Spokane, Washington provides their *Mining Cost Service* to subscribers. The *Service* contains the following cost and related information:

- Electric power;
- Natural gas;
- Cost models;
- Labor;
- Cost indexes and metal prices;
- Supplies and miscellaneous items;
- Equipment;
- Smelting;
- Transportation;
- Taxes;
- Miscellaneous Development series.

As an example of the content Table 2.46 presents the 2011 labor rates for a medium (156 employees, 12,000,000 lbs of copper/year) copper mine in Arizona. Similar information is provided on mines in other states involving a range of minerals. Table 2.47 also taken from the Service, provides cost data on rotary bits.

Typical prices for explosives and blasting accessories are provided in Table 2.48. In performing an economic analysis, capital costs for mining equipment are needed. Such costs are provided by Western Mine Engineering in the convenient form shown in Table 2.49.

The costs are updated on a regular basis.

Table 2.46. Typical labor rates (2011) for open pit copper mine in Arizona (InfoMine USA, 2012).

Job Classification	Hourly wage base	Job Classification	Hourly wage base
Shovel Operator	24.65	Concentrator Operator	22.10
Production Loader	24.65	General SX-EW Operator	22.10
General Heavy Duty Mechanics	23.80	Tailings Dam Construction Crew	22.10
Specialists Crafts – all	23.80	Environmental/Water Facility Operator	22.10
Mill Lead Operator	23.80	Reagent Mixer	21.25
Heavy Duty Mech Operator	23.80	Electrowinning Operator	20.40
Loader Operator (1 2 yards or larger)	23.80	Lube Mechanic Trainee	20.40
Refrigeration Repairman	23.80	Tire Mechanic Trainee	20.40
Environmental/Water Facility Specialist	23.80	Warehouse Person A	20.40
Heavy Equipment Operator – Conc.	22.95	Mine Trainee	19.55
Multi-Equip Operator – Mine	22.95	Helper	19.55
Shovel Qualified Equipment Operator	22.95	Reagent Mixer	19.55
Rotary Drill Operator	22.95	SX-EW Helper	19.55
Truck Heavy Duty Mechanic	22.95	Mine Trainee	18.70
Shovel Heavy Duty Mechanic	22.95	Helper	18.70
Drill Heavy Duty Mechanic	22.95	Utility Helper	18.70
MU Heavy Duty Mechanic	22.95	Lube Mechanic Trainee	18.70
Dozer Heavy Duty Mechanic	22.95	Tire Mechanic Trainee	18.70
Tire Mechanic	22.95	SX-EW Helper	18.70
Lube P.M. Mechanic	22.95	Plantman B	18.70
Heavy Duty Mechanic – Conc.	22.95	Steamcleaner	18.70
Electrician	22.95	Tailings Dam Crew	18.70
Machinist	22.95	Entrance Level Trainee	18.70
Pipe fitter	22.95	Warehouse Person B	18.70
Boilermaker	22.95	Plantman C	17.85
Equipment Operator – Mine	22.10	Laborer	17.85
Ore Truck Driver	22.10	Utility/Insitu Crew	17.85
Water Truck Driver	22.10	Laborer	17.25
Lube Service Mechanic	22.10	Plantman C	17.25
Tire Service Mechanic	22.10		

Wages increased 2% in the last 12 months, effective July 1,201 1.

Benefits				Vacation	
		% paid		Years Service	Days Vacation
Life Insurance	yes	100	$29,000	1	10
AD&D	yes	100	$29,000	3	10
Medical Insurance	yes	80		5	15
Dental Insurance	yes	80		10	20
Vision Insurance	no			15	20
Retirement Benefits	yes			20	20
Defined benefit: $40 per month for each year of service.				25/plus	25

Defined contribution: Yes–for those hired after September 1, 2005 (annual contribution $2,600)

401K plan: No match.

Sick Leave	yes
Disability Income Ins.	yes

9 days paid holidays per year.

Shift differential: Evening – $.30/hour; night – $.45/hour;

Shift Schedule: 8 hour, I0 hour, and 12 hour shifts

Table 2.47. Price of large diameter rotary blast hole drill bits (InfoMine USA, 2012).

Cutter type	Diameter (inches)	Unit	Price ($)
Tungsten carbide	7-7/8	ea.	3,775
	9	ea.	5,155
	9-7/8	ea.	6,501
	10-5/8	ea.	8,092
	12-1/4	ea.	8,110
	15	ea.	16,281
	16	ea.	18,856
Steel tooth	7-7/8	ea.	2,611
	9-7/8	ea.	3,480
	10-5/8	ea.	4,038
	12-1/4	ea.	6,016
	15	ea.	8,489
	17-1/2	ea.	9,569

Table 2.48. Price of explosives and blasting accessories (InfoMine USA, 2012).

1. Bulk explosives	Strength (cal/cm^3)	Unit	Prie ($)
ANFO	739	100 lb	49.05
Site mixed emulsion	820–945	100 lb	46.00
Repumpable emulsion	770–900	100 lb	61.32–97.10
High density emulsion	815–975	100 lb	61.32–73.58
2. Non-electric shock tube millisecond delays	Length (ft)	Unit	Price ($)
	8	100	258.57
	12	100	267.76
	16	100	304.56
	20	100	321.95
	30	100	401.65
	40	100	482.38
	50	100	562.10
	60	100	679.63
	80	100	875.85
	100	100	1031.20
	120	100	1211.07
3. Detonating cord	Strength (grains/ft)	Unit	Price ($)
	7.5	1000 ft	134.90
	15	1000 ft	153.30
	18	1000 ft	153.30
	25	1000 ft	197.25
	40	1000 ft	226.89
	50	1000 ft	229.95
	150	1000 ft	507.93
	200	1000 ft	563.12
4. Cast primers	Weight (lbs)	Unit	Price ($)
	1/3	ea.	2.80
	1/2	ea.	3.57
	3/4	ea.	3.92
	1	ea.	4.68
	2	ea.	8.44
	3	ea.	12.37
	5	ea.	20.41

Table 2.49. Capital cost mining equipment (InfoMine USA, 2012).

1. Rotary drills, electric	Hole size (ins)	Price ($1000)
	9 to 12-1/4	1,842
	9-7/8 to 16	2,898
	10-3/4 to 17-1/2	2,983
2. Graders	Blade width (ft)	Price ($1000)
	12	359
	14	465
	16	765
	24	1775
3. Wheel loaders	Bucket capacity (yd^3)	Price ($1000)
	9	793
	11	1,290
	16	1,752
	18	1,774
	21	2,224
	26	4,633
	33	5,924
	53	7,652
4. Electric cable shovels (rock)	Bucket capacity (yd^3)	Price ($1000)
	11	3,397
	16	4,458
	20	5,307
	27	8,793
	34	9,975
	55	12,655
5. Trucks, rear dump (mechanical drive)	Capacity (tons)	Price ($1000)
	40	630
	60	881
	65	895
	70	1,011
	100	1,288
	150	2,120
	170	2,517
	200	3,105
	250	3,912
	360	4,587
	380	6,050
6. Trucks, rear dump (electric drive)	Capacity (tons)	Price ($1000)
	173	2,521
	200	2,835
	280	5,583
	314	3,935
	360	6,218
7. Crawler tractors (dozers) (with ripper)	Horsepower	Price ($1000)
	260	703
	310	714
	330	735
	350	852
	410	1,047
	570	1,137
	850	2,086

REFERENCES AND BIBLIOGRAPHY

Anonymous 1909a. Monthly Average Metals Price. *The Engineering and Mining Journal* 88(11): 1054.

Anonymous 1909b. Utah Copper Quarterly Report. *The Engineering and Mining Journal* 88(11): 1028.

Benito, R., & S.D. Dessureault. 2008. Estimation of incremental haulage costs by mining historical data and their influence in the final pit limit definition. Mining Engineering. 61(10): 44–49. October.

Benning, I. 2000. Bankers' perspective of mining project finance. JSAIMM. 100(3): 145–152.

Berry, C.W. 1984. Economic Evaluation Phases of a Mining Prospect. *Mine Feasibility – Concept to Completion.* (G.E. McKelvey, compiler). Northwest Mining Association.

Bertisen, J., & G. Davis. 2007. Bias and error in mine project capital cost estimation. Paper 07-082. SME Annual Meeting. Feb 25–28. Denver, CO.

Bhappu, R.R., & J. Guzman. 1995. Mineral investment decision making. E/MJ. 196(7): 36–38.

Bigler, L.M., & J.L. Dobra. 1994. Economic analysis of a hypothetical Mexican gold mine. E/MJ. 195(7): 16EE–16FF.

Blossom, J.S. 1991. Molybdenum. *1989 Minerals Yearbook.* (Volume 1): 719–728. Bureau of Mines, U.S. Government Printing Office.

BLS (Bureau of Labor Statistics) 1991. Employment, Hours, and Earnings, United States 1989–90. Bulletin 2370, vol. 1(3). U.S. Department of Labor.

BLS 1992a. Productivity Measures for Selected Industries and Government Services. Bulletin 2406 (4). U.S. Department of Labor.

BLS 1992b. *Producer Price.* Indexes – Data for 1991 (9).

BLS 1993a. *Producer Price.* Indexes – Data for October 1992 (2).

BLS 1993b. *Employment and Earnings.* (6). U.S. Department of Labor.

BLS 2003. Productivity for mine production and nonsupervisory workers. ftp://ftp.bls.gov/pub/special.requests/opt/dipts/oaehhiin.txt.

BLS 2004a. Average hourly earnings of mine production workers (SIC base). http:/www.bls.gov/data/. Series Report. EEU10000006, EEU10100006, EEU10101006, EEU10102006, EEU10120006, EEU10122006, EEU10140006, EEU10142006.

BLS 2004b. Average hourly earnings of production workers (NAICS base). http://www.bls.gov/data/. Employment, Hours and Earnings from the Current Employment Statistics Survey (National). Create customized tables (one screen). Average hourly earnings of production workers, natural resources and mining, and mining (except oil and gas).

BLS 2004c. Producer price indexes for construction machinery and equipment. http://www.bls.gov/data/. Producer price index commodity data. Most requested statistics. Construction machinery and equipment. Machinery and equipment.

BLS 2004d. Labor productivity expressed in output per hour. http://data.bls.gov/PDQ/outside.jsp?/survey=ip/.

BLS 2005a. What is NAICS? http://www.bls.gov/sae/saewhatis.htm

BLS 2005b. NAICS 11 & 21: Natural resources and mining. http://www.bls.gov/iag/natresmining.htm

BLS 2005c. CES Series for Natural Resources and Mining Under NAICS. http://www.bls.gov/ces/naturalres/htm.

BLS. 2012a. Bureau of Labor Statistics.
(col. A) BLS Series Report CEU1021200008, http://data.bls.gov/cgi-bin/srgate
(col. B) BLS Series Report CEU1021210008, http://data.bls.gov/cgi-bin/srgate
(col. C) http://www.econstats.com/blsftp/ceu102121y.htm Employment, Hours, and Earnings–National (CE) (BLS ftp ce)
(col. D) http://www.econstats.com/blsftp/ceu1021230y.htm
(col. E) not updated, use previous edition table reference

BLS. 2012b. Bureau of Labor Statistics.
PPI Databases, Commodity Data including stage-of-processing indexes.
One-Screen Data Search, Machinery and Equipment (code 11), Construction machinery and equipment (code 112), Not seasonally adjusted
Or
Producer Price Indexes, Series Reports, WPU112 http://www.bls.gov/ppi/

BLS. 2012c. Bureau of Labor Statistics. Productivity and Costs, LPC Databases, Industry Productivity and Costs, One Screen Data Search. http://www.bls.gov/lpc/#tables

Bourne, H.L. 1994. What it's worth: A review of mineral royalty information. Mining Engineering. 46(7): 641–644.
Bourne, H.L. 1995. What it's worth: A review of mineral royalty information. Mining Engineering. 47(7): 654–657.
Bourne, H.L. 1996. What its worth: A review of mineral royalty information. Mining Engineering. 48(7): 35–38.
Camm, T.W. 1991. Simplified cost models for prefeasibility mineral evaluations. USBM. *Information Circular* 9298.
Camm, T.W. 1994. Simplified cost models for prefeasibility mineral evaluations. Mining Engineering. 46(6): 559–562.
Camus, J. 2002. Management of mineral resources. Mining Engineering. 54(1): 17–25.
Camus, J.P. 2002. Management of Mineral Resources: Creating Value in the Mining Business. SME. Littleton, Colorado. 107 pages.
Cavender, B. 2001. Leadership vs management: Observations on the successes and failures of cost-reduction programs. Mining Engineering. 53(8): 45–50.
Cherrier, T.E. 1968. A Report on the Ceresco Ridge Extension of the Climax Molybdenite Deposit. MSc. Thesis. University of Minnesota.
CMJ (Canadian Mining Journal) 1986. *Reference Manual and Buyers Guide.* Southam Business Publications.
CMJ 2004. Mining Source book. Canadian Mining Journal. www.canadian mining journal.com
CMJ 2010. Canadian Mining Journal, 2010 Mining Sourcebook.
Coal Age 1992. Buyers Guide. *Maclean Hunter Publication* (6): 57–107.
Cost Reference Guide. 2012. Equipment Watch, a Penton Media brand. New York City, NY.
Crowson, P. 2011. Copper focus – The copper conundrum. Mining Magazine. November. pp. 14–21.
Dataquest Inc. *Cost Reference Guide for Construction Equipment.* Palo Alto, Calif.: Equipment Guide-Book Company.
Davis, G. 1996. Using commodity price projections in mineral project valuation. Mining Engineering. 48(4): 67–70.
Davis, G. 1998. One project, two discount rates. Mining Engineering. 50(4): 70–74.
Davis, G.A. 1995. (Mis)use of Monte Carlo simulations in NPV analysis. Mining Engineering. 47(1): 75–79.
Dixon, S. 2004. Definition of economic optimum for leaching of high acid consuming copper ores. Pre-print Paper 04-04. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.
Dran, J.J. Jr. & H.N. McCarl 1977. An examination of interest rates and their effect on valuation of mineral deposits. *Mining Engineering* 29(6): 44–47.
Dysinger, S.M. 1997. Capital budgeting: forecasting the future. Mining Engineering. 49(9): 35–38.
E/MJ 1986. *Average Annual Metal Prices.* 1925–1985 (3) 27.
E/MJ 1992. Buyers Guide 1992. *E&MJ* 193(10): BG1-45.
E/MJ 1993. *Prices from the American Metal Market.* Friday January 15, 1993: 74WW.
Emery, X., & A. Cabañas. 2004. Forecasting copper prices in the short and medium terms using geostatistics. CIM Magazine. September. pp. 42–53.
Engineering News Record (ENR). 2012. Historical Cost Indices. http://enr.construction.com/economics/historical_indices/
ENR (Engineering News Record) 1976–1992. *Annual Summary*.
ENR 1993. *ENR Market Trends*. (2).
ENR 2005. ENR's Cost Indexes. www.enr.com.
Equipment Quest. 2003. Cost Reference Guide for Construction Equipment. Primedia Publications.
Ericsson, M., Löf, A., & O. Östensson. 2010. Iron ore review, 2009–2010: New pricing, tight supplies, and China's steel output defined the market. E/MJ. 211(8): 28–34. October.
Finlay, J.R. 1908. Cost of producing the world's supply of copper. *The Engineering and Mining Journal* 86(7): 165–168.
Finlay, J.R. 1909. Cost of pig iron made from Lake Superior ores. *The Engineering and Mining Journal* 87(15): 739–745.
Frimpong, S. & J.M. Whiting. 1996. The pricing of mineral investment options in competitive markets. In Surface Mining 1996 (H.W. Glen, editor): 37–42. SAIMM.
Gallagher, T.C. 1982. Developing a construction cost escalation index for the U.S. mining and metals industry. *Cost Engineering* 24(3): 147–152.

Gentry, D.W. 1988. Minerals project evaluation – An overview. *Trans. Instn. Min. Metall.* (*Sect. A: Min. Industry)* 97(1): A25–A35.

Gentry, D.W. 1998. 1998 Jackling Lecture: Constrained prospects of mining's future. Mining Engineering. 50(6): 85–94.

Hajdasinski, M.M. 1993. A generalized true rate of return of a project. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(280–287). Montreal: CIMM.

Hajdasinski, M.M. 2000. Internal rate of return (IRR) – a flawed and dysfunctional project evaluation criterion. Mining Engineering. 52(8): 49–56.

Hajdasinski, M.M. 2000. Internal rate of return (IRR) as a project-ranking tool. Mining Engineering. 52(11):60–64.

Hansen, B.D. 1999. Reducing Newmont's operating costs and increasing reserves and production at a mid-sized international mining company. Pre-print Paper 99-031. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Hart, W.M. 2006. 2006. Building high performance teams and enhancing cash flow. Paper 06-011. SME Annual Meeting. Mar 26–29. St. Louis, MO.

Heath, K.C.G. 1988. Mine costing. *Trans. Inst. Min. Metall. (Sect. A: Min. industry)* 97(1): A1–A8.

Hrebar, M.J. 1977. Financial analysis *Mineral Industry Costs.* (J.R. Hoskins & W.R. Green, editors): 205–224. Northwest Mining Association.

Hummel, R. 2001. The road to NPV+. JSAIMM. 101(8): 411–420.

Humphreys, D. 1989. The metal markets since 1983 – and into 1989. *International Mining* 6(1): 14–22.

Huss, M.P. 1984. Marketing planning. *Mine Feasibility Concept to Completion* (G.E. McKelvey, compiler). Northwest Mining Association.

IMF, 2012. International Monetary Fund, Primary Commodity Prices www.imf.org/external/np/res/commod/index.aspx www.indexmundi.com/commodities/?commodity=iron-ore&months=120

IMF. 2012. International Monetary Fund. www.indexmundi.com/commodities/?commodity=iron-ore&months=360

Industrial Minerals, 1992. *Prices.* 303(12): 66.

Industrial Minerals. 2012. Monthly Prices, June 2012. www.indmin.com

InfoMine USA, 2012. Mining Cost Service. Periodically updated. Spokane, WA. Available from http://www.costs/infomine.com

IRS (U.S. Internal Revenue Service) 1988a. Business Expenses. Publication 535 (Rev. Nov. 88).

IRS 1988b. Depreciation. Publication 534. (Rev. Nov. 88).

IRS 2004. Publication 535 (2004) Business Expenses. http://www.irs.gov/publications/p535/ch10.html

IRS 2004. Publication 946 (2004): How to Depreciate Property. http://www.irs.gov/publications/p946/index.html

IRC 2004C. Publication 535 (2004): Depletion. http://www.irs.gov/publications/P535/ch10.html

IRS 2011. Internal Revenue Service. Depletion. www.irs.gov/publications/p535/ch09.html

Jackling, D.C. 1909. Operations of Utah Copper Co. During 1908. *The Engineering and Mining Journal* 88(6): 1185–1188.

Ji, C. & Y. Zhang. 1999. Gray-correlation analysis of production cost. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 173–176. Colorado School of Mines: CSM.

Jolly, J.L.W. & D.L. Edelstein 1991. Copper. *1989 Minerals Yearbook.* (Volume 1): 333–386. Bureau of Mines. U.S. Government Printing Office.

Kirkman, R.C. 1979. Discussion of the H.M. Wells paper. *Mining Engineering* 31(4): 406–407.

Kirk, W.S. 1998. Iron ore reserves and mineral resource classification systems. Skillings Mining Review. June 6. pp. 4–11.

Kitco 2005. Charts and Data. http://www.kitco.com/scripts/hist_charts/yearly_graphs.cgi

Kovisars, L. 1983. Mine and mill labor productivity study shows wide range of values. *Mining Engineering* 36(11): 1536–1537.

Kuck, P.H. & C.M. Cvetic 1991. Iron ore. *1989 Minerals Yearbook.* (Volume 1): 521–564. Bureau of Mines, U.S. Government Printing Office.

Labys, W.C. 1977. Minerals commodity modeling – the state of the art. *Mineral Policies in Transition.* (J.H. DeYoung, Jr., editor). The Council of Economics of AIME (11): 80–106.

Laing, G.J.S. 1976. *An Analysis of the Effects of State Taxation on the Mining Industry In the Rocky Mountain States.* MSc. Thesis T-1900. Colorado School of Mines.

Lawrence, M.J. 2000. DCF/NPV modeling: Valuation practice or financial engineering? Pre-print Paper 00-58. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Lawrence, R.D. 2000. Should discounted cash flow projections for the determination of fair market value be based solely on proven and probable reserves? Pre-print Paper 00-64. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Lawrence, R.D. 2001. Should discounted cash flow projections for the determination of fair market value be based solely on proven and probable reserves. Mining Engineering. 53(4): 51–56.

Lesemann, R.H. 1977. Commercial and price considerations in mine feasibility and cost studies. *Mineral Industry Costs.* (J.R. Hoskins & W.R. Green, editors): 195–204. Northwest Mining Association.

Lewis, F.M., Chase, C.K. & R.B. Bhappu. 1978. Copper production costs update. *Mining Engineering* 30(4): 371–373.

Lewis, P.J. & C.G. Streets 1978. An analysis of base-metal smelter terms. *Proceedings of the 11th Common wealth Mining and Metallurgical Congress:* 753–767. Hong Kong: IMM.

Lillico, T.M. 1973. How to maximize return on capital when planning open pit mines. *World Mining* 9(6): 22–27.

LKAB. 2005. Personal communication. www.lkab.com

Long, K. 2011. Statistical methods of estimating mining costs. Paper 11-023. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Manssen, L.B. 1983. Financial evaluation of mining projects: Is 'common practice' enough? *Mining Engineering* 35(6): 601–606.

Mason, P.M. 1984. Capital and operational planning for open pit in a modern economy. *18th International Symposium on Application of Computers and Mathematics in the Mineral Industry*: 791–802. London: IMM.

Metal Bulletin 1993. Metal Prices (1). Trans-world Metals (USA) Ltd.

Metal Bulletin 2004. www.metalbulletin.com. Personal communication.

Metal Bulletin 2012. Personal communication. http://www.metalbulletin.com

Metals Bulletin 2012. Personal communication. http://www.metalbulletin.com

Metals Week 1993. Weekly Prices (2).

Metals Week 2005. July. www.plats.com.

Miner, R.B. 2003. Using long-term international concentrate agreements in major project financing. CIM Bulletin 96(1074): 56–64.

Mining Cost Service 1989. *Model Smelter Schedules.* Spokane, WA: Western Mine Engineering.

Mining Cost Service 1992. *Prices for Large Diameter Blast Hole Bits.* Spokane, WA: Western Mine Engineering.

Mining Sourcebook 2004. Canadian Mining Journal. www.canadianminingjournal.com

Mireku-Gyimah, D. & J.O. Darko. 1996. Economic risk analysis of mineral projects in Ghana – A simulation approach. In Surface Mining 1996 (H.W. Glen, editor): 71–76. SAIMM.

Mular, A.L. 1978. Mineral Processing Equipment Costs and Preliminary Capital Cost Estimates. Special Volume 18. CIM.

Mular, A.L. 1982. Mining and Mineral Processing Equipment Costs and Preliminary Capital Cost Estimates. Special Volume 25. CIM.

Mular, A.L. & R. Poulin. 1998. CAPCOSTS: A Handbook for Estimating Mining and Mineral Processing Equipment Costs and Capital Expenditures and Aiding Mineral Project Evaluations. Special Volume 47. CIM.

Murdy, W.W., & R.R. Bhappu. 1997. Risk mitigation in global mining development. Mining Engineering. 49(9): 30–34.

Nangolo, C., & C. Musingwini. 2011. Empirical correlation of mineral commodity prices with exchange-traded mining stock prices. JSAIMM. 111(7): 459–468.

Neretin, A.V. 2007. Cost/Performance Evaluation of Gold Placer Mining Methods – For a Range of Gold Placer Deposits in the Bodaibo District, Russia, AUSIMM Conference Proceedings, World Gold Conference Cairns, QLD, October 2007, pp. 57–65.

Noble, A.C. 1979. Price forecasting and sensitivity analysis for economic analysis of final pit limit. *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, eds): 127–133. New York: SME-AIME.

O'Hara, T.A. 1980. Quick guides to the evaluation of orebodies. *CIM Bulletin (no. 814)* 73(2): 87–99.

O'Hara, T.A. 1981. Chapter 6: Mine evaluation. *Mineral Industry Costs* (J.R. Hoskins, compiler): 89–99. Northwest Mining Association.

O'Hara, T.A. & S.C. Suboleski 1992. Chapter 6.3: Costs and cost estimation. *SME Mining Engineering Handbook* 2nd Edition (Volume 1) (H.L. Hartman, Senior Editor): 405–424. SME.

O'Neil, T.J. 1982. Mine evaluation in a changing investment climate (Parts 1 and 2). *Mining Engineering* Part 1 vol. 34(11): 1563–1566, Part 2, vol. 34(12): 1669–1672.

Osborne, T.C. 1985. What is the cost of producing copper? *International Mining* 2(3): 14–15.

Paulsen, K.R. 1981. Environmental and regulatory costs of mining projects. *Mineral Industry Costs* (J.R. Hoskins, compiler): 27–32. Northwest Mining Association.

Petersen, U. & R.S. Maxwell. 1979. Historical mineral production and price trends. *Mining Engineering* 31(1): 25–34.

Pfleider, E.P. & G.F. Weaton. 1968. Chapter 13.4: Iron ore mining. *Surface Mining* (E.P. Pfleider, editor): 917–921. New York: AIME.

Pierson, G. 1997. Commodity prices from a monetary perspective. Mining Engineering. 49(10): 75–80.

Platts Metals Week, 2012. Weekly Metal Prices, April 16, 2012. The McGraw Hill Companies. http://www.platts.com/Products/metalsweek

PRIMEDIA 2004. Cost Reference Guide for Construction Equipment. Primedia Publications.

Randal, R.L. 1979. Economic modeling and market forecasting *Computer Methods for the '80's*: 938–949. Society of Mining Engineers of AIME.

Randall, R.L. 1980. Sources of cyclical variation in the mineral industries. *Proceedings of the Council of Economics* AIME: 11–118.

Rendu, J-M. 2003. SME meets with the SEC – resources and reserves reporting discussed. Mining Engineering. July. P35.

Roberts, M.C. 2000. Cycles in mineral prices. Pre-print Paper 00-73. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Roberts, M.C. 2002. Cycles in metal prices. Mining Engineering. 54(2): 40–48.

Rosta, J. 1992. Gold, silver. *Mining Engineering* 44(5): 449.

Runge, I.C. 1998. Mining Economics and Strategy. SME. Littleton, Colorado. 295 pages.

Silver, D. 2008. Super Cycle: past, present and future. Mining Engineering. 60(6): 72–77. June.

Simonsen, H. & J. Perry. 1999. Risk identification, assessment and management in the mining and metallurgical industries. JSAIMM. 99(6): 321–328.

Skillings Mining Review 1993a. *Iron Ore Prices.* 82(2).

Skillings Mining Review 1993b. *Rail Freight Rates on Iron Ore and Pellets Per Gross Ton.* 82(2).

Skillings Mining Review 1993c. *Rail and Lake Tariff Rates on Iron Ore and Pellets PerGross Ton.* vol. 82(2).

Skillings 2005. Metal/ore prices. August www.skillings.net.

Smith, L.D. 1999. The argument for a "bare bones" base case. CIM Bulletin 92(1031): 143–150.

Soderberg, A. 1930. *Mining method and costs at the Utah Copper Co., Bingham Canyon, Utah. US Bureau of Mines Information Circular 6234.*

Soma, J.L. 1977. Estimating the cost of development and operating of surface mines. *Mineral Industry Costs* (J.R. Hoskins & W.R. Green, editors): 67–90. Northwest Mining Association.

Southam Mining Group 1993. *Mining Source Book.* Southam Mining Group. Don Mills, Ontario.

Staff 1987. *Bureau of Mines Cost Estimating System Handbook. Part 1. Surface and Underground Mining.* U.S. Bureau of Mines Information Circular 9142: 631 pages.

Staff 1987. *Bureau of Mines Cost Estimating System Handbook. Part 2. Mineral Processing.* U.S. Bureau of Mines Information Circular 9143: 566 pages.

Stebbins, S. 2002. Evaluating the economics of development alternatives early in the project feasibility process. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 587–600. Alaska: SME.

Stermole, F.J. 1979. Economic evaluation of open pit mines. *Open Pit Mine Planning and Design* (J.T. Crawford & W.A. Hustrulid, editors): 136–144. New York: Society of Mining Engineers of the AIME.

Stermole, F.J., & J.M. Stermole. 2012. Economic Evaluation and Investment Decision Methods. (13th edition). Investment Evaluations Corporation.

Stermole, F.J., & J.M. Stermole 1987. *Economic Evaluation and Investment Decision Methods.* (6th Edition). Investment Evaluations Corporation.

Stinnett, L.A., & J.A. Wehinger. 1995. Labor productivity and its impact on product price. Mining Engineering. 47(7): 658–659.

Taylor, H.K. 1977. Mine valuation and feasibility studies. *Mineral Industry Costs*: 1–17. Spokane, WA: Northwest Mining Association.

Taylor, H.K. 1988. Discussion of the article 'Mine Costing' by K.G.G. Heath. *Trans. Inst. Min. Metall. (Sect A*: *Min industry)* 97(10): A196–197.

Taylor, H.K. 1992. Private communication.

Tilton, J.E. 1997. Surviving in the competitive and global mining industry. Mining Engineering. 49(9): 27–29.

Torries, T.F. 1998. NPV or IRR? Why not both? Mining Engineering. 50(10): 69–73.

Ulatowski, T. 1982. Inflationary impacts on project evaluation; real and nominal hurdle rates. *Mineral Industry Costs* (J.R. Hoskins, compiler): 245–248. Northwest Mining Association.

U.S. Bureau of the Census 1992. *Statistical Abstract of the United States.* (112th edition) Washington, D.C.

U.S. Census Bureau. 2005. 1997NAICS Matched to 1987 SIC – Mining. http://www.census.gov/epcd/naics/NAICS21.htm.

USGS. 2005a. Aluminum Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/aluminum/.

USGS. 2005b. Copper Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/copper/.

USGS. 2005c. Lead Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/lead/.

USGS. 2005d. Zinc Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/zinc/.

USGS. 2005e. Gold Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/gold/.

USGS. 2005f. Platinum-Group Metals Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/platinum/.

USGS. 2005g. Silver Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/silver/.

USGS. 2005h. Molybdenum Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/molybdenum/.

USGS. 2005i. Nickel Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/nickel/.

USGS. 2005j. Tin Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/tin/.

USGS. 2005k. Iron Ore Statistics and Information. http://minerals.usgs.gov/minerals/pubs/commodity/iron_ore/.

USGS. 2005l. Historical statistics for mineral and material commodities in the United States. http://minerals.usgs.gov/minerals/pubs/of01-006/.

USGS. 2005m. Metal prices in the United States through 1998. http://minerals.usgs.gov/minerals/pubs/metal_prices/.

USGS. 2012a, U.S. Geological Survey, Mineral Commodity Summaries, 2012 http://minerals.usgs.gov/minerals/pubs/mcs/.

USGS. 2012b, U.S. Geological Survey, Data Series 140, Historical Statistics for Mineral and Material Commodities in the United States, Iron Ore, 2012 http://minerals.usgs.gov/ds/2005/140/.

Viera, F.M. 1993. How to choose projects by means of risk analysis. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(324–331). Montreal: CIMM.

Walker, S. 2005. Management in action – Surface costs: Keeping mining costs at bay. Mining Magazine. October. pp. 28–34.

Warnken, D.E. 2005. A case study in valuation of non-producing mineral rights. Pre-print Paper 05-113. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.

Wells, H.M. 1978. Optimization of mining engineering design in mineral valuation. *Mining Engineering* 30(12): 1676–1684.

Werner, A.B.T. & C. Janakiraman. 1980. Smelter contracts and their place in mineral reserve evaluation. *Proceedings of the Council of Economics* AIME: 85–92.

Western Mine Engineering 2003. Mine and Mill Equipment Costs – Estimator's Guide.

Western Mine Engineering 2004. Mining Cost Service.

Williamson, D. 1997. Into the next millennium – The long bull market in minerals. Mining Engineering. 49(6): 61–64.

World Steel Association. 2012. www.worldsteel.org

Wright, E.A. 1993. A simulation model for assessing mining investment incentives. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(332–342). Montreal: CIMM.

Yernberg, W.R. 2000. Copper 99: Mine closures help copper prices improve. Mining Engineering. 52(1): 36–36.

Yernberg, W.R. 2008. Copper mining industry in Arizona is expanding. Mining Engineering. 60(3): 41. March.

Zhang, M., Stone, P., Middleton, R., & M. Menabde. 2007. A study of the impact of price uncertainty to mining project valuation. Paper 07-054. SME Annual Meeting. Feb 25–28. Denver, CO.

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REVIEW QUESTIONS AND EXERCISES

1. What is meant by the term "future worth"?
2. What is meant by the term "present value"?
3. Assume that the interest rate is 5% compounded daily. What would be the equivalent simple interest rate?
4. If $1000 is placed in a bank savings account earning 5% annually, what is the value after 10 years?
5. Would you rather have $10,000 today or $20,000 in 5 years? Assume an interest rate of 8%.
6. Would you rather have $10,000 today or receive payments of $ 1000/year for the next 20 years?
7. What is meant by the term 'payback period'?
8. If you borrowed $10,000 from the bank today, how long would it require to repay the loan at $1000/year? Interest rate of 5%.
9. What is meant by the term 'rate of return'?
10. You invest $1000 today. It will be repaid in 15 equal payments of $200 over a period of 15 years. What is your rate of return? If you could put the money in a certificate of deposit paying 5%, what should you do?

11. What is meant by the term ‘cash flow’?
12. Redo the cash flow example in Table 2.1 assuming that the capital cost in year 0 was – \$300 and in year 1 was – \$200. What is the total cash flow?
13. For problem 12, using a discounting rate of 15%, what is the net present value?
14. What is meant by the term ‘discounted cash flow’?
15. What is meant by the term ‘discounted cash flow rate of return’?
16. For problem 12, what is the DCFROR?
17. Redo the cash flow example in Table 2.2 assuming that the capital cost in year 0 is \$150 rather than \$100.
18. What is meant by depreciation? How is it included in a cash flow calculation?
19. Redo the example in section 2.2.9 assuming that the salvage value is \$50.00.
20. What is meant by the ‘depletion allowance’?
21. What are the two types of depletion? Describe how they differ.
22. How is depletion included in a cash flow calculation?
23. Redo the cash flow including depletion example given in section 2.2.1 assuming a nickel ore rather than silver.
24. What are the different factors that must be considered when performing a cash flow analysis of a mining property?
25. What is the weight of a
    - short ton
    - metric ton
    - long ton
26. What is meant by the expression ‘long ton unit’? Short ton unit? Metric ton unit? Provide an example of a mineral which is sold in this way.
27. Assume that the price for iron ore fines is 45 g U.S./mtu. What is the value of a concentrate running 63.45% Fe?
28. How much is a troy ounce? What metals are sold by the troy ounce?
29. How is mercury sold?
30. In what form is molybdenum normally sold?
31. What is meant by FOB? What is meant by C.I.F.? Why is it important that you understand the meaning of these terms?
32. Select a metal from Table 2.6 and determine the price today.
33. Select a non-metal from Table 2.7 and determine the price today.
34. Select an ore from Table 2.8 and determine the price today.
35. Compare the price of the Cleveland-Cliffs, Inc. iron ore pellets given in Table 2.9 with the price today.
36. Table 2.10 gives the rail tariff rates for iron ore and pellets. Try to find similar rail tariff data which are applicable today.
37. Table 2.11 gives the lake freight tariff rates for iron ore and pellets. Try to find similar data which are applicable today.
38. What are the primary uses of cobalt? Using values from Platt’s Metals Week, plot the price history over the past 5 years.
39. What are the primary uses of barytes? Using values from Industrial Minerals, plot the price history over the past 5 years.
40. Tables such as shown in Table 2.14 are no longer published by the Skillings Mining Review. However, their “Iron Ore Price Report” provides the prices for the Minnesota

and Michigan products of Cliffs Natural Resources, Inc. Using this resource, plot the development of the prices over the past 5 years.

41. Select a metal from Table 2.15 and plot the price versus time. Describe the price pattern observed. Try to attribute reasons for the major fluctuations observed.
42. Using the endpoint data in problem 41, what would be the average annual percent price increase?
43. Using the data in problem 41, predict what the price might be in 5 years, 10 years and 20 years in the future.
44. Plot the data from Table 2.16 for the same metal selected in problem 41. Describe the price pattern with time. What reasons can you provide, if any, for the major fluctuations?
45. Using the data from problem 44, what price might you expect in 1 year, 5 years, 10 years and 20 years?
46. The price of iron ore fines is given in Table 2.17 for the period 1900 through 2010. Plot the data from the year that you were born until today. Any explanation for the pattern?
47. Table 2.18 presents the monthly spot price for iron ore fines, 62% Fe (China import CFR Tianjin port) expressed in US $/metric ton. These values have been plotted in Figure 2.7. Discuss the curve and its significance.
48. Table 2.19 presents the annual steel production (in thousand tonnes) for the top twenty steel producing countries for years 2000 to 2011 ranked by year 2011 production. The data are provided by the World Steel Association. The data for the top five producers are plotted in Figure 2.8. What does Figure 2.8 show? Your conclusion?
49. Trend analysis is one way to predict future prices based upon historical data but it must be carefully done. To illustrate the types of problems that can occur, select one of the metal price – time data sets for analysis. Fit a second order polynomial to the data. Fit a third order polynomial to the data. Fit an arbitrary higher order polynomial to the data. Compare the results obtained using each equation if you extrapolate 5 years, 10 years and 20 years into the future? What are your conclusions?
50. What is the difference between extrapolation and interpolation?
51. What is meant by an econometric model? When applied in mining, what factors might be included?
52. What is meant by 'net smelter return'? How is it calculated?
53. What are the major elements of a smelter contract?
54. What is meant by 'at-mine-revenue'? Why is it important?
55. Redo the example in section 2.3.5 using the Model Smelter Schedule given in Table 2.24. Make the necessary assumptions.
56. What are some of the different cost categories used at a mining operation? Describe the costs included in each.
57. What is meant by 'general and administrative' cost? What does it include?
58. How might the operating cost be broken down?
59. What are meant by fixed costs? Should they be charged to ore, waste, or both?
60. Discuss the value of information such as contained in Table 2.25 to a student performing a senior pre-feasibility type analysis.
61. Table 2.25 presents the mining costs at the Similkameen operation. Determine a percentage distribution. Present these costs in a pie-type plot.
62. Compare the information presented in Tables 2.25 and 2.26 for the Similkameen/ Similco operation.

63. What is the value of data such as contained in Table 2.26?
64. Compare the Similkameen and Huckleberry mines based upon the data provided in Tables 2.25 and 2.27.
65. The Canadian Mining Journal ceased production of its Mining Sourcebook with the 2010 edition. Hence, Table 2.28 is the last one of its type. Outline the type of information contained in the table. Identify some other sources where one might find these types of data. Choose one of the mines in the table and find the equivalent data from another source.
66. In section 2.4.3, the estimation of costs based upon the escalation of older costs is described. Estimate the cost of the building in year 2011 based upon the different ENR indices. Which value do you think might be the most appropriate? Why?
67. Select one of the categories in Table 2.30. What was the average annual percent hourly wage increase over the period 1970 to 1980? Over the period 1980 to 1990? Over the period 1990 to 2000? On this basis, what would you predict it to be in 2000 to 2010? What was it?
68. What was the reason to change from the SIC to the NAICS base?
69. Using Table 2.32, how much would you estimate the price of a mining truck has increased over the past 10 years?
70. Why is it important to include the change in productivity when escalating older costs? How can you do this?
71. In 1980, O'Hara published the curve relating mine/mill project capital cost to the milling rate. Discuss how you might try to produce a similar curve today.
72. Actual data for a number of open pit mining operations have been provided in Tables 2.25, 2.26, 2.27 and 2.28. Compare some of the estimates from section 2.4.5 with these actual data.
73. Discuss how the curves included in section 2.4.5 might be updated.
74. Compare the mine and mill operating costs predicted using the O'Hara estimators with the actual costs given in Tables 2.25, 2.26, 2.27 and 2.28.
75. A detailed cost calculation has been provided in section 2.4.6. Repeat the example but assume that the ore and waste mining rates are twice those given.
76. In section 2.4.7 a 'quick-and-dirty' mining cost estimation procedure has been provided. Sometimes it is possible to obtain a percentage cost breakdown for a mining operation even if the actual costs are unavailable. How might such a cost distribution be beneficial?
77. Apply the quick-and-dirty approach to the Similkameen data.
78. Apply the quick-and-dirty approach to the Huckleberry data.
79. Some detailed cost information is provided in section 2.4.8. Using the data in Table 2.44, what might be the expected purchase price of a Bucyrus-Erie 59-R drill?
80. When using published costs and/or cost estimators it is very important to know what is included. In Table 2.44, describe the basis/meaning for the different cost components.
81. What are typical depreciation lives for mining equipment (trucks, shovels, drills, front end loaders, graders, dozers, pickup trucks, etc.)?
82. What items are included in equipment ownership costs? What is the approximate split between ownership and operating cost for a piece of equipment?
83. Assume that you have a small equipment fleet consisting of
    - Caterpillar D9T dozer (1)
    - Caterpillar 992K wheel loader (1)

   – Caterpillar 777F trucks (3)
   – Caterpillar 16M grader (1)
   – Atlas Copco Pit Viper 271 drill (1)

   What would be the expected total hourly operating cost? Does this include the operator?

84. Using the values given in Table 2.46, what would be the cost of the operators to run the equipment in problem 83?
85. The Atlas Copco Pit Viper 271 drill will complete a 7-7/8″ diameter hole, 40 ft long in 40 minutes. The life of the tungsten carbide cutter is 5000 ft. What is the cost of each hole?
86. Except for 10 ft of stemming, the hole in problem 85 is filled with ANFO. One 1/3 lb primer is used and the downline is detonating cord (15 grains/ft). What would be the cost of explosive per hole.
87. For the fleet described in problem 83, what would be the approximate capital cost based upon the information contained in Table 2.49?

CHAPTER 3

Orebody description

3.1 INTRODUCTION

Today, most potential orebodies are explored using diamond core drilling. The small diameter core collected from each hole provides a continuous 'line' of geologic information. Each of the recovered cores is studied in detail and the contained information recorded. The process is called 'logging'. Each 'line' is subsequently subdivided into a series of segments representing a particular rock type, structural feature, type of mineralization, grade, etc. By drilling a pattern of such holes, a series of similarly segmented lines are located in space. Using this information, together with a knowledge of the geologic setting and other factors, the mining geologist proceeds to construct a 3-dimensional representation of the mineralized body. The objective is to quantify, as best possible, the size, shape and distribution of the observable geologic features. The distribution of ore grades are correlated to lithology, alteration, structure, etc. The result is a mineral inventory or geological reserve. At this point in the evaluation process, economics have not been introduced so that terms such as 'ore' or 'ore reserve' are not involved.

The development of a mineral inventory involves substantial judgement, assumptions being made regarding sample and assay quality, and the interpretation and projection of geologic features based upon very limited data. The geologic data base, properly gathered and interpreted, should remain useful for many years. It forms the basis for current and future feasibility studies, mine planning and financial analyses. The success or failure of a project can thus be directly linked to the quality of its recorded data base, the drill logs and the maps. This chapter covers some of the basic techniques involved in the development and presentation of a mineral inventory.

3.2 MINE MAPS

The fundamental documents in all stages of mine planning and design are the maps.
Maps are essential for the purpose of:

- collecting,
- outlining, and
- correlating

a large portion of the data required for a surface mining feasibility study. These maps are drawn to various *scales*. The 'scale' is the ratio between the linear distances on the map and

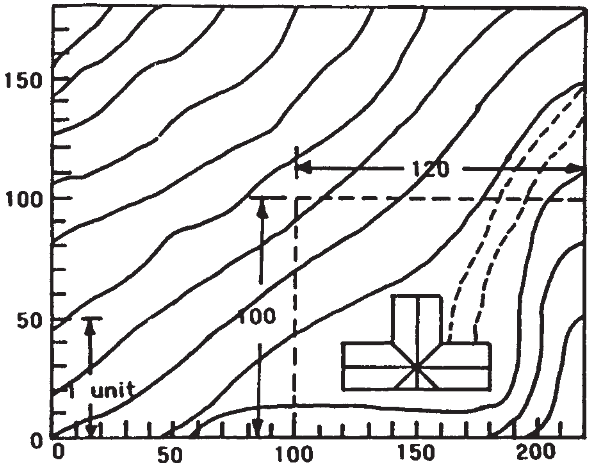


Figure 3.1. Plan map drawn to scale 1:50.

the corresponding distances at the site. In the English system this scale relates 'inches' as measured on the map to 'feet' in the field. This may be expressed as

$$\text{Map distance (in)} = K_E \times \text{Actual distance (ft)} \tag{3.1}$$

where K_E is the English map scale. A typical map scale might be

$$K_E = \frac{1}{200} = 1{:}200$$

This means that an actual distance of 200 ft would be represented by a length of 1 inch on the map. In the metric system, the map scale relates similar map and actual distance units:

$$\text{Map distance (m)} = K_M \times \text{Actual distance (m)}$$

or (3.2)

$$\text{Map distance (cm)} = K_M \times \text{Actual distance (cm)}$$

where K_M is the metric map scale. A scale of 1:1000 means that a length of 1 meter on the map represents 1000 meters in the field. Similarly a length of 1 cm represents a distance of 1000 cm. A metric scale of 1:1250 is very close to the English scale of 1 in = 100 ft.

One speaks of a map being of larger or smaller scale than another. Figure 3.1 shows a particular area drawn to a scale of 1:50. In Figure 3.2 the region within the dashed lines of Figure 3.1 has been drawn to a scale of 1:20. In this figure the building appears *larger*. Thus the scale of the 1:20 map is larger than that of the 1:50 map. A map of scale 1:40 would be of larger scale than one drawn to 1:200.

The general rule is 'the greater the ratio (50 is greater than 20), the smaller is the scale.'

The selection of the most appropriate scale for any map depends upon:

1. The size of the area to be represented.
2. The intended uses for the map.

As more detail and accuracy is required, the scale should be increased.

Mine planning, for example, should be done at a scale that keeps the whole pit on one sheet and yet permits sufficient detail to be shown. For medium to large size metal mines, common planning scales are:

1 in = 100 ft
1 in = 200 ft

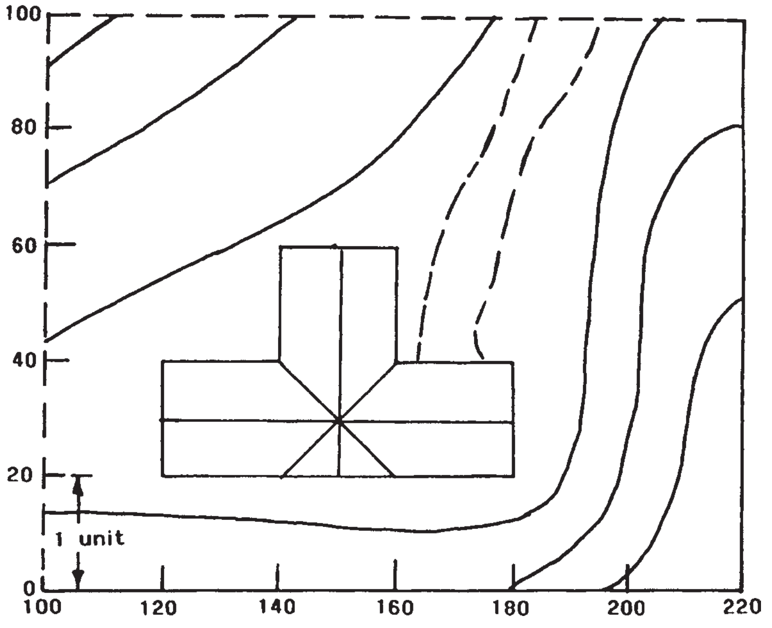


Figure 3.2. A portion of the plan map in Figure 3.1 drawn to scale 1:20.

In the metric system, common scales are:

1:1000
1:1250
1:2000

Geologic mapping is commonly done on a larger scale such as 1 in = 40 ft (the corresponding metric scale is 1:500). For planning purposes, the geologic features (outlines) are replotted onto the smaller scale maps.

The types of maps prepared and used depends upon the stage in the life of the property. At the exploration stage, satellite maps may provide important information regarding structural regimes and potential exploration sites. These can be complemented with infra-red photos, etc. For certain types of information, for example the location of smelters, a small scale map, such as a map of the U.S., may be the most appropriate. Certain materials, such as crushed rock, are highly dependent on transportation costs. Regional maps overlain with circles corresponding to different freight tariffs are useful for displaying potential markets.

A state map Figure 3.3 can provide a considerable amount of basic information:

– nearest highways,
– closest towns,
– property location,
– railroad lines, and
– gross property ownership.

A typical scale for a state map is 1 inch equal to 15 miles.

Very quickly, however, one needs maps of larger scale for the more detailed planning. In the U.S., these often are the topographic maps prepared by the U.S. Geological Survey. These ‘quadrangle’ maps are prepared in two series. The 7½ minute-series (covers an area

Figure 3.3. A section of the Colorado state highway map (Colorado Department of Highways).

of 7½ minutes latitude (high) by 7½ minutes longitude (wide)) with a unitless (metric) scale of 1:24000. This corresponds to a scale of 1 inch equal to 2000 ft. The 15-minute series includes an area 15 minutes in latitude by 15 minutes in longitude. One minute, it should be noted, represents one sixtieth of a degree.

Quadrangle maps show the topographic features, roads, rivers and drainage regions (Fig. 3.4).

Such maps can be enlarged to any desired scale (Figs 3.5 and 3.6), to serve as base maps until more detailed surveying is done. Aerial photos of the area are sometimes available through the state or county engineer's office or a federal agency. Many sections of the U.S. have been mapped on relief maps by the U.S. Corps of Engineers.

Very early in the life of a prospective mining area it is necessary to develop an ownership map. In the U.S., the best available ownership map can be obtained from the office of the county surveyor or the county clerk.

In the western U.S., much of the land and minerals are owned by either the state or by the federal government. A four-step process is followed (Parr & Ely, 1973) to determine the current status of the land.

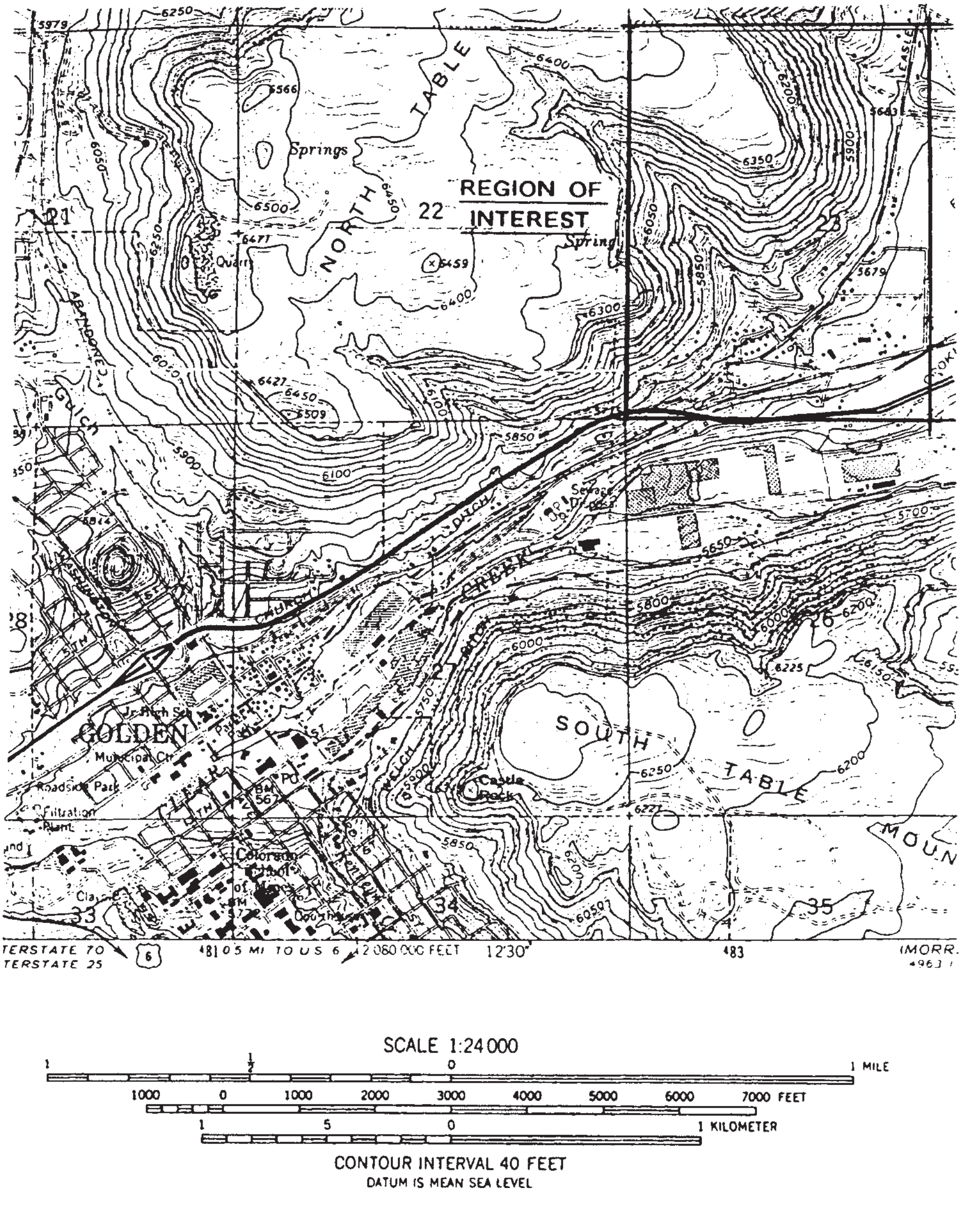


Figure 3.4. The USGS quadrangle map including Golden, Colorado (USGS, 1976).

Step 1. Consult the appropriate State land office, U.S. Geological Survey office or state or regional land office of the U.S. Bureau of Land Management. These offices can determine whether the land in question is available or if there are prospecting permits/leases in effect.

Step 2. If State lands are involved, the appropriate State land office should be consulted regarding the status.

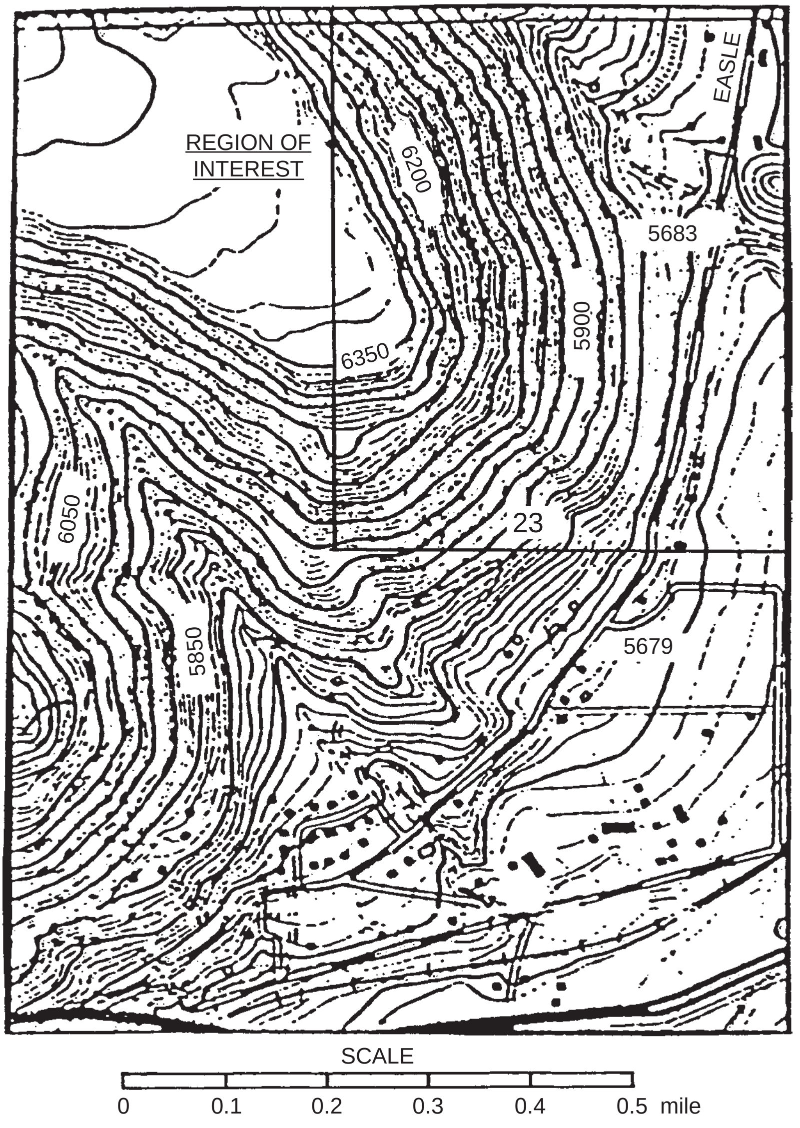


Figure 3.5. Enlargement of the Golden quadrangle map.

Step 3. The records of the appropriate County Recorder or other office should be checked regarding mining claims.

Step 4. A visit to the site should be conducted to determine whether there is anything (mining claim location markers, mine workings, etc.) not disclosed in the other check out steps.

For mine planning and design there are three map types (Phelps, 1968) of different scales:

1. General area map,
2. General mine map,
3. Detailed mine map (plans and cross sections).

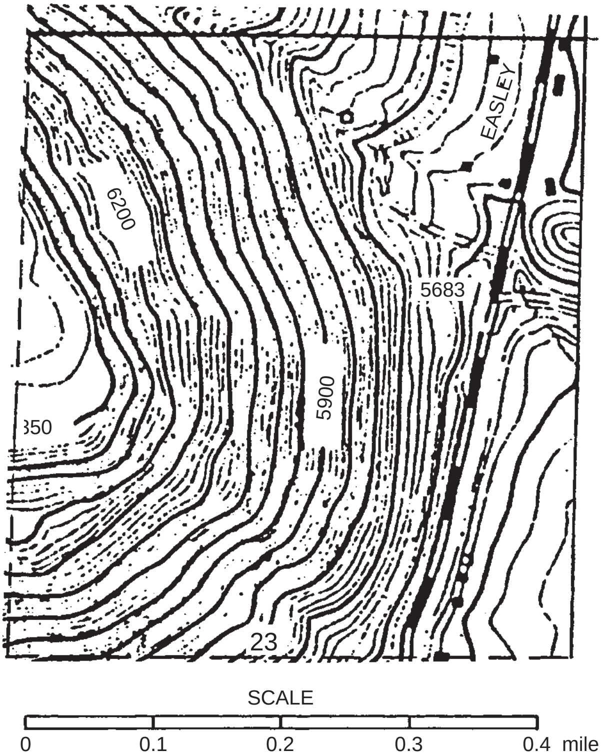


Figure 3.6. Further enlargement of the map.

The objective of the general area map is to show many pertinent features:
- geology (extent of orebodies, mineralized zones),
- transportation routes (highways, railroad, water routes),
- property ownership and control,
- distances to market, processing or transfer points (applicable freight rates),
- available access,
- location of transmission lines for power supply (capacity and construction distances required for connections),
- location of both present and future potential water supply/reservoir areas,
- areas suitable for tailings, slurry and refuse disposal in relation to mining and processing.

As such it is considered to be a small scale map. One can superimpose the data on these maps either directly or through the use of transparent overlays. Figure 3.7 is one example of a general area map. Figure 3.8 is an example of a general area geologic map.

The general mine map is a map of 'medium' scale. It covers a particular region within the general area map. Because the scale is larger, greater detail may be examined. Figure 3.9 is one example of a general mine map. The types of things which might be shown on such a map include:
- processing plant location,
- mine structures,
- power lines,

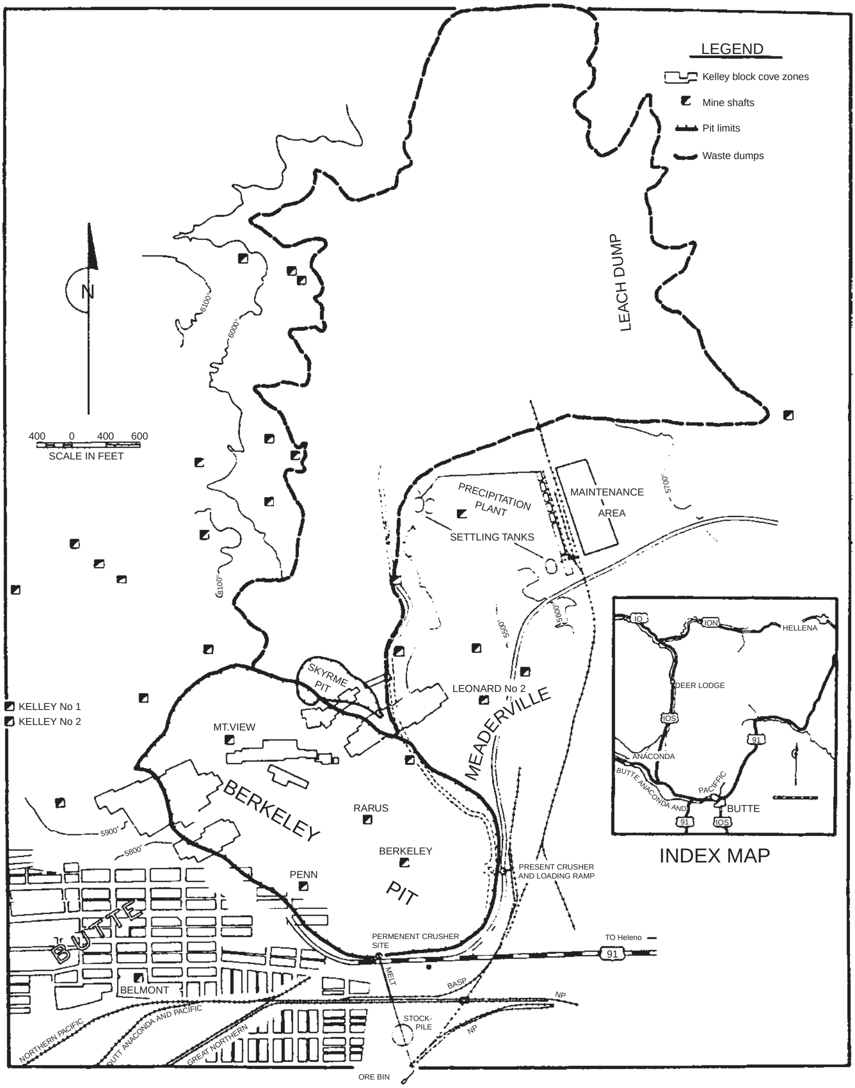


Figure 3.7. An example of a general area map (McWilliams, 1959).

- water supply,
- access roads,
- railroad lines,
- conveyor lines,
- pipelines,
- location of the orebody,
- location of a few drillholes,
- dump/tailing pond locations,
- property ownership and control,
- proposed timing of mining development.

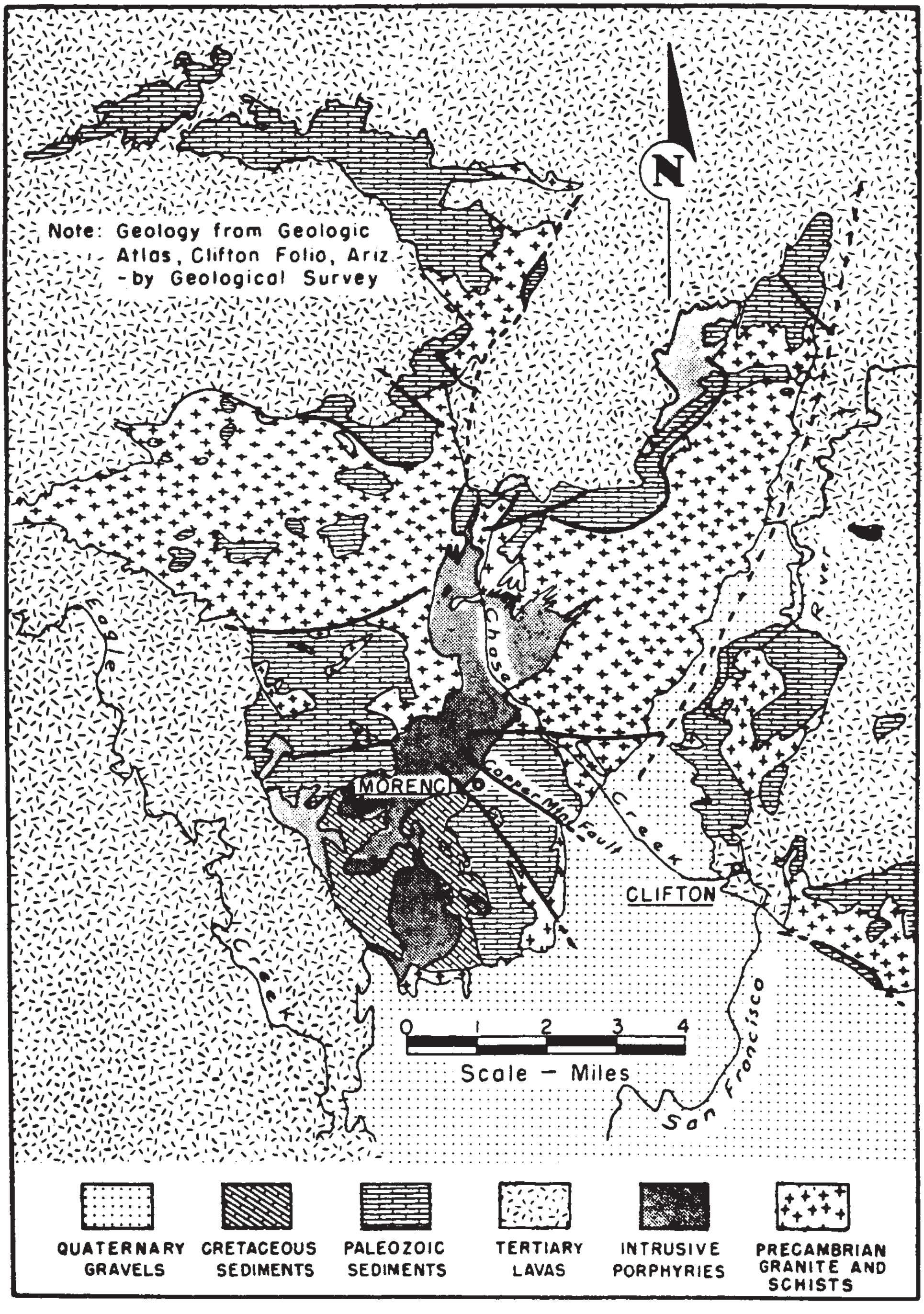


Figure 3.8. An example of a general area geologic map (Hardwick, 1959).

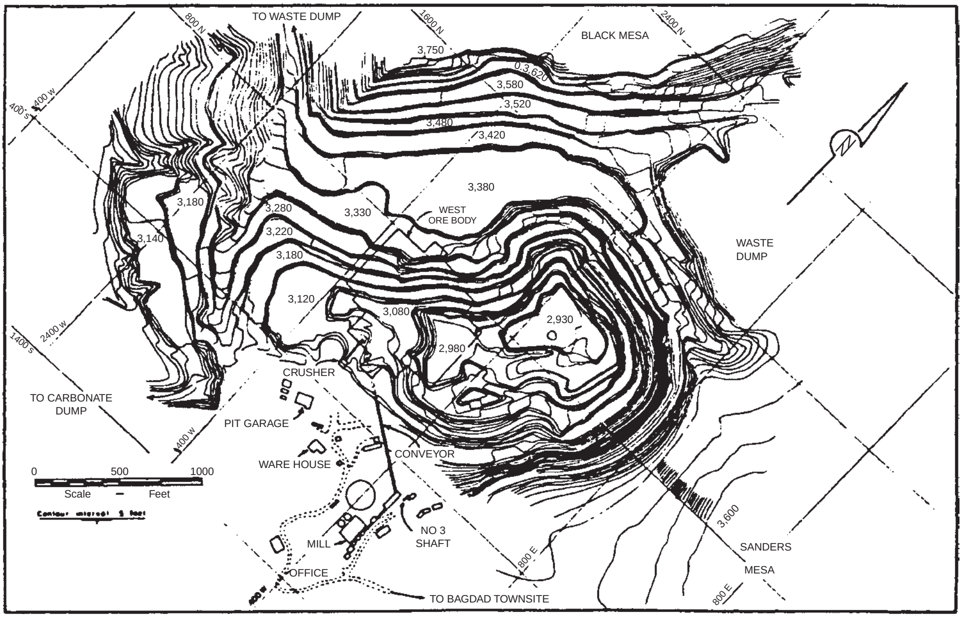


Figure 3.9. An example of a general area geologic map (Hardwick & Jones, 1959).

A general mine geologic map is shown in Figure 3.10.

The detailed mine maps are those used for the actual pit planning. The basic planning package consists of both plan maps and cross sections. Drill hole locations are laid out on plan maps at a scale appropriate to the requirements. In the Northern hemisphere the map is normally laid out in the northeast quadrant (Fig. 3.11).

The active portion of the map is selected with initial coordinates sufficiently large so that there is no danger that other quadrants will be involved. When this occurs the awkward use of $+$, $-$ coordinates or West and South nomenclature is required. Both lead to confusion and possible errors. In the Southern hemisphere, the southwestern quadrant is often used.

The selection of the map 'north' direction is not universal. Some mines select geographic north as their north. Others use magnetic north. One large mine reputedly selected mine north with respect to the direction that the mine manager looked out over the mine from his office window. For elongated deposits, such as shown in Figure 3.12, the deposit lends itself to primary sections running N-S.

Local mine North is then conveniently selected as running perpendicular to the long axis of the deposit. For massive, more or less circular deposits (Fig. 3.13), the choice is less obvious.

Sometimes there is a different coordinate system for the pit as dictated by orebody geometry and that for the mine as a whole.

The exploratory and development diamond drill holes are generally laid out in a more or less regular pattern. When orebody evaluation is to be done using sections, it is convenient, although not imperative, if the drilling and mine grids are aligned. When block models are used to represent the orebodies, this is not as important.

Table 3.1 contains some guidelines for the preparation of mine maps. The line widths for the borders and coordinate systems should be carefully selected so that they can be

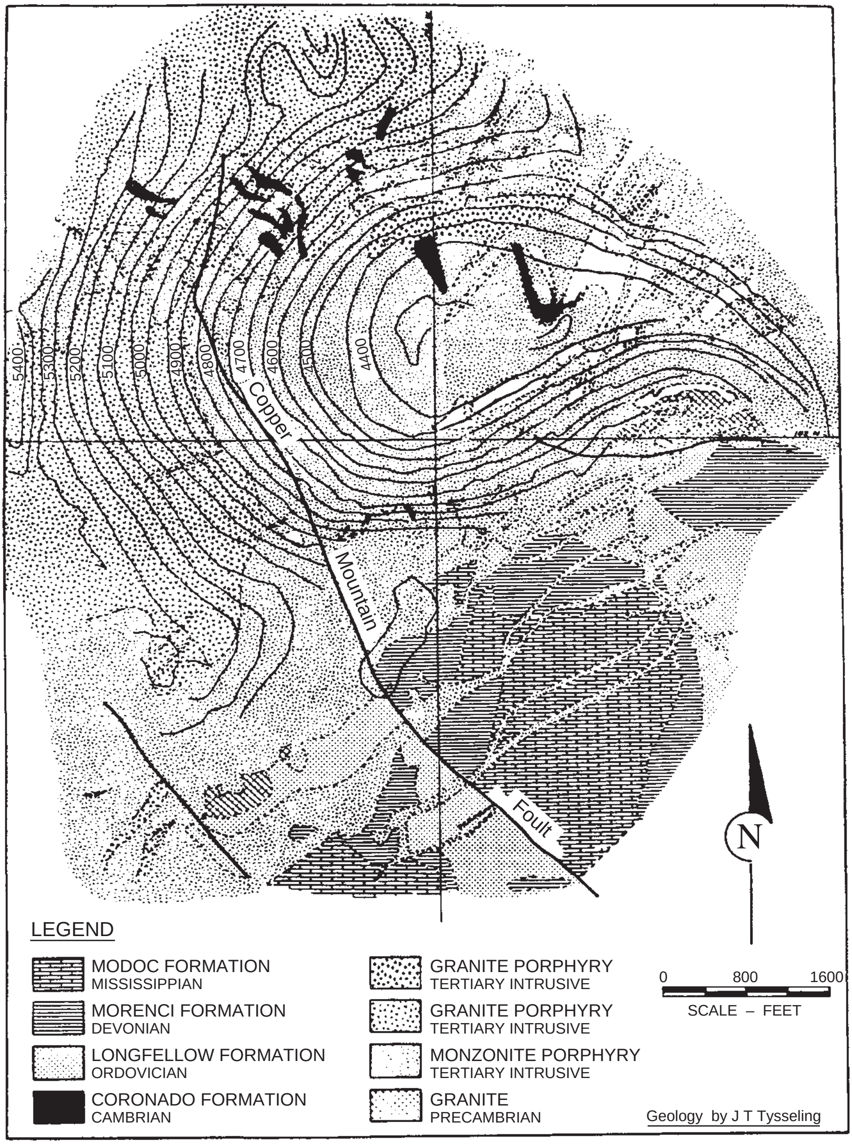


Figure 3.10. An example of a general mine geologic map (Hardwick, 1959).

easily seen, yet do not interfere with the major purpose of the map – that of presenting the graphical information. Modern CAD (computer-aided design) drafting systems have greatly simplified this previously, very tedious and time consuming job.

It is very important that revision/versions of the different maps be maintained.

The maps coordinates are labelled as 1600N, 1400E, etc. These are shown in Figure 3.14.

Vertical sections are made based upon these plan maps. As seen in Figure 3.14, there are two ways of constructing the N-S running sections.

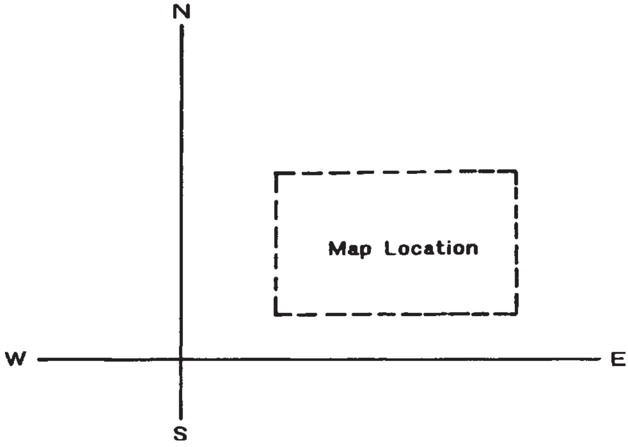


Figure 3.11. Typical NE quadrant plan map location.

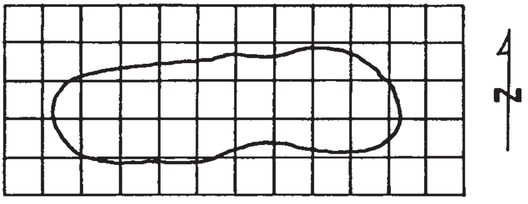


Figure 3.12. Grid system superimposed on an elongated deposit.

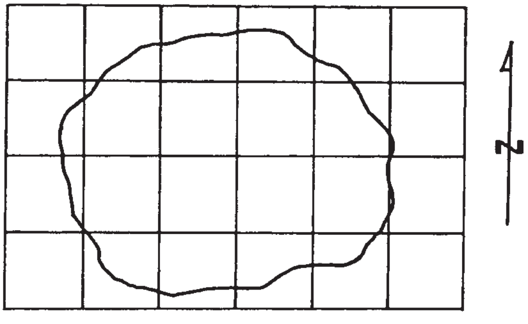


Figure 3.13. Grid system superimposed on a circular deposit.

In constructing section 1050E, one would get the results shown in Figures 3.15a,b depending on whether one is looking from east to west or west to east. Since one is used to numbers increasing from left to right, the east to west choice is made.

The location of the drill holes have been added to the plan map in Figure 3.16. The nomenclature DDH is often used to identify diamond drill holes. These holes are also added to the vertical sections (Fig. 3.17). Normally, the vertical scale for the sections is chosen to be the same as the horizontal. If this is not done, then pit slopes and other features become distorted.

Table 3.1. Guidelines for preparing mine maps (source unknown).

1. Title or subject, location of the area, and an extra identification or indexing notation on the outside or in an upper corner, and a reference to the associated report.
2. Compiler's name, the names of the field mapper, and an index diagram identifying sources of data.
3. Date of field work and date of compilation.
4. Scale; graphic and numerical.
5. Orientation of maps and sections, with magnetic declination shown on maps.
6. Isoline intervals and datum, with sufficient line labels and an explanation of heavier lines, dashed lines, and changes in interval.
7. Legend or explanation, with all units, symbols, and patterns explained.
8. Sheet identification and key, where more than one sheet or a series of overlays is involved.
9. Lines of cross section on the maps and map coordinates or key locations on the sections.
10. Reference grid and reference points.
11. Clarity: All areas enclosed by boundary lines should be labeled even in several places if necessary, so that they can still be identified if photo-reproduction in black-and-white changes the color pattern into shades of gray. Lines and letters should still be distinguishable after intended reductions in size. A trial reproduction can help in selecting colors and lines weights.
12. Size: The size should be compatible with reproduction equipment.

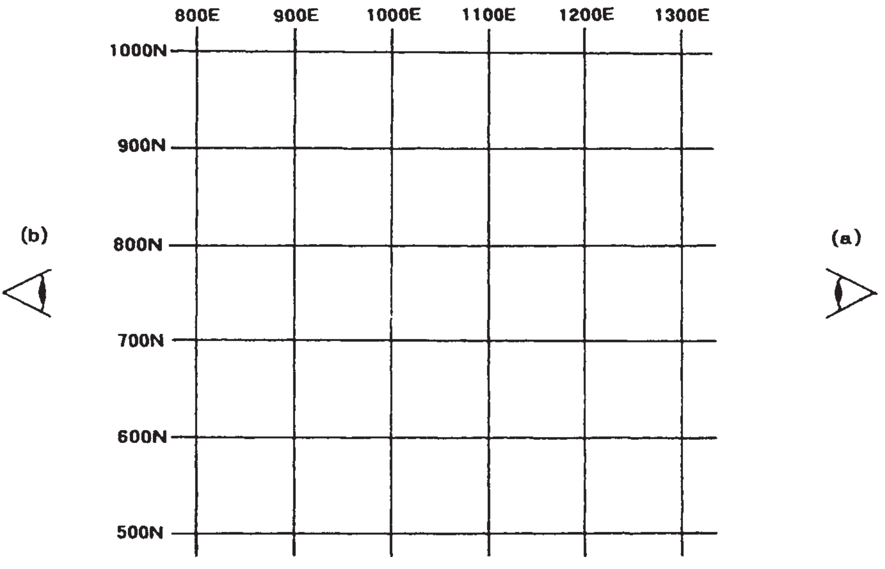


Figure 3.14. The two viewing directions for sections.

By examining the rock types and grades present in the holes on a given section, the geologist connects up similar features (Fig. 3.18). In this way a preliminary view of the size, shape and extent of the orebody is achieved. Such sections and their related plan maps form the basic elements used in mine planning and design. Sometimes, however, it is very helpful for visualization purposes to use isometric projections. Figure 3.19 is such a projection for the Bingham Canyon Mine.

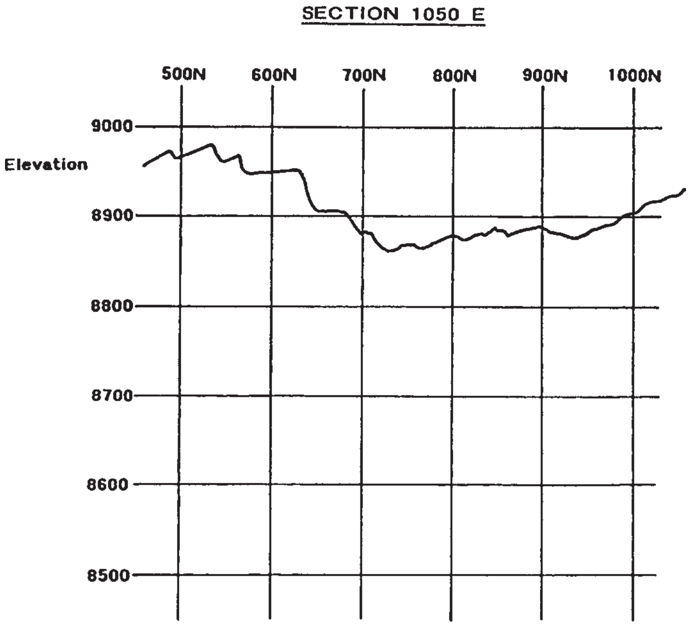


Figure 3.15a. The sections created looking from east to west.

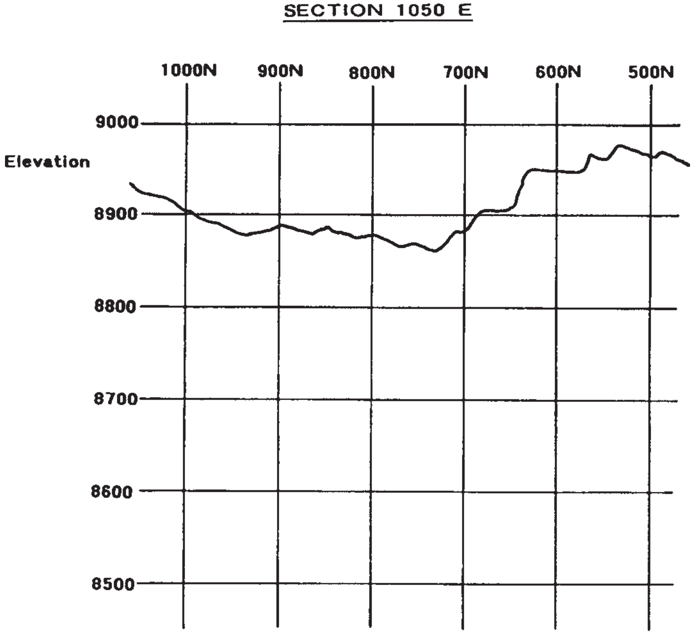


Figure 3.15b. The sections created looking from west to east.

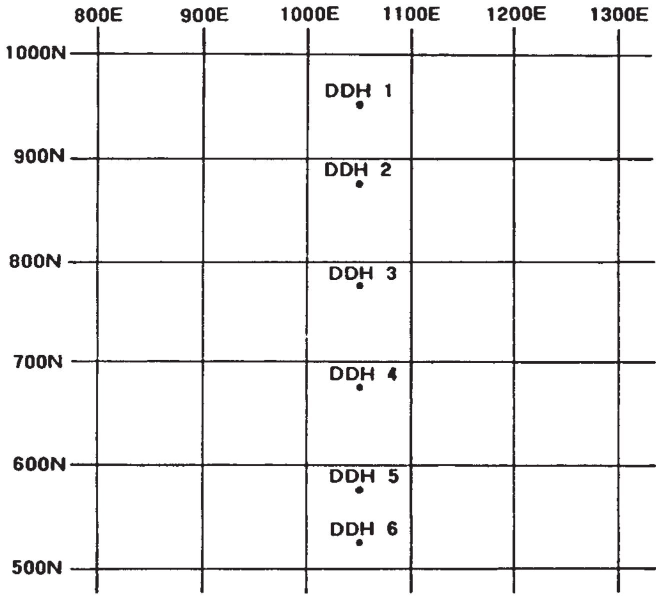


Figure 3.16. Additions of the diamond drill hole (DDH) locations to the plan map.

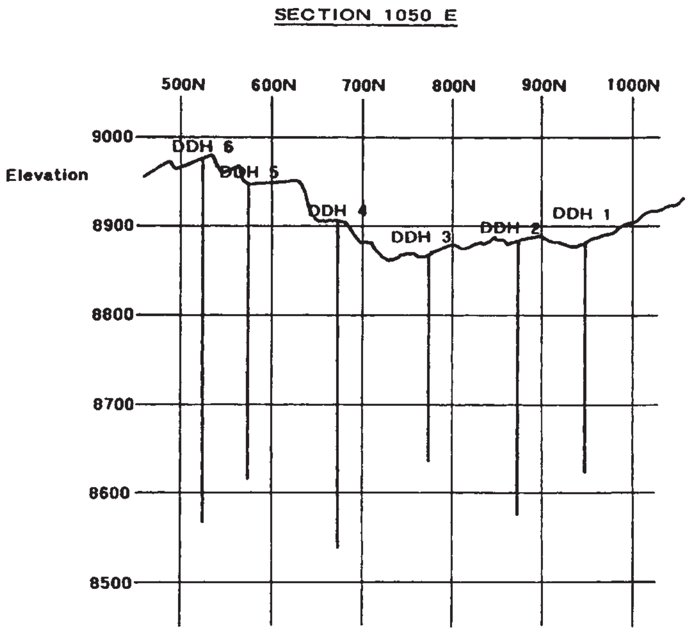


Figure 3.17. Typical sections with the drill holes added.

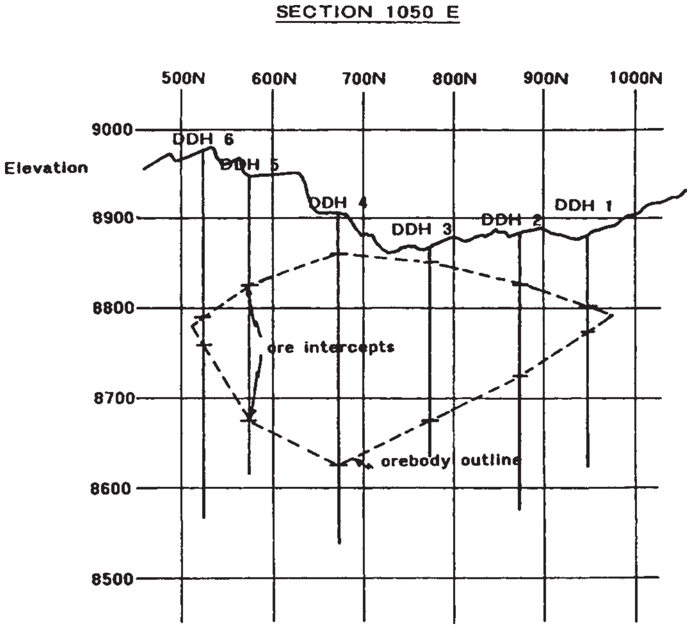


Figure 3.18. Addition of the ore zone to a section.

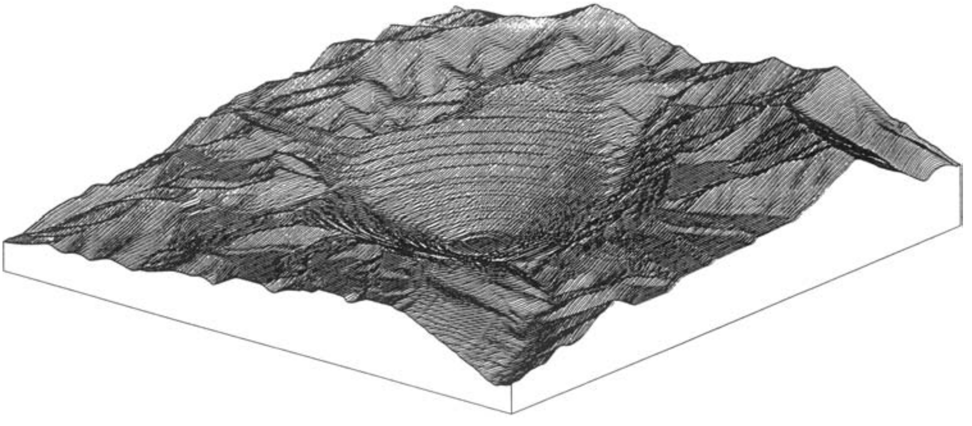


Figure 3.19. An isometric view of the Bingham Pit (Kennecott, 1966).

3.3 GEOLOGIC INFORMATION

Mining operations at any given mineral deposit may be divided into four stages. Prospecting (Stage 1), is the act of searching for valuable minerals. With the discovery of such minerals, the property becomes a mineral prospect. The property is then explored to gain some initial information regarding the size, shape, position, characteristics and value of the deposit. If this exploration stage (Stage 2) is successful, then the decision to proceed into Stage 3 (development) may be made. Detailed (Waterman & Hazen, 1968) geologic information must be collected and made available early in this stage to facilitate planning and design. The following points should be included:

1. Geology of the mineralized zone;
2. Physical size and shape of the deposit;
3. Quantitative data on grade and tons of material within pertinent cut-off limits;
4. Mineralogical and metallurgical characteristics of the ore;
5. Physical characteristics of the ore and waste; and
6. Data on ground conditions, groundwater and other factors that affect mine design and operation.

Stage 4 is the actual mining of the deposit.

Although trenching, the sinking of shafts, and the driving of tunnels are all sometimes used in the evaluation of a surface mining prospect, most often the initial and continuing collection of geologic information is accomplished through a drilling program. The various drilling methods and their characteristics are summarized in Table 3.2. Here the focus will be on diamond drilling which produces a core for logging and assaying. The two basic types of diamond core drilling are: (a) conventional and (b) wireline. In conventional drilling the core is retained by a core spring or core lifter in the core barrel which is located just behind the bit. When the core barrel is full or after drilling a certain length, the entire drill string must be removed in order to extract the core.

Table 3.2. Characteristics of exploratory/development drilling techniques (Peters, 1978).

	Diamond core	Rotary	Reverse circulation	Downhole rotary	Downhole hammer	Percussion	Churn
Geologic information	Good	Poor	Fair	Poor	Poor	Poor	Poor
Sample volume	Small	Large	Large	Large	Large	Small	Large
Minimum hole diameter	30 mm	50 mm	120 mm	50 mm	100 mm	40 mm	130 mm
Depth limit	3000 m	3000 m	1000 m	3000 m	300 m	100 m	1500 m
Speed	Low	High	High	High	High	High	Low
Wall contamination	Variable	Variable	Low	Variable	Variable	Variable	Variable
Penetration-broken or irregular ground	Poor	Fair	Fair	Fair	Good	Good	Good
Site, surface, and underground	S + U	S	S	S + U	S + U	S + U	S
Collar inclination, range from vertical and down	180°	30°	0° ↑	30°	180°	180°	0°
Deflection capability	Moderate	Moderate	None	High	None	None	None
Deviation from course	High	High	Little	Little	Little	High	Little
Drilling medium, air or liquid	L	A + L	L	A + L	A	A + L	L
Cost per unit depth	High	Low	Moderate	Low	Low	Low	High
Mobilization cost	Low	Variable	Variable	Variable	Variable	Low	Variable
Site preparation cost	Low	Variable	Variable	Variable	Variable	Low	High

↑ Reverse circulation has recently been used at inclinations up to 40°.

The wireline method uses a core barrel removable through the inside of the drill stem with a latching device on the end of a cable. With this method, core can be retrieved at any desired point. Due to the space taken up by the inner barrel, wireline cores are smaller than those obtained in conventional drilling for the same hole size. The most common core size is NX/NQ which means that for a nominal 3-inch diameter hole, a 1 $\frac{7}{8}$ inch diameter core is recovered. The cores are typically recovered over 5 to 10 ft intervals. Sometimes the core recovery is poor and the grade is obtained by analyzing the cuttings/sludge. The recovered core is placed in order in core boxes for study, transport and storage. Table 3.3 is an example of a drill hole log. The amount of core recovered has been noted and a description of the material provided for each interval. Representative samples of the core are selected and sent out for assaying. Table 3.4 shows the results for the core described in Table 3.3. Information of this type may be plotted directly on cross sections or entered into a data file for computer processing. Table 3.5 is an example of one such computer file.

Table 3.3. Drill hole log from the Comstock Mine (Blais, 1985).

Mine: Comstock | Driller | Machine | Hole No. 144
Level: Surface | Location: Iron County, Utah | Elev. 6246.50 | Date 6-11-53
Lat. 890.60 S | Dep. 2099.69 E | Angle Vert. | Bearing

Date	Drilled			Core		Material/Remarks
	From	To	ft	ft	in	
6-11-53D	0	5	5	2	0	Porphyry, plag. & K-feldspars about 1 to 1.5 mm long in gray, siliceous looking groundmass. Some euhedral biotite present. Rock somewhat altered.
	5	10	5	2	6	Porphyry, same as 0–5 except for 4″ hard, black, magnetite veinlet at 10′.
	10	16	6	1	0	Porphyry, same as 0–5
	16	26	10	8	0	Porphyry, highly argililized & bleached, very friable, some limonite staining.
	26	36	10	4	6	Porphyry, same as 16–26 except no limonite staining.
	36	46	10	10	0	Porphyry, same as 26–36
	46	56	10	1	6	Porphyry, same as 16–26
	56	66	10	5	0	Porphyry, same as 26–36
	66	71	5	1	6	2′ porphyry, same as 26–36 3′ siltstone, fine grained, light gray.
	71	75	4	1	6	Siltstone, very dense, fine grained, med gray color, occasional small cherty nodules.
	75	77	2	1	6	Siltstone, same as 71–75
	77	80	3	1	0	Siltstone, same as 71–75
	80	86	6	2	0	Siltstone, same as 71–75
	86	90	4	2	6	Siltstone, med grained, sandy, gray to purplish gray color.
	90	96	6	2	0	Siltstone, same as 71–75
	96	100	4	1	6	Siltstone, very dense, fine grained, dark gray with well developed joints dipping 45°, usually filled with calcite.

(Continued)

Table 3.3. (Continued).

100	106	6	1	4	Siltstone, same as 96–100
106	116	10	4	6	Siltstone, fine grained, slightly sandy, somewhat banded with bands of light gray, purplish gray, and greenish gray about 1/16″ to ¼″ thick joints are filled with calcite.
116	126	10	7	6	Siltstone, fine grained, dense, gray with many wide calcite filled joints
126	136	10	2	6	Siltstone, same as 116–126 except ¼″ magnetite vein as 126–127.
136	146	10	6	0	Siltstone, med grained, slightly sandy, gray with many calcite and magnetite veinlets and stringers at 136–140.
146	150	4	0	6	2′ siltstone, fine grained, greenish-gray, slightly altered. 2′ magnetite, black, soft, with much admixed silty material.
150	153	3	2	6	Magnetite, soft, sooty, black with many silty bands and streaks.
153	156	3	2	7	Magnetite, soft, sooty, black, with some streaks and bands of shiny black, med crystalline magnetite. Silty material present as streaks and tiny blobs.
156	161	5	4	4	Magnetite, same as 153–156
161	166	5	3	10	Magnetite, same as 153–156 except locally quite vuggy.
166	171	5	3	4	Magnetite, same as 153–156
171	176	5	4	6	Magnetite, same as 153–156 except silty material is green.
176	181	5	5	0	Magnetite, same as 153–156 except many calcite streaks and blobs from 177–178.
181	183	2	1	6	Magnetite, fine grained, soft to med hard, dark gray, very little silt.
183	188	5	4	9	Magnetite, same as 181–183.
188	189½	1½	1	6	Magnetite, same as 181–183.
189½	195	5½	3	4	Magnetite, same as 181–183 except some med crystalline, shiny black magnetite.
195	201	6	6	0	Magnetite, same as 189½–195 except much pyrite.
201	203	2	0	11	Magnetite, fine grained, soft, sooty, black with some med grained shiny black magnetite.
203	206	3	2	6	Limestone, very fine grained, dense, bluish gray with some magnetite as disseminated fine grains and veinlets.
206	211	5	4	10	Limestone, same as 202–206 except no magnetite.
					Bottom of hole

In addition to the type of information needed to compute the grade and tonnage, rock structural data are important for pit slope design. A simplified and a more comprehensive data sheet used in the logging of structural information are given in Tables 3.6 and 3.7 respectively. These data are the foundations for the planning and design steps. The information represented by a single hole is extended to a rather large region including the hole. Thus mistakes in evaluation, poor drilling practices, poor core recovery, sloppy record keeping, etc. may have very serious consequences. As soon as these data are entered into the computer or onto

Table 3.4. Drill sample analysis from the drill hole in Table 3.3 (Blais, 1985).

Mine: Comstock — Driller — Machine — Hole No. 144
Depth 211′ — Core identified by P. Kalish — Date 6/11/53
Core recovery % — Analysis by — Date 2/23/53

Interval		Sample No.		Fe	Mn	SiO_2	Al_2O_3	P	S	CaO	MgO	Insol.	R_2O_3
From	To	Core	Sludge										
150	153	6177		50.7		12.0	3.8	0.270	0.03	2.5	1.7		
153	156	6178		51.8		8.1	2.7	0.280	0.03	3.7	2.0		
156	161	6179		55.3		9.1	2.8	0.280	0.04	1.5	1.2		
161	166	6180		56.0		10.2	2.5	0.216	0.05	1.8	1.0		
166	171	6181		58.5		8.8	1.9	0.058	0.03	1.2	1.0		
171	176	6182		53.6		9.6	3.0	0.148	0.03	2.0	1.6		
176	181	6183		56.6		8.8	2.7	0.206	0.02	1.5	1.0		
181	183	6184		51.2		12.0	2.5	0.020	0.02	2.6	2.1		
183	188	6185		54.7		8.0	2.3	0.170	0.03	1.8	1.6		
183	189½	6186		55.0		5.9	1.8	0.025	0.03	1.3	2.3		
189½	195	6187		56.3		4.9	1.7	0.057	0.03	1.3	2.3		
195	201	6188		54.6		6.9	1.9	0.033	0.67	1.1	2.5		
201	203	6189		54.2		7.6	2.1	0.027	0.61	1.3	1.7		
153	156		6190	46.8		13.2	4.4	0.075	0.03	1.6	1.0		
156	161		6191	52.4		9.9	3.1	0.130	0.02	2.5	1.3		
161	166		6192	54.4		8.9	2.0	0.318	0.02	2.0	1.1		
166	171		6193	57.0		7.2	2.4	0.116	0.02	1.9	1.5		
189½	195		6194	53.8		10.2	2.9	0.138	0.06	1.7	1.5		

sections and plans, the level or uncertainty associated with them vanishes. Good numbers and less good numbers all carry the same weight at that stage. Hence it is imperative that utmost care be exercised at this early stage to provide a thorough and accurate evaluation of all information. Each hole is quite expensive and there is pressure to keep them to a minimum. On the other hand, poor decisions based on inadequate data are also expensive. This weighing of real costs versus project benefits is not easy.

3.4 COMPOSITING AND TONNAGE FACTOR CALCULATIONS

3.4.1 *Compositing*

As discussed in the previous section, after the diamond core has been extracted it is logged by the geologist and representative samples are sent out for assaying. Upon receipt, the assays are added to the other collected information. These individual assay values may represent core lengths of a few inches up to many feet. Compositing is a technique by which these assay data are combined to form weighted average or composite grades representative of intervals longer than their own. The drill log shown diagrammatically in Figure 3.20 contains a series of ore lengths l_i and corresponding grades g_i.

In this case the boundaries between ore and waste are assumed sharp. The first question which might be asked is ‘What is the average grade for this ore intersection?’ The weighted average is found by first tabulating the individual lengths l_i and their corresponding grades g_i.

Table 3.5. Typical computerized drill hole data file, after Stanley (1979).

								Page No.	1
								Run date:	03/03/77
Report No: 01			Collar coordinates					Run time:	14:30:06
Site	Hole	Type	East	North	Elevation	Azimuth	Inclination	Interval	Hole depth	Remarks
HU	0002	D	08054	05796	05509	+0	090	010	00500	none

Seq.	Coordinates		Elevation	Distance					Percent	Max	PC	RK	R	RK	Alteration	Mineral
No	East	North			AZ1	INC	INT	MOS2	WO3	LNG	CR	QD	C	TP	PSSACFT	MFPFU
001	8054	05796	05504	5	+0	90	10	0.382	NA	0.4	37	54	6	U1	332405	0000
002	8054	05796	05494	15	+0	90	10	0.305	0.004	0.3	48	4	6	U1	332405	000R
003	0854	05796	05484	25	+0	90	10	0.246	0.002	0.2	59	0	5	U1	342406	0000
004	8054	05796	05474	35	+0	90	10	0.257	0.002	0.2	69	4	4	U1	232505	0000
005	8054	05796	05464	45	+0	90	10	0.229	0.002	0.4	60	4	5	U1	232505	0000
006	8054	05796	05454	55	+0	90	10	0.411	0.001	0.7	48	25	6	U1	132505	000R
007	8054	05796	05444	65	+0	90	10	0.277	0.004	0.7	38	18	7	U1	132304	P000
008	8054	05796	05434	75	+0	90	10	0.400	0.003	0.7	35	42	7	U1	132304	000R
009	8054	05796	05424	85	+0	90	10	0.287	0.001	0.5	42	12	6	U1	131304	000F
010	8053	05796	05414	95	−1	90	10	0.283	0.002	0.9	32	61	7	U1	132303	000R
011	8053	05796	05404	105	−1	90	10	0.290	NL	0.6	60	18	5	U1	122403	000F
012	8053	05796	05394	115	−1	90	10	0.504	NL	0.6	38	7	7	U1	144304	0000
013	8053	05796	05384	125	−1	90	10	0.286	0.002	0.9	26	76	7	U1	123306	0000
014	8053	05796	05374	135	−1	90	10	0.390	0.002	0.5	65	30	5	U1	122303	0000
015	8052	05796	05364	145	−1	90	10	0.545	0.002	0.8	44	42	6	U1	232204	000B
016	8052	05796	05354	155	−2	90	10	0.429	0.001	0.4	57	26	5	U1	232303	000F
017	8052	05796	05344	165	−2	90	10	0.346	NL	0.9	55	27	6	U1	123203	000R
018	8051	05796	05334	175	−2	90	10	0.253	NL	0.6	68	5	6	U1	122203	0000
019	8051	05796	05324	185	−2	90	10	0.374	NL	0.7	50	34	7	U1	122303	0000
020	8051	05796	05314	195	−2	90	10	0.248	NL	0.9	65	21	5	U1	332303	0000
021	8050	05796	05304	205	−2	90	10	0.483	NL	0.7	63	13	6	U1	253303	0000

Table 3.6. Example data collection form for core recovery and RQD (Call, 1979).

RQD Data sheet

Page ——of——

Hole number ________ Collar elev. ________ By ________

Coordinates ________ Core box length ________ Date ________

________ Core diameter ________

Scale	Interval		Recovery	RQD ( )			Longest	Rock type	Altera-tion
	From	To		+1 in.	+4 in.	+1 ft.	( )		

Rock type abbreviations				Alteration abbreviations			

Table 3.7. Example data collection form for oriented core (Call, 1979).

Data sheet for core structure

Hole no. ________ Location ________ Date ________ By ________

Collar elev. ________ Inclination ________ Bearing ________ Diam. ________

Depth (ft)	Rock Type 1	Rock Type 2	Structure Type	Structure Inclin. to datum	Structure App. Dip.	Geometry P	Geometry C	Geometry R	Thickness (ft)	Filling	τ	Comments

Rock type abbreviations		Structure type				Geometry	
		SJ	SJ	BZ	Broken zone	P-Planarity -P.W.1	C-continuity-C.D
		JS	Joint set	C	Contact	MD-Minimum dip	R-Roughness-S.R
		FT	Fault	BX	Breccia		
		SZ	Shear zone				
Water D-Dry W-Wet F-Flowing S-Squirting							

Filling abbreviations				τ Shear strengths	MD Medium
N	None	Q	Quartz or silicate	Vl Very low	H High
O	Oxide	C	Clay	LW Low	VH Very high
S	Sulfide	G	Gouge		

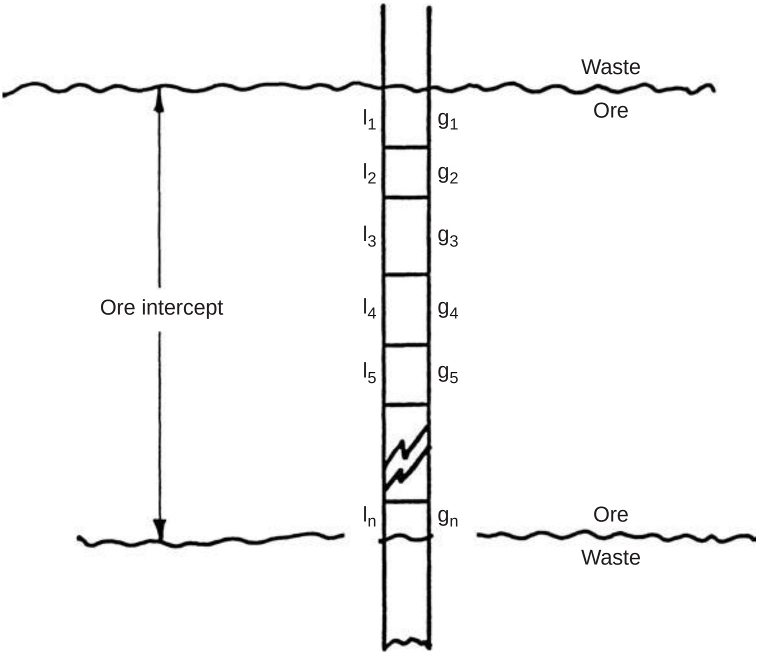


Figure 3.20. Ore intercept compositing.

The products $l_i g_i$ are formed and summed $\sum l_i g_i$. This sum is then divided by the sum of the lengths $\sum l_i$ to yield the desired grade. This is written out below:

Length	Grade	Length × Grade
l_1	g_1	$l_1 g_1$
l_2	g_2	$l_2 g_2$
l_3	g_3	$l_3 g_3$
.	.	.
.	.	.
.	.	.
l_n	g_n	$l_n g_n$
$\sum l_i$	$\boxed{\bar{g}}$	$\sum l_i g_i$

The average grade is

$$\bar{g} = \frac{\sum l_i g_i}{\sum l_i} \tag{3.3}$$

This value would then be filled into the box on the table. In this case $\bar{g}$ is called the *ore-zone composite*. Although compositing is usually a length-weighted average, if the density is extremely variable, the weighting factor used is the length times the density (or the specific gravity).

This procedure is repeated for each of the holes. Note that each ore intercept would, in general, be of a different length. The top and bottom elevations would also be different.

For large, uniform deposits where the transition from ore to waste is gradual (the cut-off is economic rather than physical) the compositing interval is the bench height and fixed

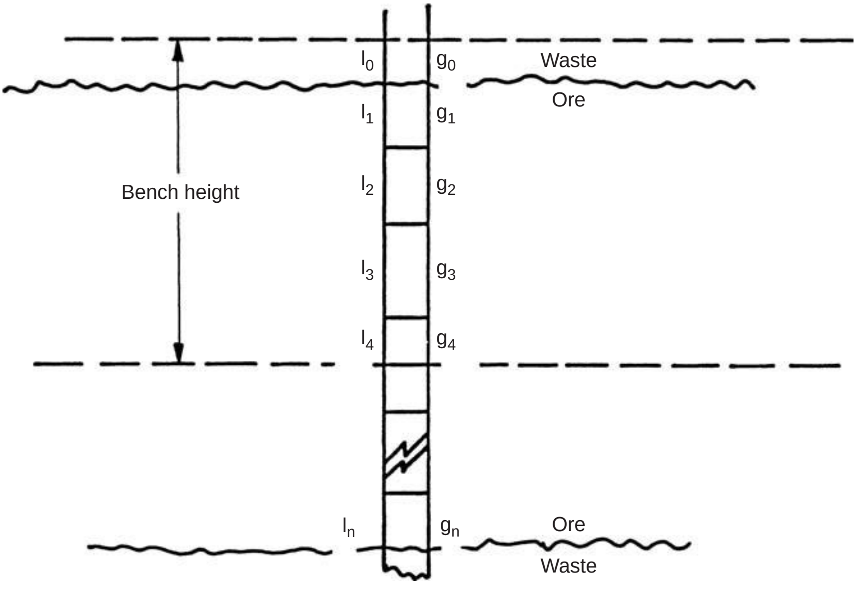


Figure 3.21. Bench height compositing.

elevations are selected. This *bench compositing* is the method most often used for resource modeling in open pit mining today. In Figure 3.21, a bench is shown by the horizontal dashed lines.

In this particular case, the upper portion of the bench lies in 'waste.' The bench composite grade is

Length	Grade	Length × Grade
l_0	g_0	$l_0 g_0$
l_1	g_1	$l_1 g_1$
l_2	g_2	$l_2 g_2$
l_3	g_3	$l_3 g_3$
l_4	g_4	$l_4^* g_4$
$\sum l_i$	$\boxed{\bar{g}}$	$\sum l_i g_i$
$\sum l_i = H$		

where H is bench height. Hence

$$\bar{g} = \frac{\sum l_i g_i}{H}$$

Compositing with fixed intervals and elevations makes it very easy to present and analyze the results for a deposit containing a number of drill holes.

Some of the reasons for and the benefits of compositing include:

1. Irregular length assay samples must be composited to provide representative data for analysis.

2. Compositing incorporates dilution such as that from mining constant height benches in an open pit mine.

3. Compositing reduces erratic variation due to very high or very low assay values.

4. By compositing, the number of data, and hence the required computational times, are reduced.

Table 3.8. Example of drill-hole log, after (Davey, 1979).

Drill-hole ID = C-22
Collar location = 1800.0 N 800.0 E
Elevation = 45198.0
Azimuth = 0.0 Attitude = −90

Depth	Assay
5	0.400
10	0.560
15	0.440
20	0.480
25	0.400
30	0.380
35	0.330
40	0.590
45	0.480
50	0.600
55	0.560
60	0.320
65	0.700
70	0.210
75	0.180
80	0.080
85	0.200
90	0.070

To illustrate the principles presented, consider the simplified drill hole log (Davey, 1979) given in Table 3.8.

It has been decided that 40 ft high benches and a 5200 ft reference elevation will be used. This means that bench crest elevations would be at 5200 ft, 5160 ft, 5120 ft, etc. The upper 38 ft of hole C-22 would lie in bench 1. The next 40 ft would be in bench 2 and the hole would terminate in bench 3. Using the procedure outlined above, the composite grade at this hole location for bench 2 is determined as:

Length (ft)	Grade (%)	Length × Grade (ft%)
2	0.590	1.18
5	0.480	2.40
5	0.600	3.00
5	0.560	2.80
5	0.320	1.60
5	0.700	3.50
5	0.210	1.05
5	0.180	0.90
3	0.080	0.24
40	0.417	16.67

$$\bar{g} = \frac{16.67}{40} = 0.417$$

The mid-elevation of the bench 2 is 5140.0 ft. Composites of the remaining portions of the drill hole lying above and below this bench may be found in the same way. The results are given below.

Bench	Center coordinates			Grade
	E	N	Elevation	
1	800.00	1800.00	5179.00	0.440
2	800.00	1800.00	5140.00	0.417
3	800.00	1800.00	5114.00	0.126

If material running 0.3% and higher is understood to be ore, then the ore-zone at this hole extends from the surface to a depth of 65 ft. The ore-zone composite would be:

Length (ft)	Grade (%)	Length × Grade (ft%)
5	0.40	2.00
5	0.56	2.80
5	0.44	2.20
5	0.48	2.40
5	0.40	2.00
5	0.38	1.90
5	0.33	1.65
5	0.59	2.95
5	0.48	2.40
5	0.60	3.00
5	0.56	2.80
5	0.32	1.60
5	0.70	3.50
65	0.48	31.20

$$\bar{g} = \frac{31.2}{65} = 0.48$$

In this case when the lengths are all equal, the average grade is just the simple average of the grades.

$$\bar{g} = \frac{6.24}{13} = 0.48$$

The same compositing technique can be used when dealing with grades representing different areas or volumes. This will be demonstrated in Section 3.5.

3.4.2 *Tonnage factors*

In mining, although *volumes* of material are removed, payment is normally received on the basis of the *weight* of the valuable material contained. This is in contrast to civil construction projects where normally payment is received based simply upon the material volume removed or emplaced. Even here, however, the conversion from volume to weight must often be made due to the lifting and carrying limitations of the loading and hauling equipment used. The conversion from volume V to weight W and vice versa is done in the English system of units with the help of a tonnage factor TF (volume/weight):

$$V = \mathrm{TF} \times W \tag{3.4}$$

where TF is the tonnage factor (volume/weight), V is the volume, and W is the weight. The determination of a representative factor(s) is quite important to mining operations.

In the English system of measurement, the basic unit for describing the weight of materials is the weight of a cubic foot of water. The density W_D of water is

$$W_D(\mathrm{H_2O}) = 62.4\,\mathrm{lb/ft^3} \tag{3.5}$$

and its specific gravity SG is 1. If the mined material has a specific gravity of 2.5, its weight density is

$$W_D = \mathrm{SG} \times W_D(\mathrm{H_2O}) = 2.5 \times 62.4\,\mathrm{lb/ft^3} = 156\,\mathrm{lb/ft^3} \tag{3.6}$$

The tonnage factor TF for the material (assuming that the short ton (st) applies) is

$$\mathrm{TF} = \frac{2000\,\mathrm{lb/st}}{156\,\mathrm{lb/ft^3}} = 12.82\,\mathrm{ft^3/st} \tag{3.7}$$

In the metric system, the density of water is

$$W_D(\mathrm{H_2O}) = 1\,\mathrm{g/cm^3} = 1000\,\mathrm{kg/m^3} = 1\,\mathrm{t/m^3} \tag{3.8}$$

Since the specific gravity of the mined material is 2.5, the density is $2.5\,\mathrm{t/m^3}$. The tonnage factor is

$$\mathrm{TF} = \frac{1}{2.5} = 0.4\,\mathrm{m^3/t} \tag{3.9}$$

Although the tonnage factor as defined here with units of volume per weight is probably the most commonly used, the inverse (TF*) is also used:

$$\mathrm{TF}^* = \frac{W}{V}$$

Other units such as $\mathrm{yd^3}$ instead of $\mathrm{ft^3}$ are sometimes used for convenience.

Although simple in principle, it is not as easy in practice to determine the appropriate material densities to be used in the calculations. There can be many different materials involved in an open pit mine and each 'material' can vary in density from point to point.

Three techniques are available for determining material density:

1. Density testing of small samples in the laboratory.
2. Careful excavation and weighing of a large volume.
3. Calculation based upon composition (mineralogy) using published densities such as given in Tables 3.9 through 3.11.

Depending upon the requirements, all three are sometimes used. For Technique 1, there are two primary tests which are done. In the first, the sample is first weighed (W) in air. The sample volume V is then determined by water displacement (the water level in a graduated cylinder is, for example, compared before immersion and after immersion of the sample).

Table 3.9. Average density of minerals (Westerfelt, 1961).

Material	Mineral	Density (g/cm^3)	Material	Mineral	Density (g/cm^3)
Antimony	Native	6.7	Iron	Arsenopyrite	6.0
	Stibnite	4.6	(continued)	Hematite	5.0
Arsenic	Orpiment	3.5		Magnetite	5.0
	Realgar	3.5		Limonite	3.8
Barium	Barite	4.5		Siderite	3.8
	Witherite	4.3	Lead	Galena	7.5
Calcium	Calcite	2.7		Cerussite	6.5
	Aragonite	3.0		Anglesite	6.3
	Gypsum	2.3		Crocoite	6.0
	Fluorspar	3.2		Pyromorphite	7.0
	Apatite	3.2	Manganese	Pyrolusite	4.8
Coal	Anthracite	1.5		Psilomelane	4.2
	Bituminous	1.3		Rhodochrosite	3.6
Cobalt	Linnaite	4.9		Rhodonite	3.6
	Smaltite	6.5	Mercury	Native	14.4
	Cobaltite	6.2		Cinnabar	8.1
	Erythrite	3.0	Molybdenum	Molybdenite	4.7
Copper	Native	8.9	Nickel	Millenite	5.6
	Chalcocite	5.7		Niccolite	7.5
	Chalcopyrite	4.2	Platinum	Native	17.5
	Bornite	5.0	Silver	Native	10.5
	Enargite	4.4		Argentite	7.3
	Tetrahedrite	4.9		Sylvanite	8.0
	Atacamite	3.8		Pyrargyrite	5.8
	Cuprite	6.0		Cerargyrite	5.4
	Chalcanthite	2.2	Sulphur	Native	2.1
	Malachite	3.9	Tin	Cassiterite	6.8
	Azurite	3.7		Stannite	4.5
	Chrysocolla	2.2	Tungsten	Wolframite	7.3
	Dioptase	3.3		Scheelite	6.0
Gold	Native	19.0	Zinc	Blende	4.0
Iron	Pyrite	5.1		Zincite	5.7
	Marcasite	4.8		Smithsonite	4.4
	Pyrrhotite	4.6			

The density d is then calculated:

$$d = \frac{W}{V} \tag{3.10}$$

In the second type of test, the sample is first weighed (W) in air and then weighed (S) when suspended in water. The specific gravity is

$$\mathrm{SG} = \frac{W}{W - S} \tag{3.11}$$

Care must be taken to correct both for porosity and moisture.

Technique 2 is the most expensive and time consuming, but provides the best site specific results. Such tests would have to be made for different locations in the mine.

Table 3.10. Average density of some common rock types (Reich, 1961).

Origin	Rock type	Density (g/cm^3)	Origin	Rock type	Density (g/cm^3)
1. Igneous (plutonic)	Nepheline syenite	2.62	3. Metamorphic (continued)	Phyllite	2.74
	Granite	2.65		Marble	2.78
	Quartz	2.65		Chlorite schist	2.87
	Anorthosite	2.73		Serpentine	2.95
	Syenite	2.74	4. Sedimentary (consolidated)	Greywacke	2.69
	Quartz diorite	2.79		Sandstone	2.65
	Diorite	2.93		Limestone	2.73
	Gabbro	3.00		Argillaceous shale	2.78
	Peridotite	3.06		Calcareous shale	2.67
	Pyroxene	3.22		Chert	2.76
2. Igneous (hypabasal/ volcanic)	Quartz porphyry	2.63	5. Sedimentary (unconsolidated)	Humus soil	1.45
	Porphyry	2.67		Surface soil	1.73
	Diabase	2.94		Clayey sand/sandy	1.93
	Rhyolite	2.50		Gravel, very damp	2.00
	Phonolite	2.56		Dry, loose soil	1.13
	Trachyte	2.58		Very fine, sandy	1.33
	Dacite	2.59		alluvium	1.51
	Andesite	2.62		Carbonaceous loam	1.65
	Basalt	2.90		Glayey, sandy soil	2.25
3. Metamorphic	Orthoclase gneiss	2.70		Very wet, quartz sand	
	Plagioclase gneiss	2.84			
	Quartz schist	2.68		Loess	2.64
	Mica schist	2.73		Clay	2.58

Table 3.11. Density of some metals (CRC Handbook of Chemistry and Physics, 1991–1992).

Metal	Density (g/cm^3)
Chromium	6.92
Copper	8.89
Gold	19.3
Iron	7.86
Lead	11.34
Molybdenum	9.0
Nickel	8.6
Platinum	21.37
Silver	10.5
Zinc	7.13

To illustrate the use of Technique 3, consider a gold ore made up of 94% quartz and 6% iron pyrite by weight. From Tables 3.9 and 3.10 one finds that the respective specific gravities are:

- Quartz: 2.65
- Iron pyrite: 5.1

The overall specific gravity for the ore is

$$\mathrm{SG} = 2.65 \times 0.94 + 5.1 \times 0.06 = 2.80$$

and the tonnage factor (English system) is

$$\mathrm{TF} = \frac{2000}{2.80 \times 62.4} = 11.45\,\mathrm{ft}^3/\mathrm{st}$$

A margin of safety is introduced when applying the results of any of the techniques. In this case a value of 12 or even greater might be used. This is the in-situ or in-place tonnage factor.

To illustrate the principles involved in the conversion from volumes to weight and vice versa assume that a mining company has a contract to sell 5000 tons of metal X per year. The mined material contains 1% of the contained metal and the processing plant recovers 50%. The total tonnage T_A which must be mined and processed each year is given by

$$T_A = \frac{5000\,\mathrm{st}}{0.01 \times 0.50} = 1{,}000{,}000\,\mathrm{st}$$

Assuming that the layer being mined has a thickness t of 20 ft, the question becomes how large a plan area A must be exposed to produce the required tonnage.

The annual volume V_A is

$$V_A = tA$$

To solve the problem, the relationship between the volume V_A and the weight T_A must be known. Assuming that the specific gravity of the mined material is 2.5, the tonnage factor is 12.82 ft^3/st.

Hence, the volume removed per year is

$$V_A = 12.82\,\mathrm{ft}^3/\mathrm{st} \times 1{,}000{,}000\,\mathrm{st} = 12{,}820{,}000\,\mathrm{ft}^3$$

Hence, the area to be exposed is

$$A = \frac{12{,}820{,}000}{20} = 641{,}000\,\mathrm{ft}^2$$

The acre is commonly used to describe land area:

$$1\,\mathrm{acre} = 43{,}560\,\mathrm{ft}^2$$

Thus, a total of 14.72 acres would be mined each year.

The same problem will now be worked using the metric system. It is assumed that 4537 t of mineral are produced from a seam 6.1 m thick. The numerical value of the density and the specific gravity are the same in this system, which simplifies the calculations.

Since the specific gravity of the mined material is 2.5, the density is 2.5 t/m^3. The tonnage factor is

$$\mathrm{TF} = \frac{1}{2.5} = 0.4\,\mathrm{m}^3/\mathrm{t}$$

Therefore

$$V_A = \frac{4{,}537}{0.5 \times 0.01} \times 0.4 = 362{,}960\,\mathrm{m}^3$$

$$A = 59{,}502\,\mathrm{m}^2$$

In the metric system, land area is expressed in terms of the hectare:

$$1\,\text{hectare} = 100\,\text{m} \times 100\,\text{m} = 10,000\,\text{m}^2$$

Thus a total of 5.95 hectares would be mined each year.

3.5 METHOD OF VERTICAL SECTIONS

3.5.1 *Introduction*

The traditional method for estimating ore reserves has been through the use of sections. The method has a number of advantages, the primary one is that it can be done by hand. Other advantages are that it can be easily depicted, understood and checked. It will be assumed that the method is done by hand. However, a number of computer techniques are available to allow designer input/flexibility while doing the calculations by machine. Some computer programs have been designed to essentially reproduce the interpretation logic currently done by engineers and geologists by hand.

3.5.2 *Procedures*

The general procedures described below have been used by the Office of Ore Estimation-University of Minnesota (Weaton, 1972, 1973) for preparing and/or reviewing iron ore reserve estimates for the State of Minnesota. They can easily be adapted to other types of mineralization and deposits.

Planning materials
1. A current, up-to-date plan map. This is made to a convenient scale (usually 1 in = 100 ft) and shows the following:
 (a) Pit surface conditions, existing banks and details of the immediate vicinity.
 (b) Location of all drill holes.
 (c) Location of all quarter section lines and property lines.
 (d) Location of pit cross sections.
2. A complete set of cross sections. These are drawn to any convenient scale (usually 40 feet to the inch) and contain the following:
 (a) All exploration drill holes which fall on or close to the section (half way to the next section), with the detailed analysis of each sample taken. Results of hand wash or heavy density tests if they were made. Also, the location and analysis of any bank samples that have been taken.
 (b) A line showing the current top of the material remaining in the ground undisturbed.
 (c) Geologic structure lines showing an interpretation of the limits of the ore areas, and the various lean ore (low grade) or waste formations.

Planning procedures
1. Drill samples are evaluated on the cross sections and zones of different types of material are color coded for convenience. If the pit has been operating, any pit operations or observations which may disprove drilling samples in any way are taken into consideration in outlining zones of the various types of materials.

2. Limits of ore materials are transposed to the plan map as a general outline for the pit area.

3. The pit plan layout is developed to recover all of the ore that is economically minable with the necessary removal of the waste materials. Many factors enter into this plan, and govern the amount of material which must be removed. Some of these are:

(a) The nature of the surface capping; i.e., sand, clay, gravel, muskeg, etc., and the angle at which this material will remain stable in the bank.

(b) The nature of the rock and waste material and the angle at which it will remain stable when exposed.

(c) The local terrain and the location of the mine facilities, plant and dump areas in relation to the pit.

(d) The grade or steepness of the haulage road and the width required by the haulage trucks.

(e) The number of berms or protective benches that will be required to insure pit safety and bank stability.

4. After the pit plan is laid out, and the bank slopes are drawn on the cross sections, tonnages can be computed.

5. Unless disproven by other drilling or samplings, the material on each cross section is assumed to extend to a point one half the distance to the section on each side or 100 feet beyond the end section.

6. Computation of volumes in cubic feet are made by measuring the area of each type of material as shown on the cross section, and multiplying this by the distance represented by the section (one-half the distance to each adjacent section). By experience, the factor of cubic feet per ton has been established, for both ore and other materials. Concentration tests on drill samples of materials requiring plant treatment establish the recovery figure or how much concentrate will remain after being run through a concentration plant.

7. Tonnages of each section are totalled to give the final reserve tonnage figures.

8. A weighted average of the chemical analyses of each type of material is computed to produce the final estimated grade of the products included in the estimate.

3.5.3 *Construction of a cross-section*

An E-W section (640 N) taken through an iron deposit is shown in Figure 3.22. The objective is to begin with the drill hole data and proceed through to the determination of the areas of the different materials which would be included in the final pit. The symbols which have been used to denote the layers are:

SU = surface (overburden) material (soil, glacial till, etc.) which can be removed without drilling and blasting.

DT = decomposed taconite.

OP = ore and paint rock.

OT = ore and taconite.

SWT = sandy wash ore and taconite.

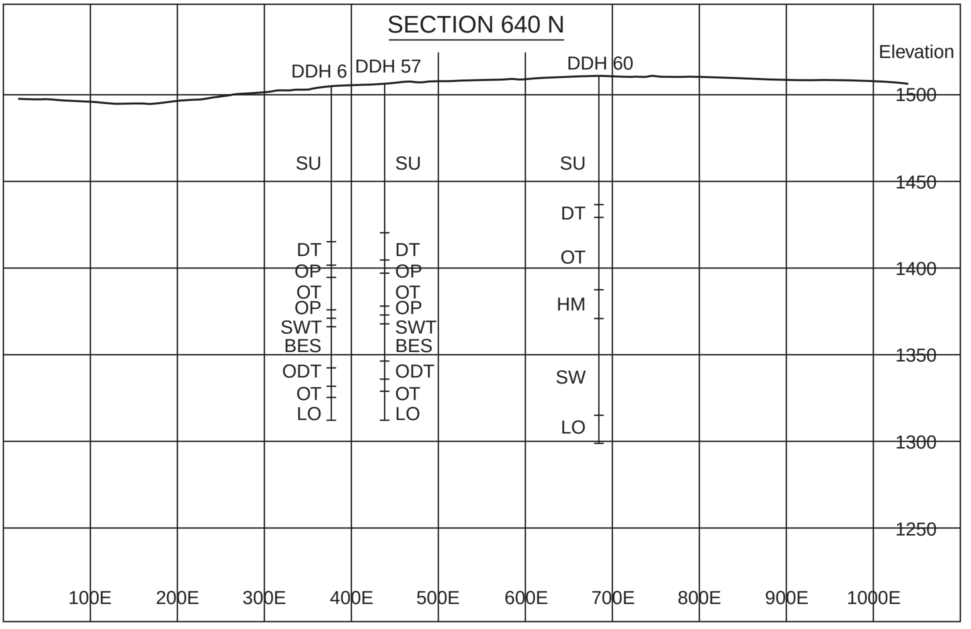


Figure 3.22. Drill holes and topography added to the section (Step 1).

BES = Bessemer ore.
ODT = ore and decomposed taconite.
LO = lean (low grade) ore.
SW = sandy wash ore.
HM = heavy media ore.

Step 1. The drill holes and surface topography are plotted on the section.

Step 2. The bisector between DDH57 and DDH60 is constructed. The surface-rock interface is drawn. Points common to all 3 holes are connected (Fig. 3.23).

Step 3. Starting from the surface, connect the remaining common points in holes DDH57 and DDH6 and extend them to the left of DDH6. To the right of DDH57 extend the layers over to the bisector line. These are drawn parallel to the known overlying surfaces. Fill in the region between DDH57 and the bisector by extending the layers parallel to the known overlying trend lines. (Fig. 3.24).

Step 4. The remaining layers intersected by DDH60 are extended left to the bisector and to the right. (Fig. 3.25).

Step 5. The pit outline is superimposed on the section. In this case the following rules have been used:

– the lean ore intercept forms the pit bottom,
– an extension of 50 ft outside of the drill holes at the pit bottom is assumed,
– the allowable pit slope angle in the surface material is 27° whereas in the rock layers near the pit bottom it is 54°. A transition of 41° is used between these. (Fig. 3.26).

Step 6. An access road 50 ft in width crosses this section at the position indicated (Fig. 3.27).

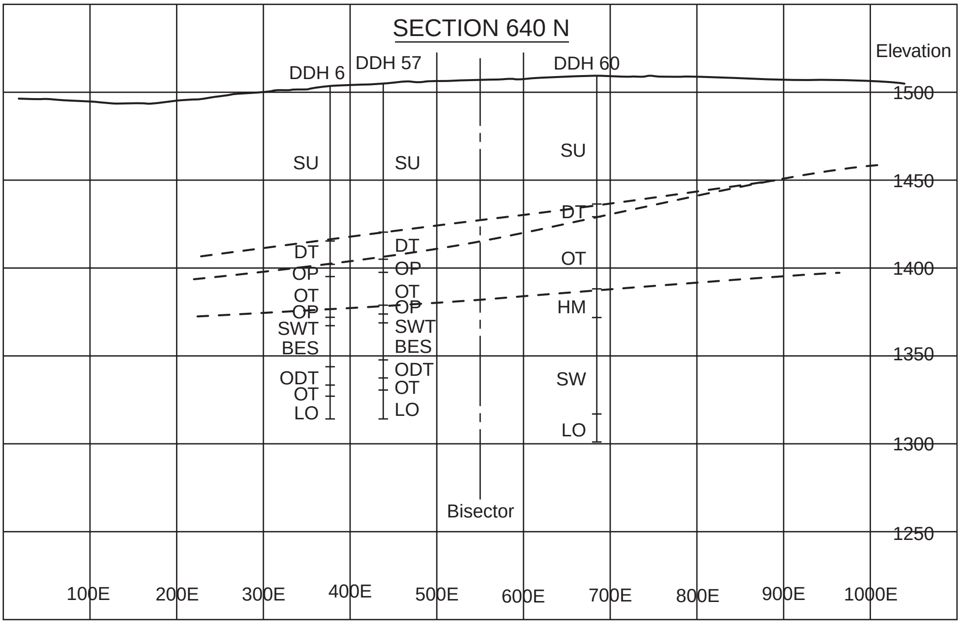


Figure 3.23. The Step 2 section.

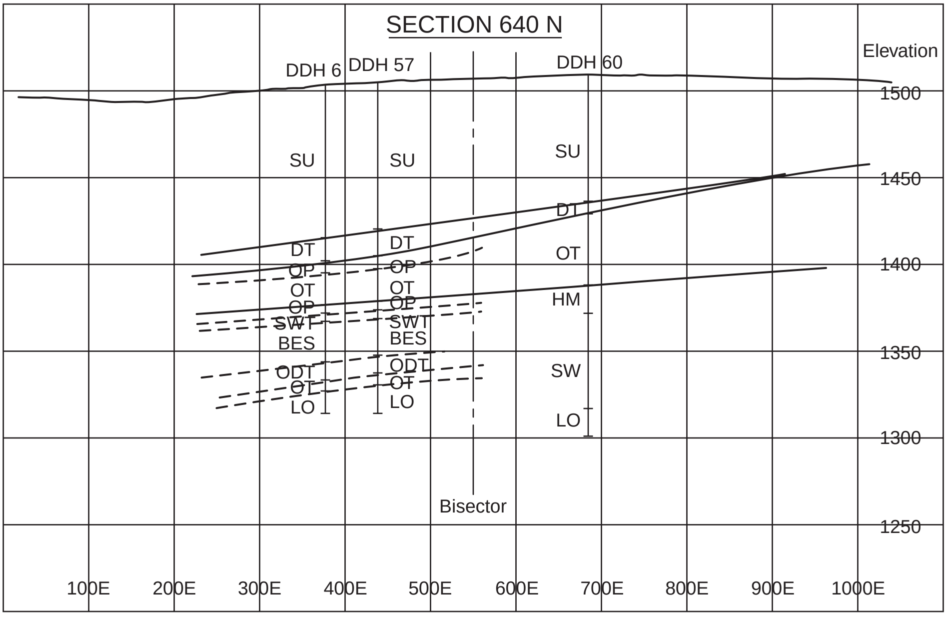


Figure 3.24. The Step 3 section.

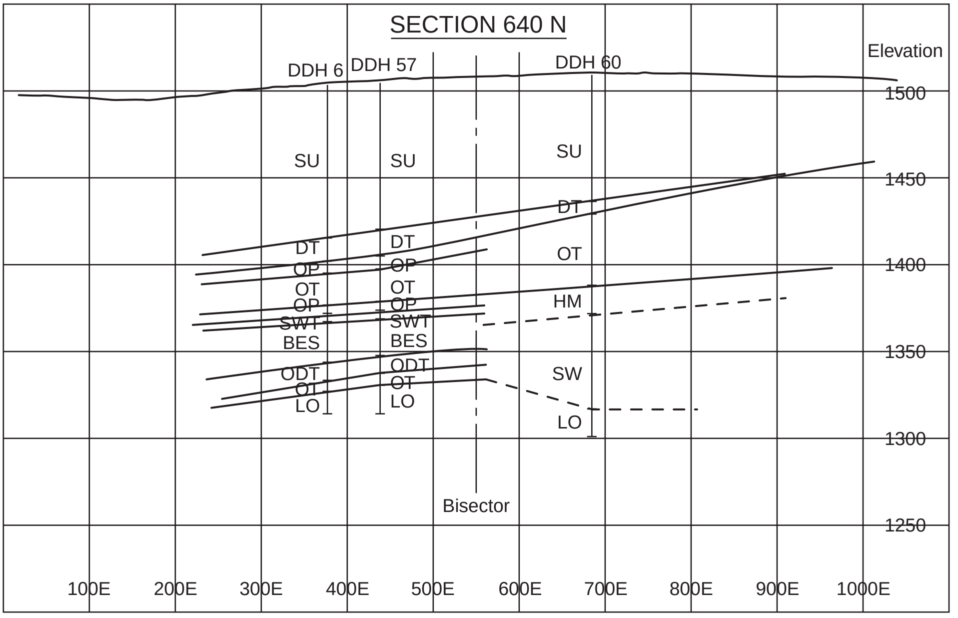


Figure 3.25. The Step 4 section.

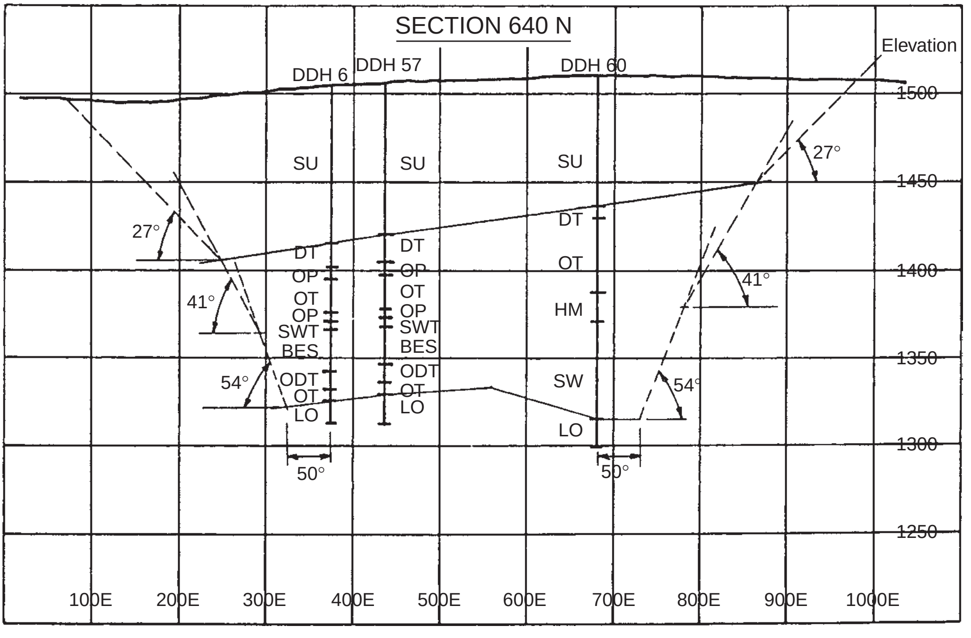


Figure 3.26. The Step 5 section.

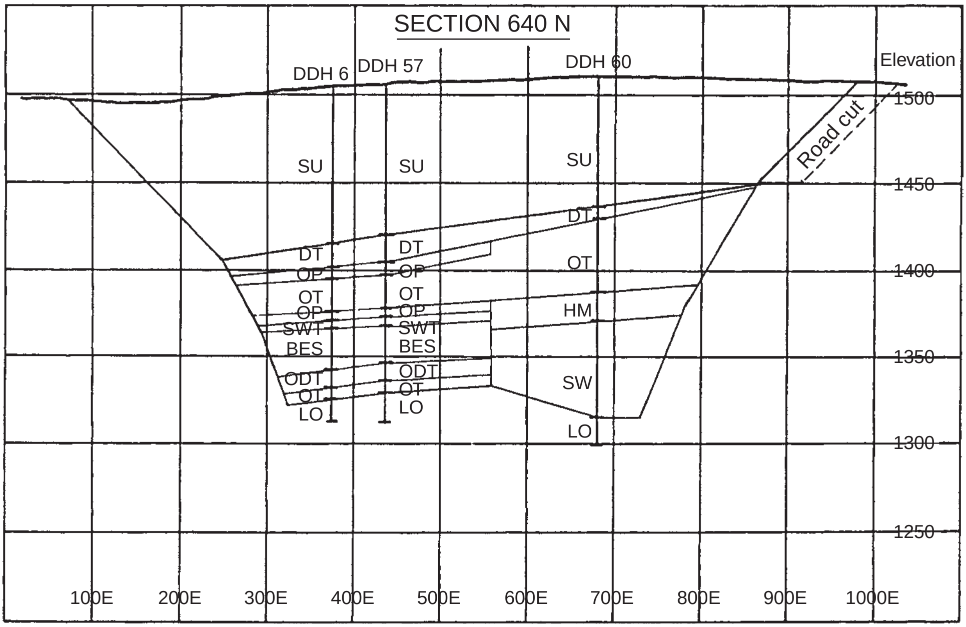


Figure 3.27. The Step 6 section.

Step 7. The areas of the different materials on the section are determined using a planimeter (Fig. 3.28).

Even in this relatively simple case, it is clear that a good knowledge of the structure of the orebody is required (Fig. 3.29) as well as some judgement in order to create such sections. As will be discussed in more detail later, final pit outlines can only be determined by considering all sections together.

3.5.4 *Calculation of tonnage and average grade for a pit*

This simplified example has also been taken from iron mining practice. The following concepts will be illustrated:

1. Side completion for sections.
2. Development of a final pit outline including pit ends.
3. Determination of tons and average grade for a section.
4. Determination of tons and average grade for the pit.

Although most of the discussion will revolve around section $1+00$, the same approach would be used on all sections.

Side completion

As described in the previous example, the section $1+00$ (Fig. 3.30) has been extended 50 ft past the positions of the outermost drill holes. On the left side of the section, the ore appears to pinch out within this zone. The pit slope of 27° has been drawn to pass through the mid-height of this extension. The width associated with hole 6 would be 50 ft plus half the distance between holes 6 and 1. On the right-hand side, the ore is quite thick (25 ft) and

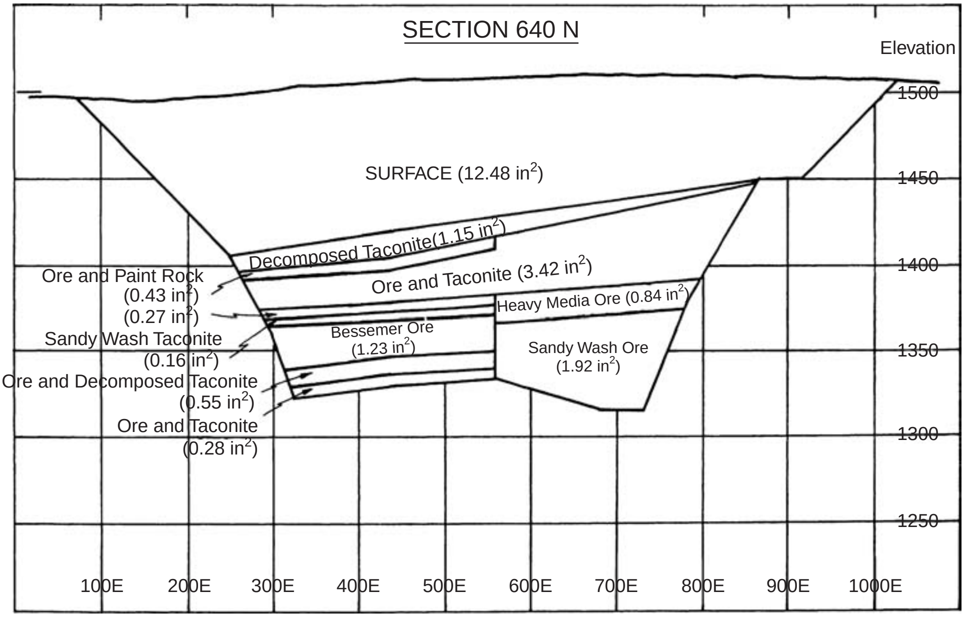


Figure 3.28. The Step 7 (final) section.

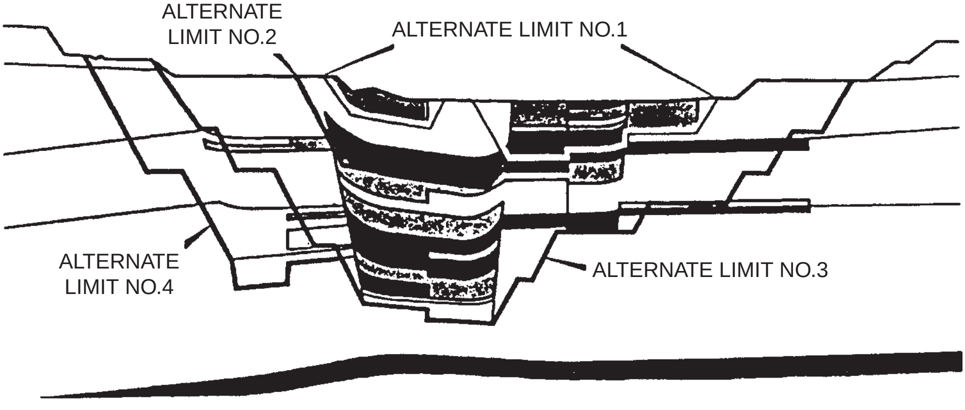


Figure 3.29. A typical cross section showing the structure of a Minnesota Mesabi range iron orebody (Axelson, 1963).

would appear to continue. The slope has been drawn at a point measured 50 ft along the pit bottom. The ore width associated with hole 5 becomes 75 ft.

Final pit outline

The surface is assumed to be flat and at 0 elevation. A bench height of 25 will be used. Through an examination of all sections, it is seen that the overburden-rock interface lies at an average elevation of about 100 ft. In the plan view shown in Figure 3.31, the surface (×) and 100 ft intercepts (○) read from the 6 sections have been marked.

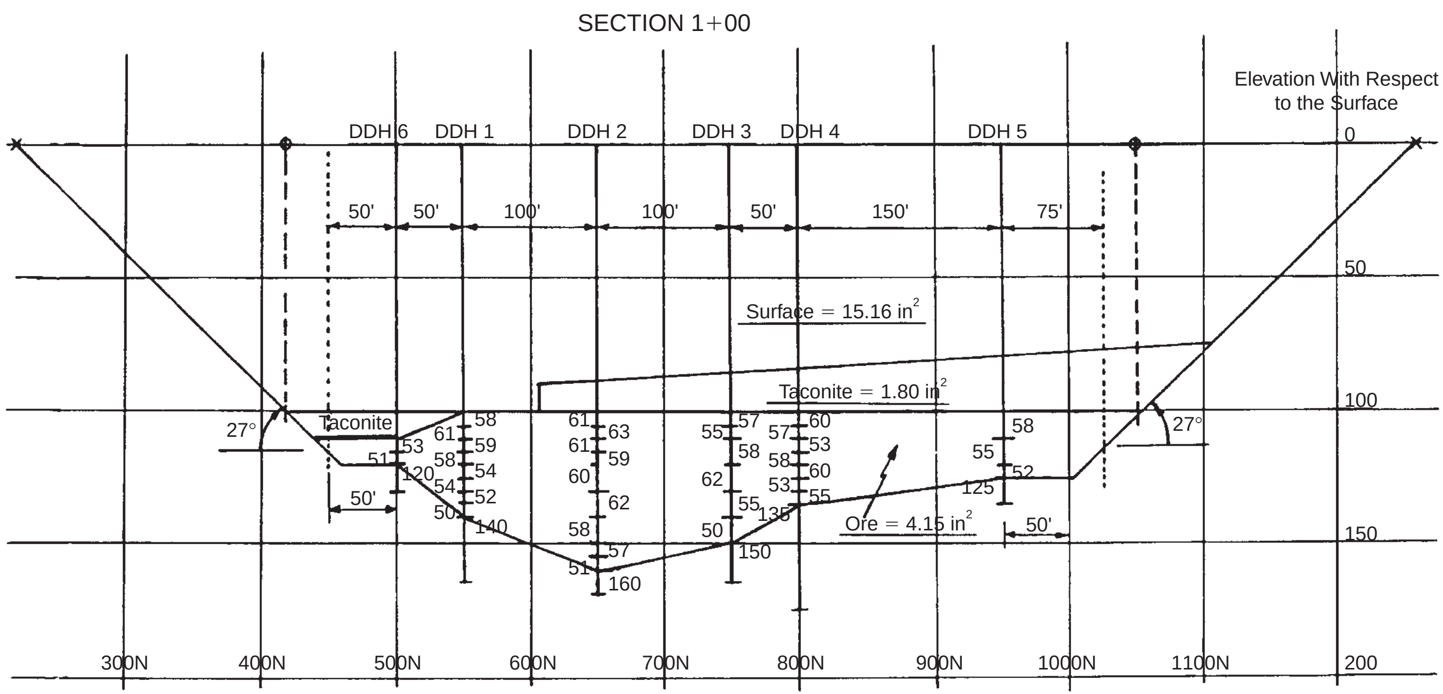


Figure 3.30. Section used for the average sectional grade and tonnage calculation.

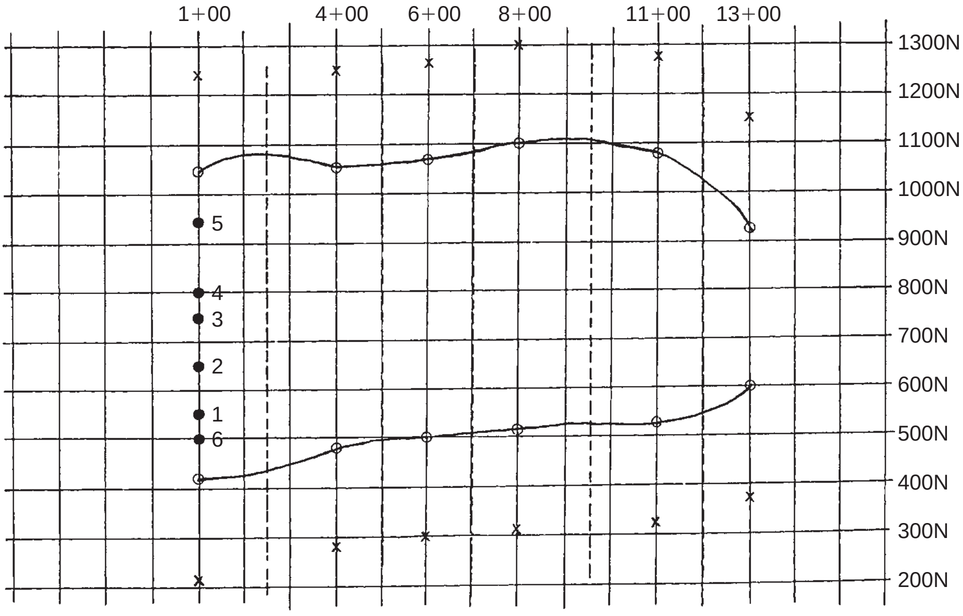


Figure 3.31. Plan smoothing of the pit limits.

The ends of the pit must now be completed. There are several possible constructions which might be used.

1. Construct longitudinal sections through the orebody and estimate ore continuation by projection.

2. Extend the ore a distance equal to the ore thickness observed in the end section.

3. Extend the ends some fixed distance past the last section. For example 100 ft or a distance equal to half the distance between sections.

In this particular case, the second method was selected. As can be seen in Figure 3.30, the maximum ore thickness of 60 ft occurs in hole 2. This thickness extends in plan from holes 1 to 4. Projection of the lines from the pit bottom to the 100 ft and surface elevations yields the points a, b, c and d shown on Figure 3.32. The same procedure has been followed on the east end of the pit where the ore thickness is 40 ft. The final step is to connect the points by smooth curves. In this case, the transition between sections is smooth and no further adjustment is required. If the individual sections do not fit together as nicely as in Figure 3.32, then obviously an iterative procedure of going from plan to section to plan, etc. is required.

Tonnage and average grade for a section

Section 1 + 00 is defined by 6 drillholes. The first step in determining average grade for the section is to find the average grade for each drill hole. If the sample interval was always the same then a simple average of the assays would suffice. In general this is not the case and compositing must be done. The results of this are given in Table 3.12. An influence area for each hole must now be calculated. This area is the ore intercept height times a width. For interior holes, the width extends halfway to adjacent holes. For side holes the width extends

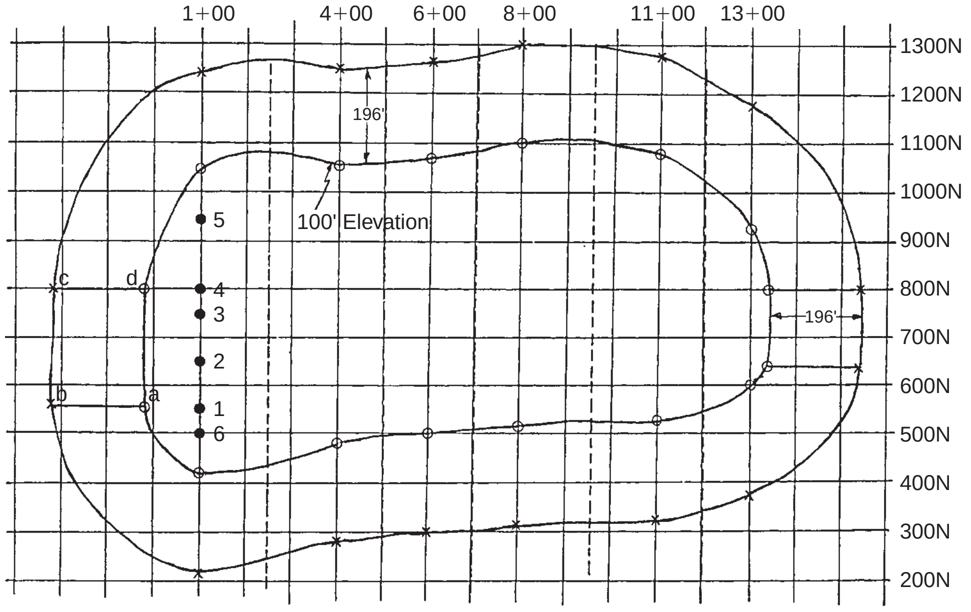


Figure 3.32. Addition of pit ends.

from the side line halfway to the adjacent hole. For this section, the influence areas are as shown in Figure 3.33 and summarized in Table 3.13. Using the grades and influence areas, an average grade for the section is determined by compositing. In this case it is 56.8% Fe.

Tons and average grade for the pit

The ore volume attributed to the section is obtained by multiplying the section area (determined by calculation or planimeter) by the section interval which for section 1 + 00 is 210 ft (Table 3.14). For this section the ore volume is 4,357,500 ft^3. Applying a tonnage factor of 14 ft^3/lt yields 311,000 tons.

Similar figures are developed for each of the other sections. The average grade for the pit is obtained by compositing. The total ore tonnage is just the sum of the tons (A).

The associated waste consists of two types:

– surface material, and
– taconite.

On each section, the areas of each material type are determined by planimetering. The results for section 1 + 00 are given in Tables 3.15 and 3.16.

Pit end tonnage

A somewhat troublesome problem is deciding how to include the material making up the pit ends. The west end of the pit will be examined in this example assuming that all of the material above the 100 ft contour is surface material. In Figure 3.33, the end has been divided into five sectors which will be approximated by the following two shapes.

– prism (A_1), and
– frustum of a right cone ($A_2 \rightarrow A_5$).

Table 3.12. Calculating average hole analyses (Pfleider, 1962).

Cross-sect.	DH	From	To	Length (ft)	Avg Analysis-dry (Fe%)	Length × Analysis (ft × %Fe)
1 + 00	1	100	105	5	58	290
		105	110	5	61	305
		110	115	5	59	295
		115	120	5	58	290
		120	125	5	54	270
		125	130	5	54	270
		130	135	5	52	260
		135	140	5	50	250
	Avg	100	140	40	55.8	2230
	2	100	105	5	61	305
		105	110	5	63	315
		110	115	5	61	305
		115	120	5	59	295
		120	130	10	60	600
		130	140	10	62	620
		140	150	10	58	580
		150	155	5	57	285
		155	160	5	51	255
	Avg	100	160	60	59.3	3560
	3	100	105	5	57	285
		105	110	5	55	275
		110	120	10	58	580
		120	130	10	62	620
		130	140	10	55	550
		140	150	10	50	500
	Avg	100	150	50	56.2	2810
	4	100	105	5	60	300
		105	110	5	57	285
		110	115	5	53	265
		115	120	5	58	290
		120	125	5	60	300
		125	130	5	53	265
		130	135	5	55	275
	Avg	100	135	35	56.6	1980
	5	100	110	10	58	580
		110	120	10	55	550
		120	125	5	52	260
	Avg	100	125	25	55.6	1390
	6	110	115	5	53	265
		115	120	5	51	255
	Avg	110	120	10	52.0	520

An isometric drawing of sectors A_1, A_2 and A_3 is shown in Figure 3.34. The individual parts are shown in Figure 3.35. The general formula for the volume of a prism is

$$V_p = \frac{1}{2}(S_1 + S_2)h \tag{3.12}$$

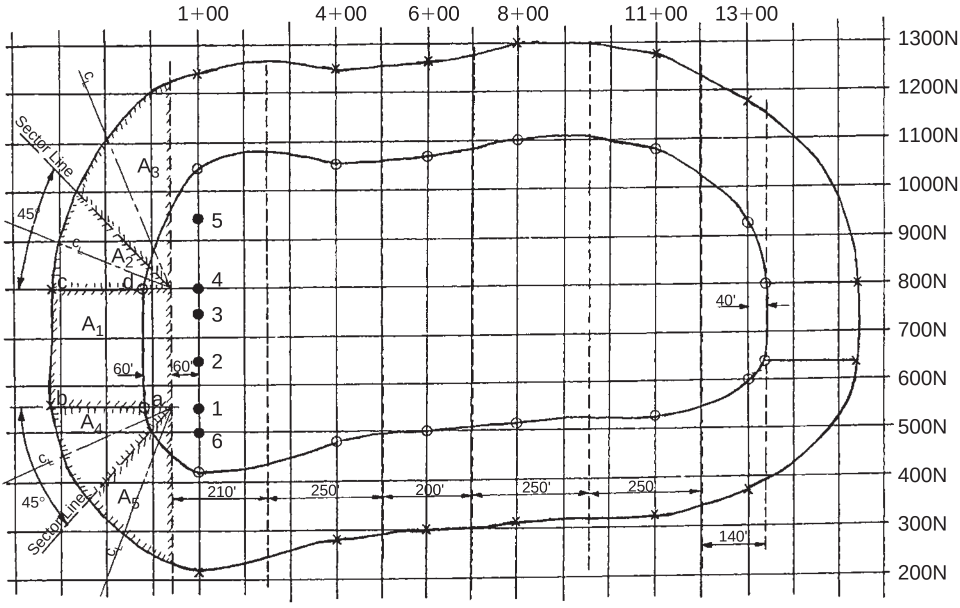


Figure 3.33. Pit end volume calculations.

Table 3.13. Calculating average section analyses (Pfleider, 1962).

Cross-sect.	Hole No.	From	To	Length	Dist. bet holes	Area (ft^2)	Avg Analysis (%Fe)	Area × % (ft^2 × %Fe)
1 + 00	1	100	140	40	75	3,000	55.8	167,400
	2	100	160	60	100	6,000	59.3	355,800
	3	100	150	50	75	3,750	56.2	210,750
	4	100	135	35	100	3,500	56.6	198,100
	5	100	125	25	150	3,750	55.6	208,500
	6	110	120	100	75	750	52.0	39,000
	Total					20,750	56.8	1,180,000

where S_1, S_2 are the areas of the top and bottom surfaces, respectively, and h is the altitude. The formula for the volume of a right cone is

$$V_c = \frac{1}{3}\pi r^2 h \tag{3.13}$$

where r is the radius of the base.

The formula for the volume of the frustum of a right cone is

$$V_{fc} = \frac{\pi h}{3}(r_1^2 + r_1 r_2 + r_2^2) \tag{3.14}$$

where r_1 is the radius of the base and r_2 is the radius of the top.

Table 3.14. Summary sheet for ore tons and grade (Pfleider, 1962).

Cross-sect.	Planimetered area (in^2)	Area factor (ft^2/in^2)	Section area (ft^2)	Section interval (ft)	Volume (ft^3)	Tonnage factor (ft^3/lt)	Tons (lt)	Grade (%) (%Fe)	Tons × % (lt × %Fe)
1 + 00	4.15	50 × 100	20,750	210	4,357,500	14	311,000	56.8	17,664,800
4 + 00				250					
6 + 00				200					
8 + 00				250					
11 + 00				250					
13 + 00				140					
Grand total							A	$\frac{B}{A}$	B

Table 3.15. Summary sheet for surface material (Pfleider, 1962).

Cross-sect.	Planimetered area (in^2)	Area factor (ft^2/in^2)	Section area (ft^2)	Section interval (ft)	Volume (ft^3)	Volume (yd^3)	Tonnage factor (ft^3/st)	Tons (st)
1 + 00	15.16	50 × 100	75,800	210	15,918,000	590,000	19	838,000
4 + 00								
6 + 00								
8 + 00								
11 + 00								
13 + 00								
Total						A		B

Table 3.16. Summary sheet for taconite (Pfleider, 1962).

Cross-sect.	Plan. area (in^2)	Area factor (ft^2/in^2)	Section area (ft^2)	Section interval (ft)	Volume (ft^3)	Tonnage factor (ft^3/st)	Tons (st)
1 + 00							
4 + 00							
6 + 00							
8 + 00							
11 + 00							
13 + 00							
Total							C

Applying formula (3.12) to sector A_1, one finds

$$V_{A_1} = \frac{1}{2}(60 \times 240 + 256 \times 240)100 = 3,792,000 \text{ ft}^3$$

For sector A_2, base and top radii are determined along the sector centerlines. These become:

$$r_1 = 293 \text{ ft}$$

$$r_2 = 60 \text{ ft}$$

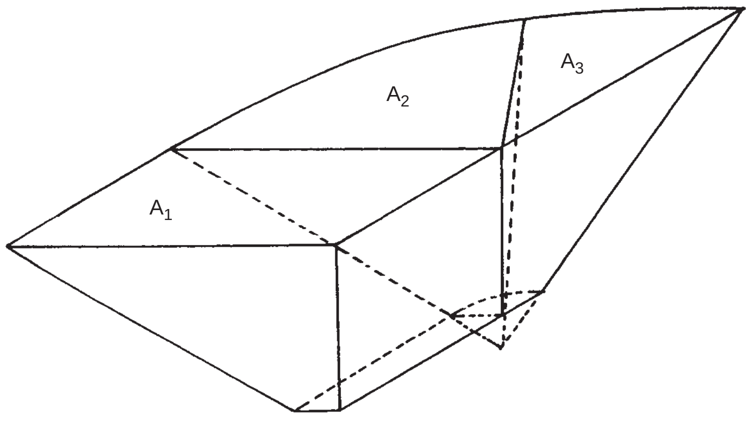


Figure 3.34. Isometric view of sections A_1, A_2 and A_3.

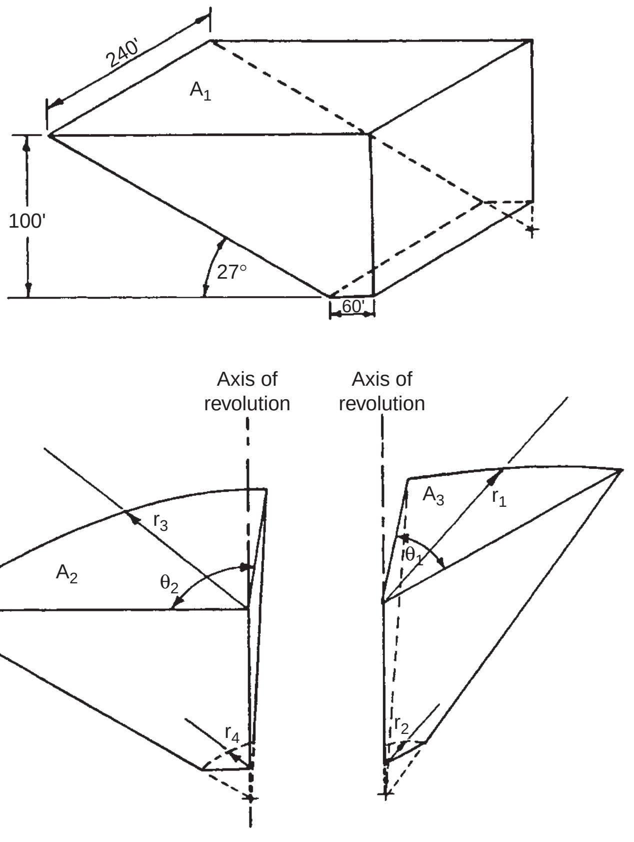


Figure 3.35. Exploded view of the pit end volumes.

The included angle (θ) of the sector is 45°. Hence

$$V_{A_2} = \frac{\pi}{3}100(293^2 + 293 \times 60 + 60^2)\frac{45}{360} = 1,401,010\ \text{ft}^3$$

For sector A_3:

$$\theta_2 = 45°$$

$$r_3 = 387\ \text{ft}$$

$$r_4 = 93\ \text{ft}$$

$$V_{A_3} = \frac{\pi}{3}100(387^2 + 387 \times 93 + 93^2)\frac{45}{360} = 2,544,814\ \text{ft}^3$$

Similarly for sectors A_4 and A_5:

$$V_{A_4} = \frac{\pi}{3}100(293^2 + 293 \times 53 + 53^2)\frac{45}{360} = 1,363,808\ \text{ft}^3$$

$$V_{A_5} = \frac{\pi}{3}100(346^2 + 346 \times 93 + 93^2)\frac{45}{360} = 2,101,508\ \text{ft}^3$$

The total volume of the west end then becomes

$$V_{\text{we}} = 11,204,000\ \text{ft}^3$$

Applying a tonnage factor of 19 ft^3/st, yields 590,000 st. The split between taconite and rock can be found by including the interface in the drawings.

In estimating actual grade and tonnage from the pit one must take into account:
- ore losses in pit,
- dilution, and
- mill recovery.

3.6 METHOD OF VERTICAL SECTIONS (GRADE CONTOURS)

A less commonly used technique to represent grades on sections is through iso-grade contours. Although there are several reasons for adopting such an approach, the primary one in the example to be considered is the fact that the directions of the exploratory drill holes were highly variable. This example, using the data and approach of Cherrier (1968), will demonstrate the application of the technique for determining tons and average grade and also the application of longitudinal sections for completing pit ends. Figure 3.36 is a plan map for a molybdenum orebody showing the location of the drill holes and the surface topography. The grid system has been superimposed. Sections 4, 8, 12, 16 and 20 are given in Figures 3.37 through 3.41. The drill holes on Sections 2 and 22 revealed no ore present. Using the drill hole data and a knowledge of the deposit form, the grade contours have been created. A pit outline has been developed using procedures which will be described in Chapter 5. In viewing the 5 transverse sections, it can be seen that longitudinal section 3200 N runs approximately along the proposed pit axis. This section is shown in Figure 3.42. Rock mechanics studies have suggested a 45° slope angle at the east end and 50° at the west. From the drilling results, the orebody must terminate between Sections 2 to 4 on the west and 20 to 22 on the east. The grade contour lines have been drawn in to represent this (Fig. 3.43) condition. In examining the plan representation (Fig. 3.44) of the final pit,

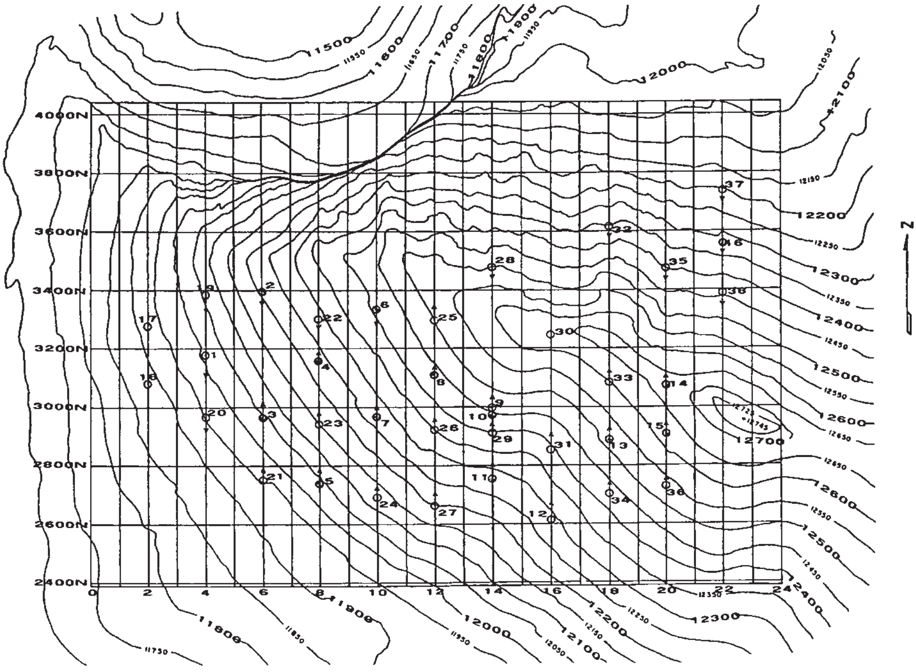


Figure 3.36. Plan map showing drill hole locations (Cherrier, 1968).

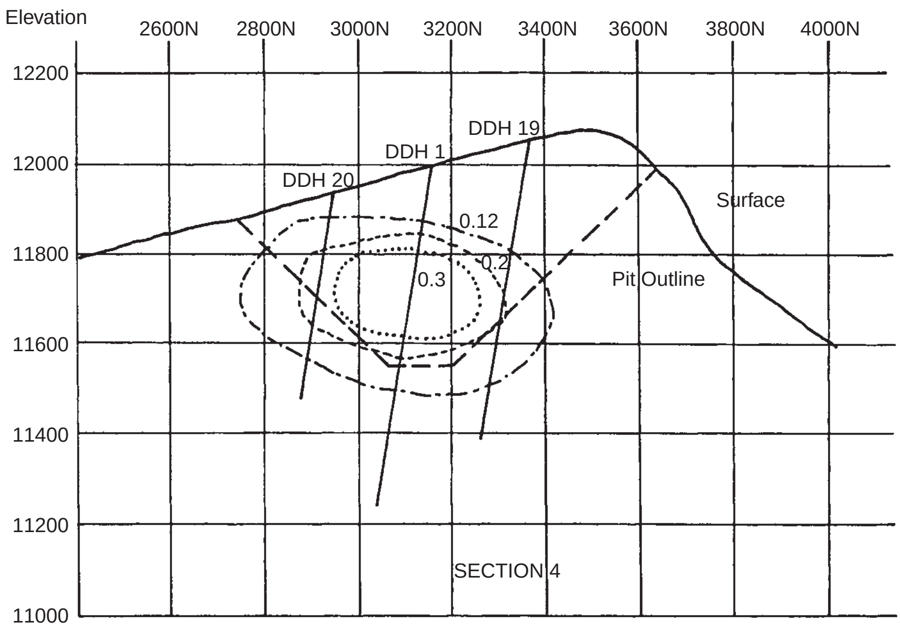


Figure 3.37. Section 4 showing the grade contours and pit outline (Cherrier, 1968).

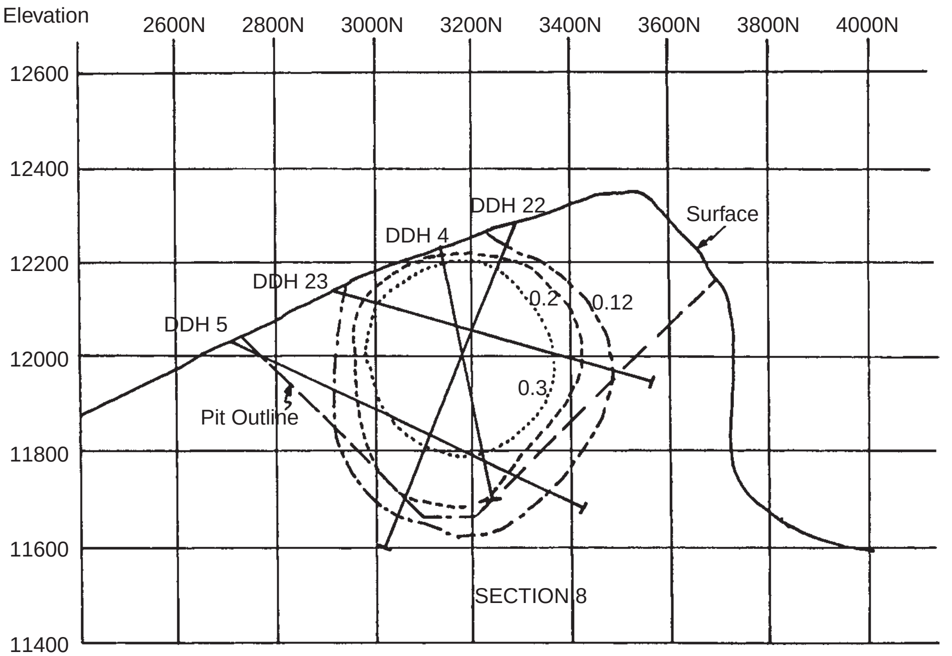


Figure 3.38. Section 8 showing the grade contours and pit outline (Cherrier, 1968).

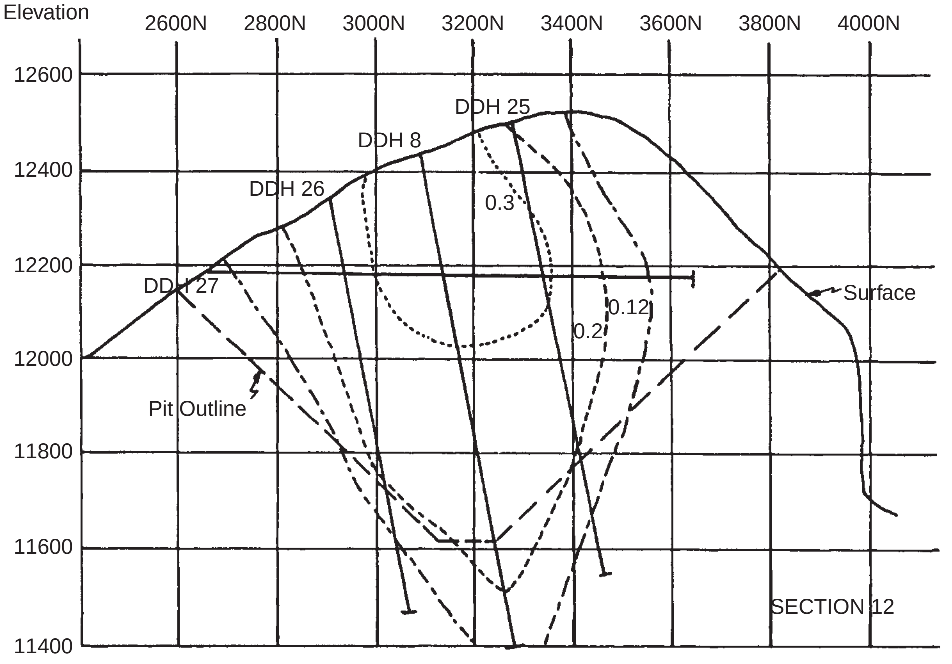


Figure 3.39. Section 12 showing the grade contours and pit outline (Cherrier, 1968).

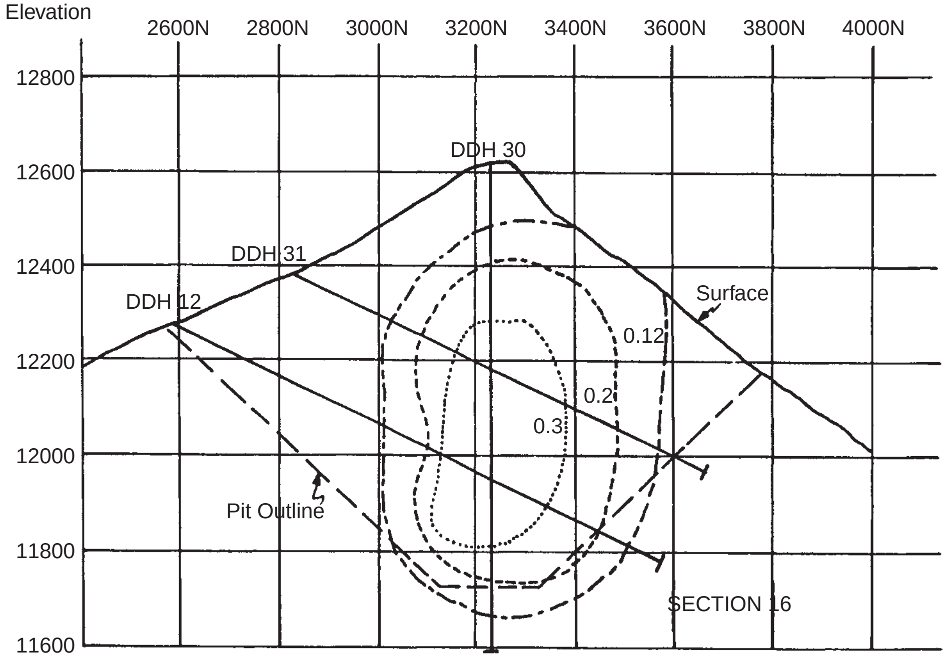


Figure 3.40. Section 16 showing the grade contours and pit outline (Cherrier, 1968).

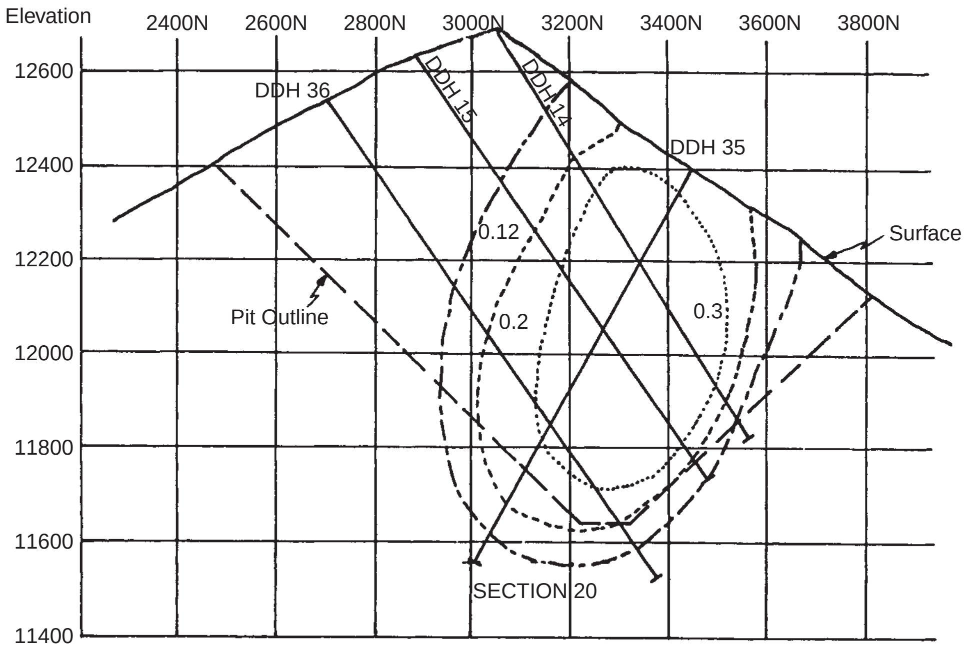


Figure 3.41. Section 20 showing the grade contours and pit outline (Cherrier, 1968).

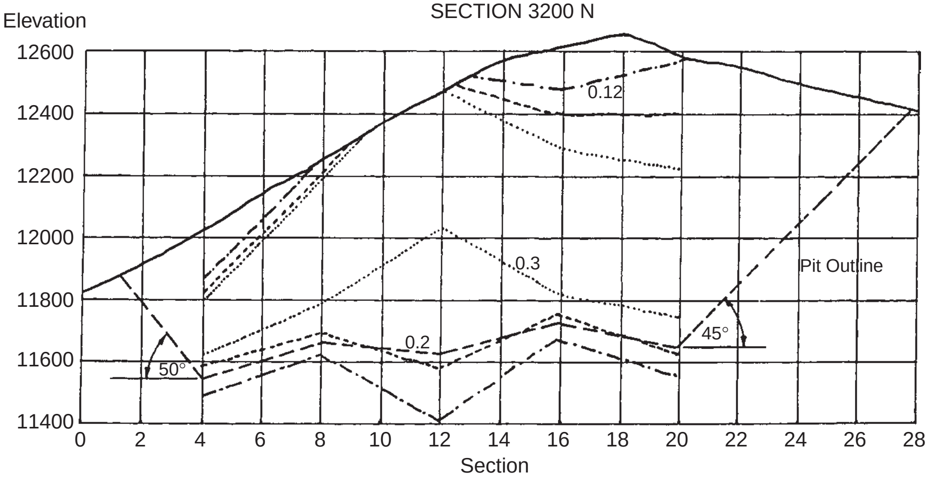


Figure 3.42. Longitudinal Section 3200 N showing the grade contours, surface topography and pit outline (Cherrier, 1968).

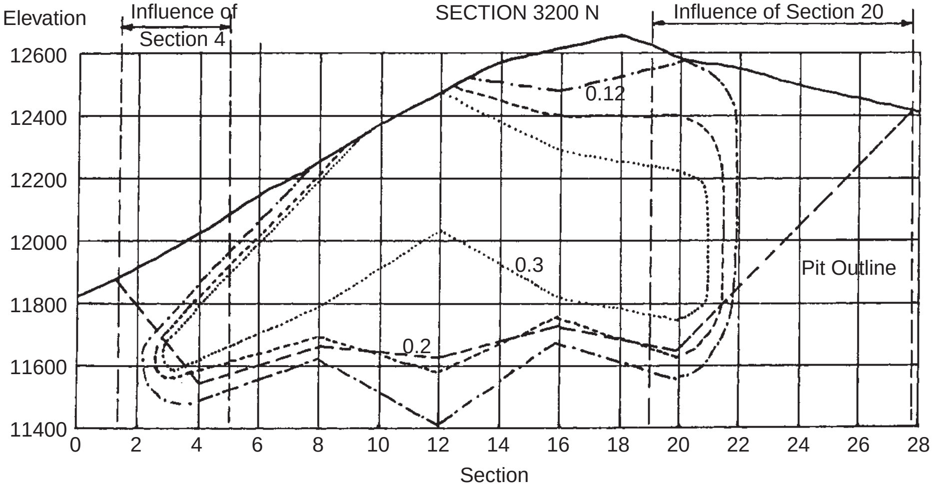


Figure 3.43. Grade contours completed at the pit ends.

and the longitudinal section, it has been decided to incorporate the end volumes of the pit into Sections 4 and 20 rather than to treat them separately. This is accomplished by varying the longitudinal length of influence so that the resulting volumes are correct. For the east and west ends the following have been used:

Section 4	Grade zone	Influence distance (ft)
	+0.3	200
	0.2–0.3	225
	0.12–0.2	250
	Overburden	300

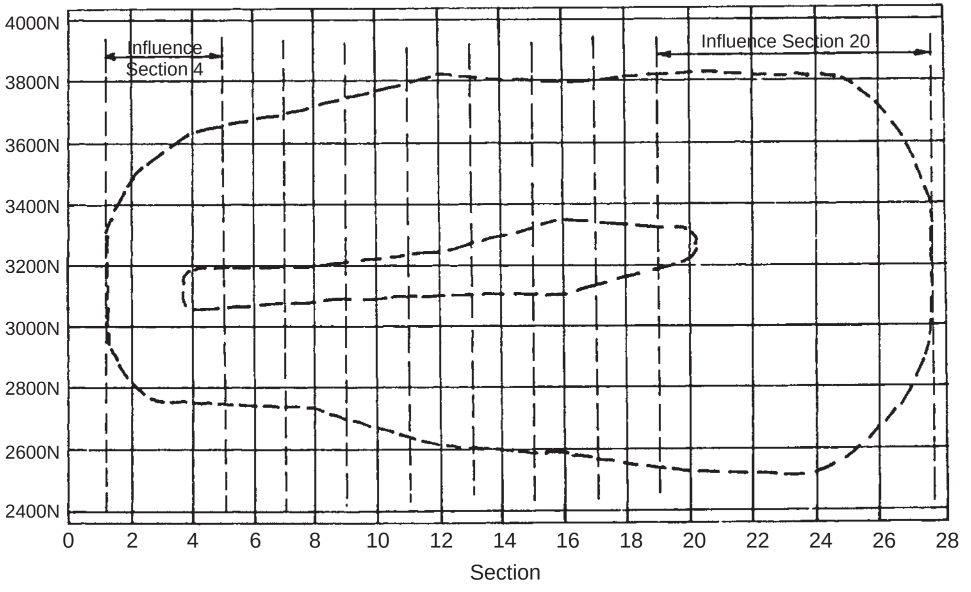


Figure 3.44. Plan view showing outline of pit crest and toe.

20	+0.3	200
	0.2–0.3	250
	0.12–0.2	300
	Overburden	400

The reader is encouraged to check these approximations. For each section the areas corresponding to each grade zone within the pit outline are determined using a planimeter. These are then converted into volumes and tons as summarized in Table 3.17. A summary of ore and waste tons as well as the ratio of the two (average stripping ratio) by section is given in Table 3.18. The overall stripping ratio for the pit is also calculated.

One assay was performed for each 10 ft of core in every drill hole. For each section, the assays lying within a particular grade zone were added together. This sum was then divided by the number of assays to obtain an average. Each average assay was weighted by the volume of influence for the respective zone. These weighted averages were totalled and divided by the total volume for the three zones. This figure is then the average assay for the entire ore zone on that section. The results of this calculation are given in Table 3.19. The overall average grade for the pit is found in Table 3.20.

In summary:

Tonnage of ore = 53,288,000 st
Average grade = 0.281 percent (MoS_2)
Tonnage of overburden = 32,311,000 st
Average stripping ratio = 0.61

Some dilution and ore loss can be expected from this orebody due to the difficulty of defining the ore cut-off grade and mining to this grade in the upper half of the orebody. The following calculation illustrates how one might account for these effects and determine an average grade for the ore actually mined.

Table 3.17. Calculation of tonnage of ore and overburden (Cherrier, 1968).

Section	Grade zone	Area[1] (in^2)	Area[1] ($10^4\ ft^2$)	Influence distance (ft)	Volume ($10^6\ ft^3$)	Tons[2] ($\times 10^3$)	Total tons ($\times 10^3$)
4	+0.3	1.32	5.28	200	10.56	845	
	0.2–0.3	0.84	3.36	225	7.56	605	= 2,266
	0.12–0.2	1.02	4.08	250	10.20	816	
	Overburden	2.81	11.24	300	33.72	2,698	2,698
6	+0.3	2.52	10.08	200	20.16	1,613	
	0.2–0.3	3.46	13.84	200	27.68	2,214	= 5,075
	0.12–0.2	1.95	7.80	200	15.60	1,248	
	Overburden	2.17	8.68	200	17.36	1,389	1,389
8	+0.3	2.97	11.88	200	23.76	1,901	
	0.12–0.3	1.66	6.64	200	13.28	1,062	= 3,763
	0.12–0.2	1.25	5.00	200	10.00	800	
	Overburden	2.62	10.48	200	20.96	1,677	1,677
10	+0.3	2.17	8.68	200	17.36	1,389	
	0.2–0.3	3.00	12.00	200	24.00	1,920	= 4,423
	0.12–0.2	1.74	6.96	200	13.92	1,114	
	Overburden	5.66	22.64	200	45.28	3,622	3,622
12	+0.3	3.11	12.44	200	24.88	1,990	
	0.2–0.3	6.64	26.56	200	53.12	4,250	= 7,872
	0.12–0.3	2.55	10.20	200	20.40	1,632	
	Overburden	3.17	12.68	200	25.36	2,029	2,029
14	+0.3	2.46	9.84	200	19.68	1,574	
	0.2–0.3	4.30	17.20	200	34.40	2,752	= 5,657
	0.12–0.2	2.08	8.32	200	16.64	1,331	
	Overburden	6.90	27.60	200	55.20	4,416	4,416
16	+0.3	2.57	10.28	200	20.56	1,645	
	0.2–0.3	3.18	12.72	200	25.44	2,035	= 5,760
	0.12–0.2	3.25	13.00	200	26.00	2,080	
	Overburden	5.51	22.40	200	44.08	3,526	3,526
18	+0.3	4.09	16.36	200	32.72	2,618	
	0.2–0.3	5.29	37.52	200	75.04	6,003	= 10,445
	0.12–0.2	2.85	11.40	200	22.80	1,824	
	Overburden	6.10	24.40	200	48.80	3,904	3,904
20	+0.3	4.99	19.96	200	39.92	3,194	
	0.2–0.3	3.27	13.08	250	32.70	2,616	= 8,181
	0.12–0.2	2.47	9.88	300	29.64	2,371	
	Overburden	7.07	28.28	400	113.12	9,050	9,050

[1] Planimetered: $1\ in^2 = 200' \times 200' = 40,000\ ft^2$.
[2] Based on a tonnage factor of $12.5\ ft^3/st$.

	Tons (st)	Grade (%)	Tons × Grade (st × %)
Ore in place	53.0×10^6	0.281	14.893×10^6
Dilution (est. 5%)	$+2.65 \times 10^6$	0.100	$+0.265 \times 10^6$
Ore loss (est. 5%)	-2.65×10^6	0.140	-0.371×10^6
Total	53.0×10^6		14.787×10^6

$$\text{Average grade of ore mined} = \frac{14.787}{53} = 0.279 \text{ percent.}$$

Table 3.18. Calculation of the average stripping ratio.

Section	Tons of ore ($st \times 10^3$)	Tons of overburden ($st \times 10^3$)	Avg stripping ratio
4	2,266	2,698	1.191
6	5,075	1,389	0.274
8	3,763	1,677	0.446
10	4,423	3,622	0.819
12	7,872	2,029	0.258
14	5,657	4,416	0.781
16	5,760	3,526	0.612
18	10,445	3,904	0.374
20	8,181	9,050	1.106
Total	53,442	32,311	

Overall average stripping ratio: $\dfrac{32,311}{53,442} = 0.605$.

3.7 THE METHOD OF HORIZONTAL SECTIONS

3.7.1 *Introduction*

Although vertical sections have played a dominant role in ore reserve estimation in the past, today, for many, if not most, deposits this function is rapidly being replaced by techniques based upon the use of horizontal sections. The primary reason being the widespread availability of computers for doing the tedious, time consuming calculations involved and the development of new techniques for estimating the grades between drill holes. Sections taken in the plane of the orebody have generally been used for evaluating relatively thin, flat lying deposits such as uranium, coal, sand, gravel, placer gold, etc. They may be of relatively uniform or varying thickness. Thick deposits are mined in a series of horizontal slices (benches) of uniform thickness. For extraction planning, bench plans showing tons and grade are of utmost importance. Hence even if vertical sections are used for initial evaluation, bench (horizontal) sections are eventually required. In this section hand methods for calculating tons and grade based on triangles and polygons will be discussed. These discussions will use as a basis the drillholes shown in Figure 3.45. The corresponding grades and location coordinates are given in Table 3.21.

3.7.2 *Triangles*

In the *triangular* method, diagrammatically illustrated in Figure 3.46, each hole is taken to be at one corner of a triangle. If the triangular solid formed is of constant thickness t, its volume is just equal to the plan area A times this thickness. To obtain tons, the appropriate tonnage factor is applied. The average grade $\bar{g}$ is given by

$$\bar{g} = \frac{g_1 + g_2 + g_3}{3} \tag{3.15}$$

where g_i are the grades at the three corners.

The area of the triangle can be found using a planimeter or through calculation if the coordinates (x_i, y_i) of the corners are known. This method, shown in Figure 3.47, readily adapts to calculator/computer application. The average grade for the block is strictly correct

Table 3.19. Calculation of average assay for each section (Cherrier, 1968).

Section	Grade zone		Avg assay (%)	Volume ($\times 10^6$ ft^3)	Assay × Volume (% $\times 10^6$ ft^3)	Avg assay for section
4	+0.3		0.452	10.56	4.7731	
	0.2–0.3		0.270	7.56	2.0412	
	0.12–0.2		0.135	10.20	1.3770	
		Total		28.32	8.1913	0.289
6	+0.3		0.435	20.16	8.7696	
	0.2–0.3		0.242	27.68	6.6986	
	0.12–0.2		0.155	15.60	2.4180	
		Total		63.44	17.8862	0.282
8	+0.3		0.485	23.76	11.5236	
	0.2–0.3		0.246	13.28	3.2669	
	0.12–0.2		0.153	10.00	1.5300	
		Total		47.04	16.3205	0.347
10	+0.3		0.379	17.36	6.5794	
	0.2–0.3		0.261	24.00	6.2640	
	0.12–0.2		0.151	13.92	2.1019	
		Total		55.28	14.9453	0.270
12	+0.3		0.411	24.88	10.2257	
	0.2–0.3		0.243	43.12	12.9082	
	0.12–0.2		0.142	20.40	2.8968	
		Total		98.40	26.0307	0.265
14	+0.3		0.403	19.68	7.9310	
	0.02–0.3		0.242	34.40	8.3248	
	0.12–0.2		0.168	16.64	2.7955	
		Total		70.72	19.0513	0.269
16	+0.3		0.398	20.56	8.1829	
	0.2–0.3		0.247	25.44	6.2837	
	0.12–0.2		0.155	26.00	4.0300	
		Total		72.00	18.4966	0.257
18	+0.3		0.393	32.72	12.8590	
	0.2–0.3		0.274	75.04	20.5610	
	0.12–0.2		0.158	22.80	3.6024	
		Total		130.56	37.0224	0.284
20	+0.3		0.405	39.92	16.1676	
	0.2–0.3		0.261	32.70	8.5347	
	0.12–0.2		0.164	29.64	4.8610	
		Total		102.26	29.5633	0.289

only for equilateral triangles. For other triangle shapes the area associated with each grade is not equal as assumed in the formula. For triangular solids which are not of constant thickness, then some additional calculations are required. The average thickness $\bar{t}$ is given by

$$\bar{t} = \frac{t_1 + t_2 + t_3}{3} \tag{3.16}$$

and the average grade $\bar{g}$ is

$$\bar{g} = \frac{g_1 t_1 + g_2 t_2 + g_3 t_3}{3\bar{t}} \tag{3.17}$$

Table 3.20. Calculation of overall average grade (Cherrier, 1968).

Section	Average assay	Volume ($\times 10^6$ ft^3)	Assay $\times$ Volume (% $\times 10^6$ ft^3)
4	0.289	28.32	8.1845
6	0.282	63.44	17.8901
8	0.347	47.04	16.3229
10	0.270	55.28	14.9256
12	0.265	98.40	26.0760
14	0.269	70.72	19.0237
16	0.257	72.00	18.5040
18	0.284	130.56	37.0790
20	0.289	102.26	29.5531
Total		668.02	187.5589

Overall average grade of orebody: $\dfrac{187.5589}{668.02} = 0.281$ percent.

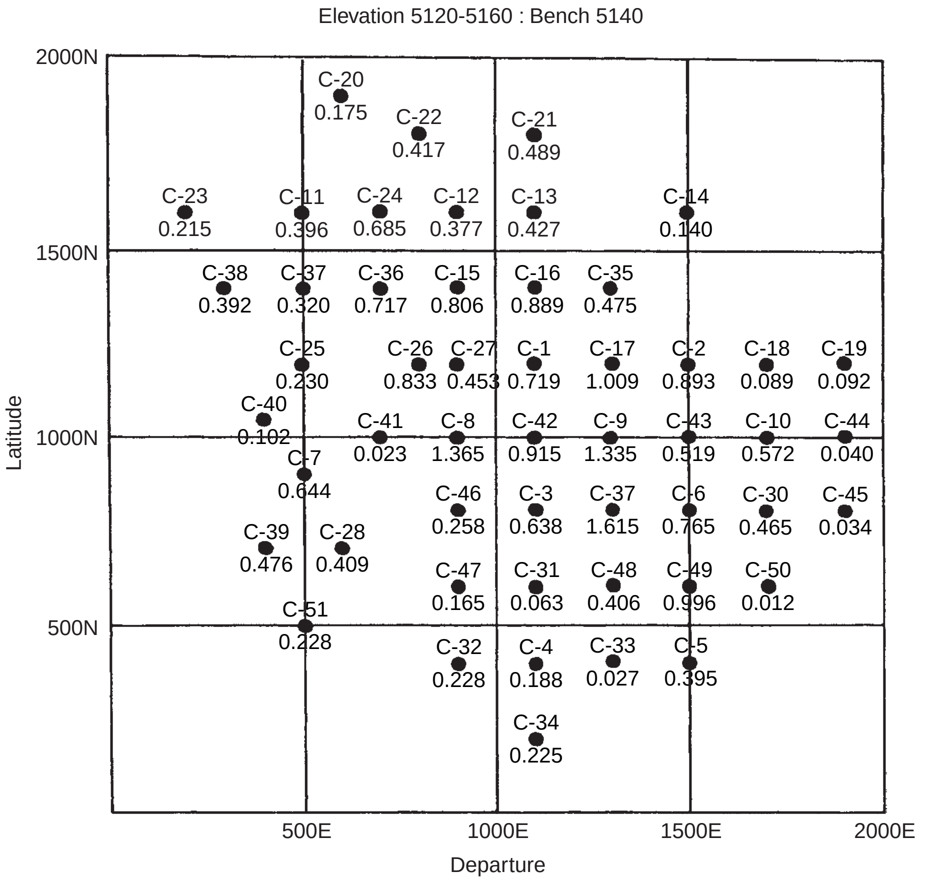


Figure 3.45. Borehole locations and grades for Bench 5140 (Hughes & Davey, 1979).

Table 3.21. Coordinates and grades for the holes of Figure 3.45. The composite grades are for Bench 5140 (Hughes & Davey, 1979).

Hole	Coordinates		Grade (% Cu)	Hole	Coordinates		Grade (% Cu)
	East	North			East	North	
C-1	1100	1200	0.719	C-27	900	1200	0.453
C-2	1500	1200	0.893	C-28	600	700	0.409
C-3	1100	800	0.638	C-29	1300	800	1.615
C-4	1100	400	0.188	C-30	1700	800	0.465
C-5	1500	400	0.395	C-31	1100	600	0.063
C-6	1500	800	0.765	C-32	900	400	0.224
C-7	500	900	0.644	C-33	1300	1400	0.027
C-8	900	1000	1.365	C-34	1100	200	0.225
C-9	1300	1000	1.335	C-35	1300	1400	0.475
C-10	1700	1000	0.072	C-36	700	1400	0.717
C-11	500	1600	0.396	C-37	500	1400	0.320
C-12	900	1600	0.377	C-38	300	1400	0.392
C-13	1100	1600	0.427	C-39	400	700	0.476
C-14	1500	1600	0.140	C-40	400	1050	0.102
C-15	900	1400	0.806	C-41	700	1000	0.023
C-16	1100	1400	0.889	C-42	1100	1000	0.915
C-17	1300	1200	1.009	C-43	1500	1000	0.519
C-18	1700	1200	0.089	C-44	1900	1000	0.040
C-19	1900	1200	0.092	C-45	1900	800	0.034
C-20	600	1900	0.175	C-46	900	800	0.258
C-21	1100	1800	0.489	C-47	900	600	0.165
C-22	800	1800	0.417	C-48	1300	600	0.406
C-23	200	1600	0.215	C-49	1500	600	0.996
C-24	700	1600	0.685	C-50	1700	600	0.012
C-25	500	1200	0.230	C-51	500	500	0.228
C-26	800	1200	0.833				

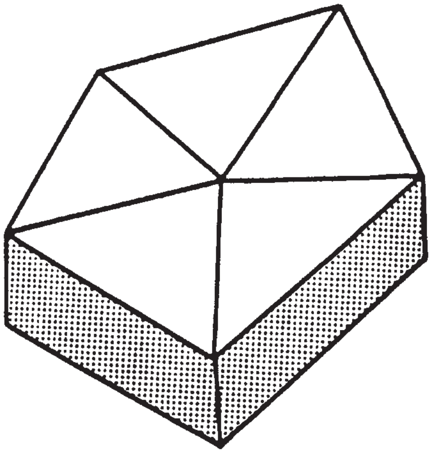
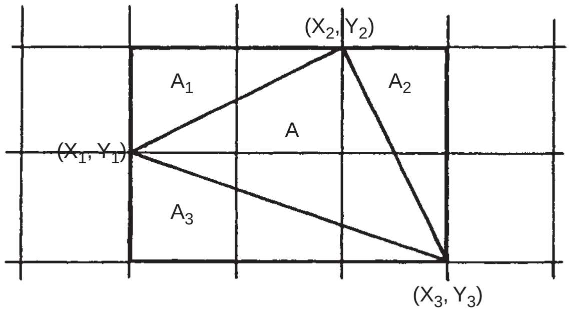


Figure 3.46. Triangular prisms (Barnes, 1979b).



$$A = (X_3 - X_1)(Y_2 - Y_1) - 1/2\,(X_3 - X_1)(Y_1 - Y_3) - 1/2\,(X_3 - X_2)(Y_2 - Y_3)$$

$$- 1/2\,(X_2 - X_1)(Y_2 - Y_1)$$

$$A = \text{Area of Rectangle} - A_1 - A_2 - A_3$$

$$A = (X_{max} - X_{min})(Y_{max} - Y_{min}) - 1/2 \sum_{\substack{i=1,3 \\ j=1,3}} (X_i - X_j)(Y_i - Y_j)$$

Figure 3.47. Calculation of triangular area based upon corner coordinates.

Using the average thickness, the plan area and tonnage factor, the tons can be found. The triangular element formed by holes C-30, C-40, and C-50 (assuming a constant bench thickness of 40 ft and a tonnage factor of 12.5 ft^3/ton), contains 64,000 tons with an average grade 0.17.

3.7.3 *Polygons*

In the *polygonal* method, each drill hole lies within the center of a polygon. The polygon is constructed such that its boundary is always equidistant from the nearest neighboring hole. Within the polygon, the grade is assumed constant and equal to that of the hole it includes. The thickness of the polygon is also constant and equal to the ore intercept/bench thickness. The steps followed in forming a polygon around hole C-41 are illustrated in Figure 3.48. In step 1, radial lines (similar to the spokes of a wheel) are drawn from the drill hole to its nearest neighbors. The perpendicular bisectors to these lines are constructed and extended until they meet those from adjacent holes (step 2). The area of the polygon is then determined and the tonnage calculated (step 3). At the drilling boundary, since there are holes on only one side, some special procedures are required. Here it will be assumed that an appropriate radius of influence R is known. This concept will be discussed in detail in the following section. Figure 3.49 illustrates the steps necessary to construct the polygon around hole C-14. Step 1 proceeds as before with radial lines drawn to the surrounding holes. To supply the missing sides a circle of radius R is drawn (Step 2). In this case $R = 250$ ft. Chords are drawn parallel to the property boundary (grid) lines along the top and side (Step 3). The

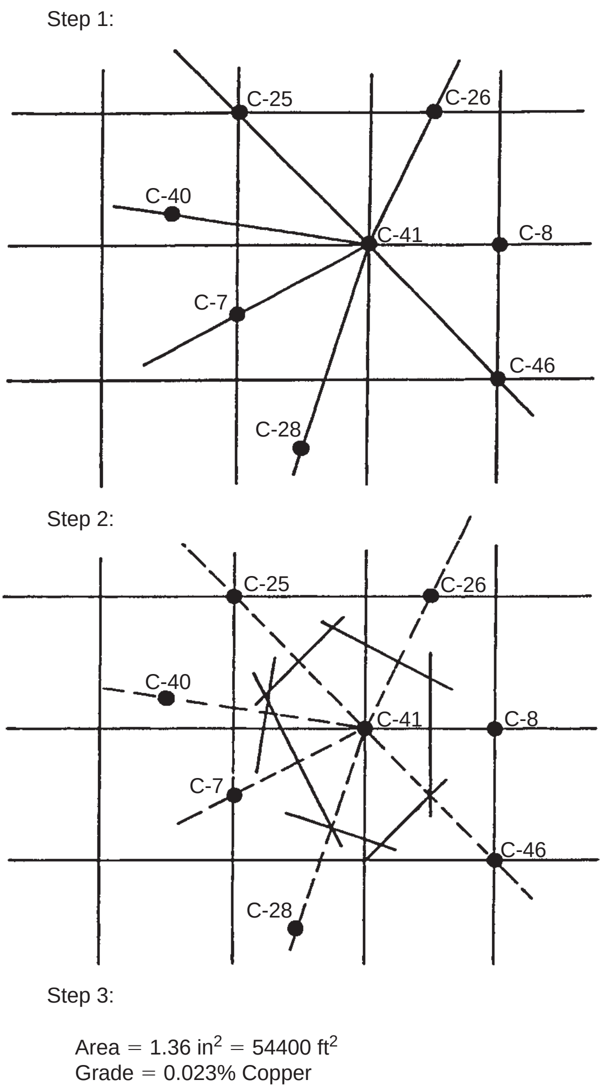


Figure 3.48. The practice of forming polygons for a hole internal to the array.

remaining chords are drawn at angles of 45°, tangent to the circle. In the final step (Step 4) the area is determined, the tonnage calculated and the grade assigned.

Rules developed by Hughes & Davey (1979) which can be followed when constructing polygons are given in Table 3.22. Figure 3.50 shows the hand generated polygons for the drill hole data of Figure 3.45.

Having gone through these two examples it is perhaps of value to list the general steps (after Hughes & Davey, 1979) which are followed:

1) Locations of drill holes and other samples are established for a specified level using available drill-hole survey data. Usually, the drill-hole location and assays of interest are depicted on a horizontal section.

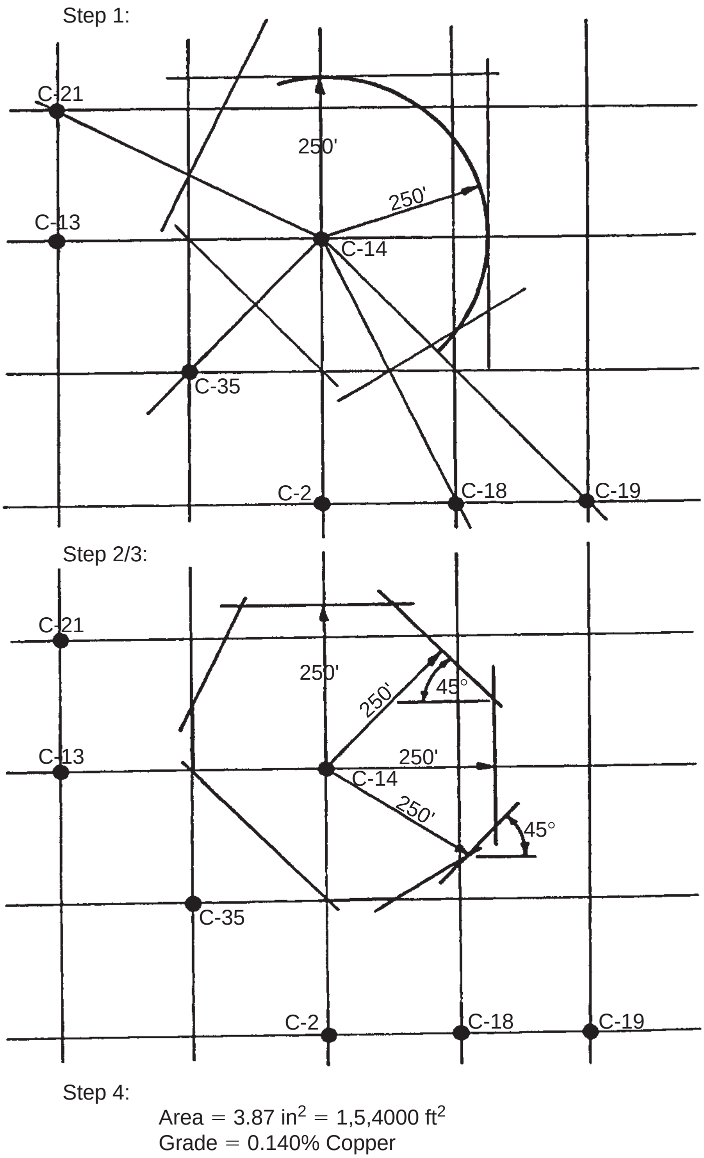


Figure 3.49. The practice of forming polygons at the boundary of the hole array.

2) Drill-hole interval assay data are composited to intervals consistent with bench height. The elevation of the sample is typically determined at the midpoint of the bench.

3) Area of influence or radius of influence is established by geologic and mining experience.

4) Lines are drawn between drill holes that are within two times the radius of influence of each other. This step may be altered by rules such as those in Table 3.22.

5) Perpendicular bisectors are constructed on each of these connecting lines.

Table 3.22. Example of polygon interpolation rules (Hughes & Davey, 1979).

1. Ultimate polygon shape is octagonal (eight-sided).
2. Radius of influence is R ft.
3. No polygon exceeds $2R$ ft from a sample point.
4. If drill holes are in excess of $5R$ ft apart, use a radius of R ft to show trend into undrilled area.
5. If holes are in excess of $4R$ ft, but less than $5R$ ft apart:
   a) Construct an R ft radius circle if assays are of unlike character, i.e., rock types, different mineralization, or one ore and the other waste.
   b) Use a $2R$ ft radius if assays are of like character to locate a point on a line between the holes; a line is then drawn to the point and tangent to the R ft diameter circle.
6. For holes less than $4R$ ft apart, construct a perpendicular bisector between the holes.
   a) If the holes are between $3R$ and $4R$ ft apart, use an R ft radius circle and connect the circles by drawing wings at a 30° angle from the center of the R ft radius circle to a $2R$ ft radius circle. The perpendicular bisector constructed above becomes the dividing line between $2R$ ft arc intersections.
   b) If the holes are less than $3R$ ft apart and a polygon cannot be constructed entirely from perpendicular lines from adjacent holes, then use a R ft radius circle and connect the circles with tangent lines. The dividing line is the perpendicular bisector between the holes.
7. After one pair of holes has been analyzed using these rules, another pair is evaluated, and this procedure is repeated until all combinations have been evaluated.
8. The assay value of the polygon will be the composited assay value of the drill hole that the polygon was constructed around.

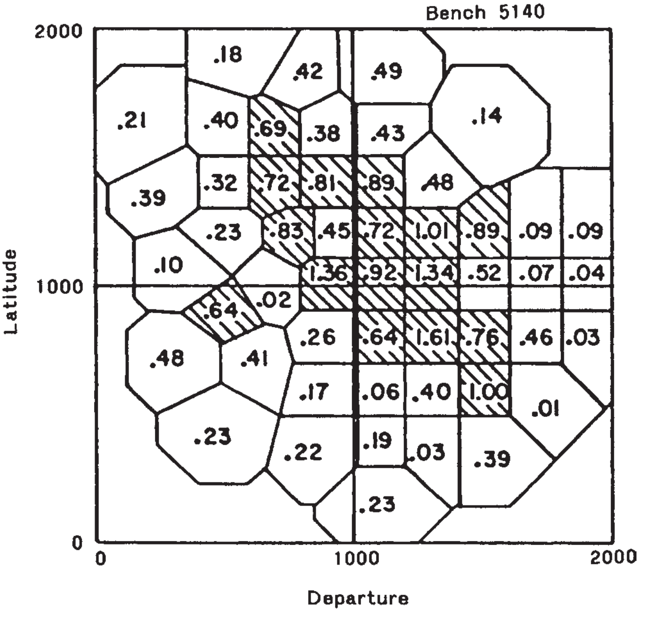


Figure 3.50. Hand-generated polygons from samples in Figure 3.45 (Hughes & Davey, 1979).

6) Bisectors are extended until they intersect. If two lines run parallel or approximately parallel, and it is obvious that they will not intersect before the line closest to the drill hole intersects another line, the bisector that is closest to the drill hole is accepted as the polygon boundary.

7) In areas where drill holes are separated by distances greater than two times the radius of influence, an eight-sided polygon (octagon) form is drawn around the hole location, representing the maximum area of influence. This step may also be altered by rules such as those in Table 3.22.

8) Drill holes along the periphery of the ore body are extrapolated to the radius of influence and the octagonal form is drawn around the drill hole.

If ore is defined as that material for which the grade is

$$g \geq 0.6\%$$

then for this bench the projected tonnage is 1,990,000 st at an average grade of 0.93. A tonnage factor of 13 ft^3/st (specific gravity SG = 2.47) and a 40 ft bench height has been assumed.

It is obvious that the zone/radius of sample influence is over ridden under a number of special conditions. Some examples of such special conditions are:

– Grades should not be projected from one type of formation, mineralization, rock type, etc. to another.

– Sample grades should not be projected from one side of a post mineralization structure such as a fault to the other.

Rules should be determined to deal with assigning a metal grade to in-place material which is less than a full bench height thick, such as near the surface of the deposit.

Using a computer, lists like Table 3.22 can automatically be considered as well as rules concerning mineralization controls for the specific deposit. However, it is very complicated to have a computer draw lines representing polygon boundaries and to assign area grades according to the procedures described previously.

3.8 BLOCK MODELS

3.8.1 *Introduction*

Basic to application of computer techniques for grade and tonnage estimation is the visualization of the deposit as a collection of blocks. Such a block model is shown in Figure 3.51.

Some guidance for the size of the blocks chosen has been provided by David (1977).

Typically in the profession, people like to know as much as possible about their deposit and consequently they ask for detailed estimation on the basis of the smallest possible blocks. This tendency, besides being possibly unnecessarily expensive will also bring disappointing results. One will find that small neighboring block are given very similar grades. One should remember that as the size of a block diminishes, the error of estimation of that block increases. Also, dividing the linear dimensions of a block by 2, multiplies the number of blocks to be estimated and probably the system of equations to be solved by 8! As a rule of thumb, the minimum size of a block should not be less than ¼ of the average drill hole interval, say 50 ft blocks for a 200 ft drilling grid and 200 ft for an 800 ft drilling grid.

The height of the block is often that of the bench which will be used in mining. Furthermore the location of the blocks depends on a variety of factors. For example a key elevation

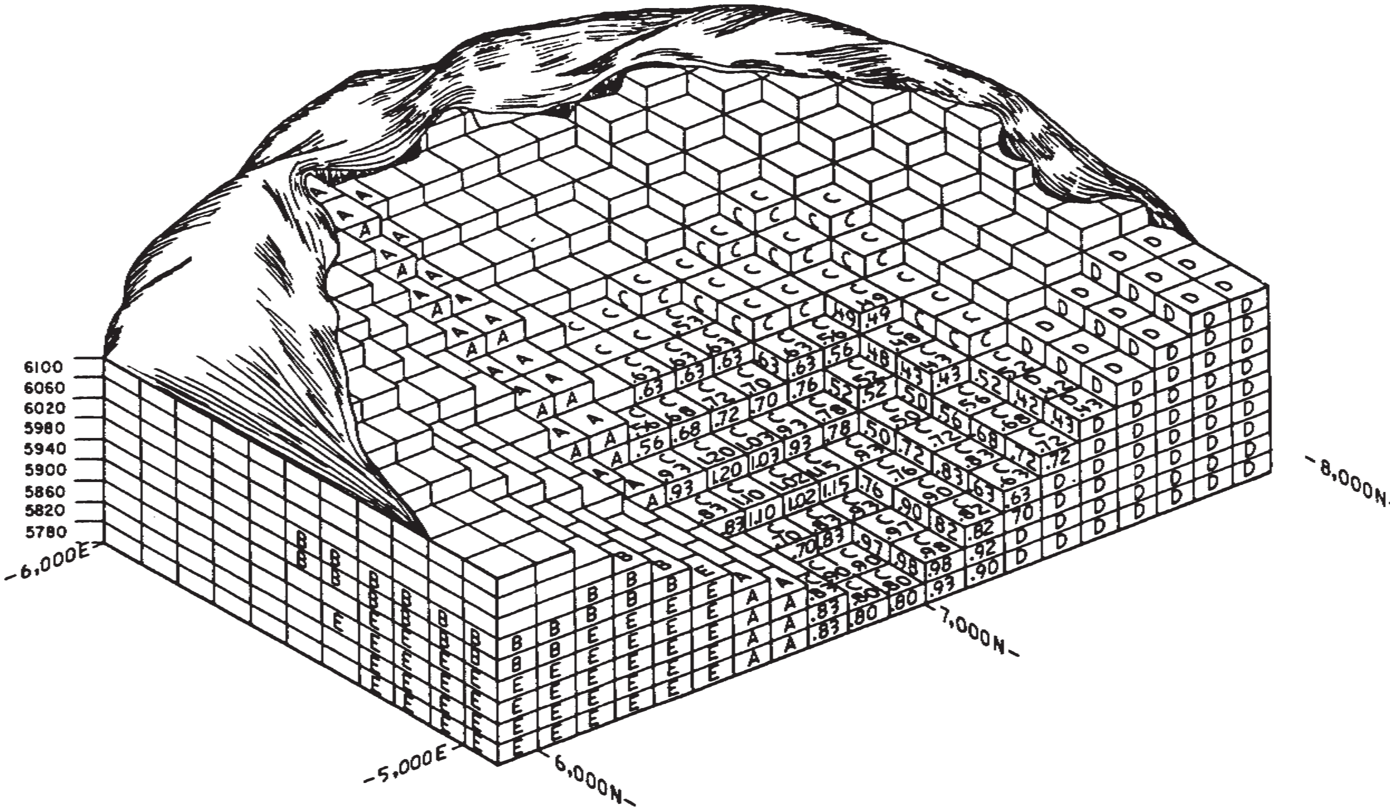


Figure 3.51. Diagrammatic view of a 3-D block matrix containing can orebody (Crawford & Davey, 1979).

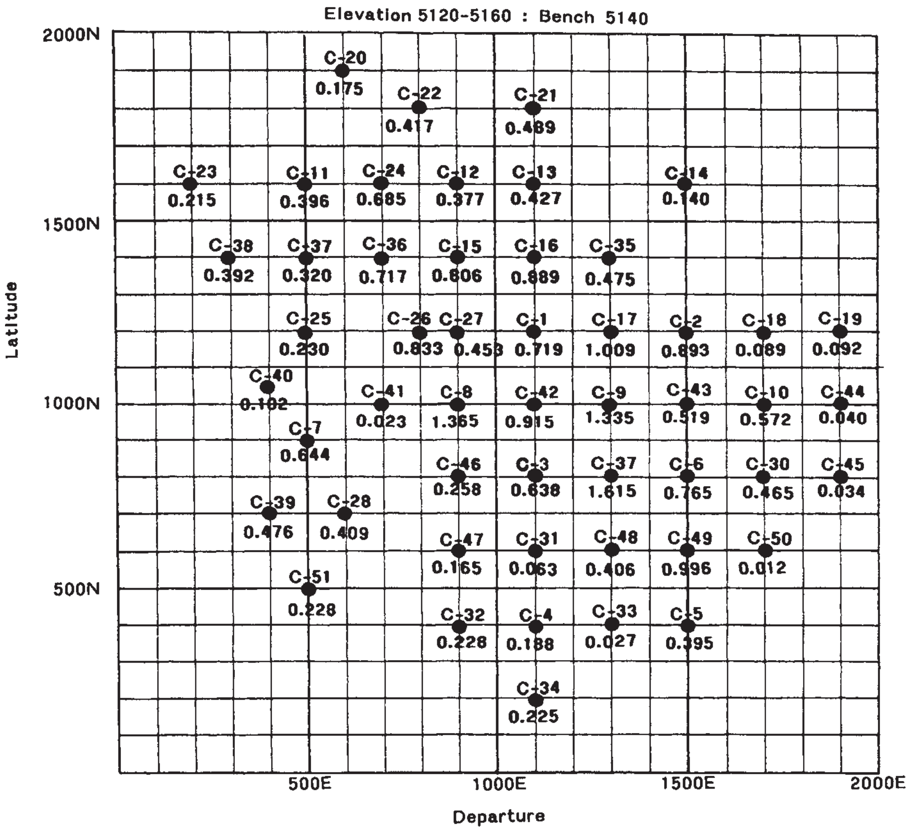


Figure 3.52. Block grid superimposed on Figure 3.45 (Hughes & Davey, 1979).

might be based upon overburden ore contact, the interface between types of mineralization (oxides-sulfides), high grade-low grade zones, etc.

Superposition of a 100 ft × 100 ft block grid on the drill hole data from Figure 3.45 is shown in Figure 3.52. As can be seen, some of the blocks have drill holes in them but most do not.

Some technique must be used to assign grades to these blocks. The tonnage of each block can be easily found from the block volume (the same for all blocks) and the tonnage factor (which may vary). Two techniques will be discussed in this section and an additional one in Section 3.10. They are all based upon the application of the 'sphere of influence' concept in which grades are assigned to blocks by 'weighting' the grades of nearby blocks. Variations in how the weighting factors are selected distinguish the three methods. A simplification which will be made in this discussion is to consider blocks as *point* values rather than as *volumes*. This distinction is illustrated in Figure 3.53. By treating the block as a point one would make one calculation of average block grade based upon the distance from the block center to the surrounding points. If the block is divided into a mesh of smaller blocks, the calculation would be made for each sub-block and the results summed. In the literature this volumetric integration is denoted by integral or summation symbols. Hughes & Davey (1979) has

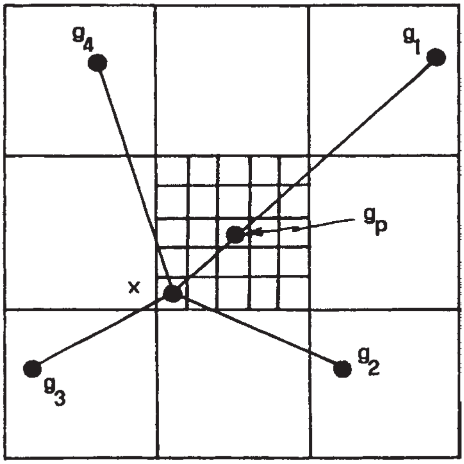


Figure 3.53. Block versus point representation.

indicated that the difference between the point and volume approach is small. We have chosen to take the least complicated approach in presenting the principles. Furthermore, a two-dimensional approach will be focussed upon with only passing reference to extensions into 3 dimensions. The examples used will focus on assignment of grades for a bench using composite grades for that bench alone. Grades lying above or below the bench in question will not be included in the calculations. Finally unless specifically mentioned, all of the grades will be assumed to belong to the same mineralization type and are all useable in assigning grades to the blocks, i.e. there are no characteristics which eliminate certain values (change in mineralization, formation, rock type, structural features). The reader will see how these can be considered.

3.8.2 *Rule-of-nearest points*

The polygon approach described in the previous chapter is an example of the rule-of-nearest points. The area surrounding a drill hole is defined in such a way that the boundary is always equidistant from nearest points. Although computer programs now do exist for doing this procedure, Hughes & Davey (1979) suggests that little accuracy is lost using a regular grid. The computer calculates the distances from the block centers to the surrounding known grade locations, and assigns the grade to the block of the closest grade. If the closest distance is greater than R, no value is assigned. In some cases, the block center may be equidistant from two or more known grades. A procedure must be established to handle this. Sometimes an average value is assigned.

Figure 3.54 shows the application of a computerized polygonal interpolation to the composited values shown as level 5140 in Figure 3.45. If the block contains a hole, it is assigned that value. Blocks without holes are assigned the value of the nearest hole within a 250 ft radius. For blocks having centers outside of this radius a value of 0 has been assigned. The shaded area has been interpolated as mineralization $\geq$ 0.6% Cu. Because the distance from block to composite is computed from the block center, results vary slightly from the polygons defined in Figure 3.50. Accumulation of blocks with projected grades $\geq$ 0.6% Cu is calculated as 2,033,778 st at an average grade of 0.92%.

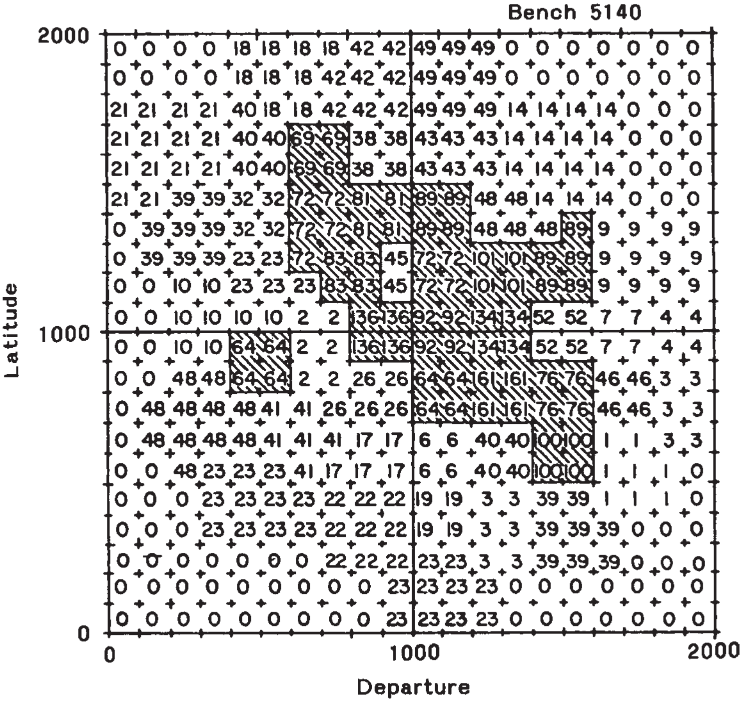


Figure 3.54. Computer generated polygons for Figure 3.45 (Hughes & Davey, 1979).

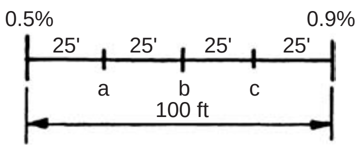


Figure 3.55. Simple example for grade calculation.

3.8.3 *Constant distance weighting techniques*

In the previous technique, the grade was assumed to remain constant over a region extending halfway to the adjacent hole. As the boundary between blocks is crossed, the grade drops to that in the adjacent region. The grade at a point was determined only by the closest grade and none other. A more sophisticated approach would be to allow all of the surrounding grades to influence grade estimation at a point. Figure 3.55 illustrates the assignment of grades along a line between two known grades. Assuming a linear change in grade between the two known grades (Fig. 3.56) one can calculate the expected grades at points a, b, and c. The formula used to calculate this can be written as

$$g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i}}{\sum_{i=1}^{n} \frac{1}{d_i}} \tag{3.18}$$

where g_i is the given grade at distance d_i away from the desired point.

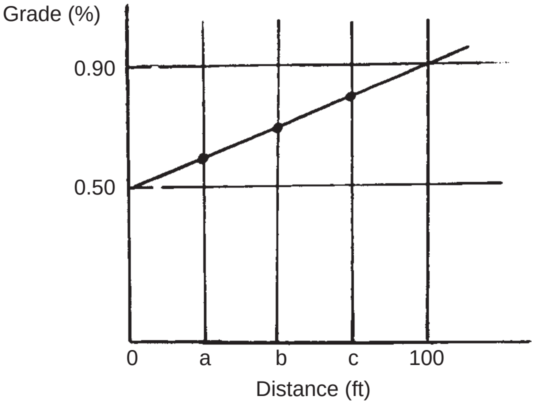


Figure 3.56. Linear variation of grade with separation distance.

For point a

$$g_a = \frac{\frac{0.5}{25} + \frac{0.9}{75}}{\frac{1}{25} + \frac{1}{75}} = \frac{1.5 + 0.9}{3 + 1} = \frac{0.24}{4} = 0.6\%$$

For point b

$$g_b = \frac{\frac{0.5}{25} + \frac{0.9}{75}}{\frac{1}{25} + \frac{1}{75}} = \frac{1.4}{2} = 0.7\%$$

For point c

$$g_c = \frac{\frac{0.5}{75} + \frac{0.9}{25}}{\frac{1}{75} + \frac{1}{25}} = \frac{3.2}{4} = 0.8\%$$

The two-dimensional application of this to the Hughes & Davey (1979) data is shown in Figure 3.57. The calculated grade at the point is given by 0.45%.

This method is called the *inverse distance* weighting technique. The influence of surrounding grades varies inversely with the distance separating the grade and the block center.

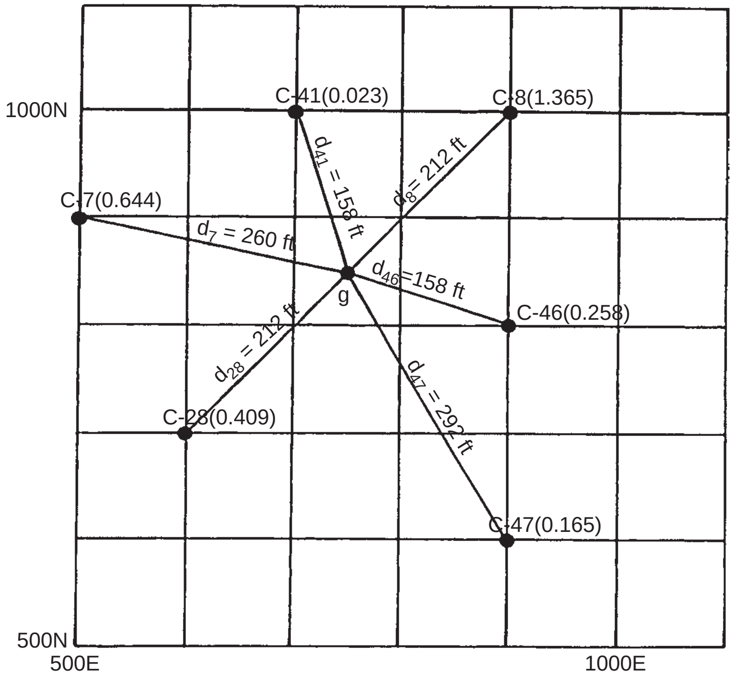
It is obvious that the grade of the block should be more similar to nearer points than those far away. To emphasize this dependence, the weighting with distance can be increased. This is done by changing the power of d_i in Equation (3.18). If the dependence varies inversely with the square of the distance rather than linearly, Equation (3.18) becomes

$$g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^2}}{\sum_{i=1}^{n} \frac{1}{d_i^2}} \tag{3.19}$$

This is the commonly used inverse distance squared (IDS) weighting formula.

Applying it to the calculation of grades at points a, b, and c along the line (Fig. 3.55) as before one finds that

$$g_a = \frac{\frac{0.5}{(25)^2} + \frac{0.9}{(75)^2}}{\frac{1}{(25)^2} + \frac{1}{(75)^2}} = \frac{4.5 + 0.9}{10} = 0.54\%$$



$$g = \frac{\frac{0.644}{260} + \frac{0.023}{158} + \frac{1.365}{212} + \frac{0.258}{158} + \frac{0.165}{292} + \frac{0.409}{212}}{\frac{1}{260} + \frac{1}{158} + \frac{1}{212} + \frac{1}{158} + \frac{1}{292} + \frac{1}{212}}$$

$g = 0.450\%$

Figure 3.57. Application of the inverse distance technique.

$$g_{\mathrm{b}} = \frac{\frac{0.5}{(60)^2} + \frac{0.9}{(50)^2}}{\frac{1}{(50)^2} + \frac{1}{(50)^2}} = \frac{0.5 + 0.9}{2} = 0.70\%$$

$$g_{\mathrm{c}} = \frac{\frac{0.5}{(75)^2} + \frac{0.9}{(25)^2}}{\frac{1}{(75)^2} + \frac{1}{(25)^2}} = \frac{0.5 + 8.1}{10} = 0.86\%$$

It is obvious that the results are quite different from before. Applying the technique to the 2-D example from Hughes & Davey (Fig. 3.57), one finds that

$$g = \frac{\frac{0.644}{(260)^2} + \frac{0.023}{(158)^2} + \frac{1.365}{(212)^2} + \frac{0.258}{(158)^2} + \frac{0.165}{(292)^2} + \frac{0.409}{(212)^2}}{\frac{1}{(260)^2} + \frac{1}{(158)^2} + \frac{1}{(212)^2} + \frac{1}{(158)^2} + \frac{1}{(292)^2} + \frac{1}{(212)^2}} = 0.412\%$$

If one were to select a different power for d, the results would change. The general formula is

$$g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^m}}{\sum_{i=1}^{n} \frac{1}{d_i^m}} \tag{3.20}$$

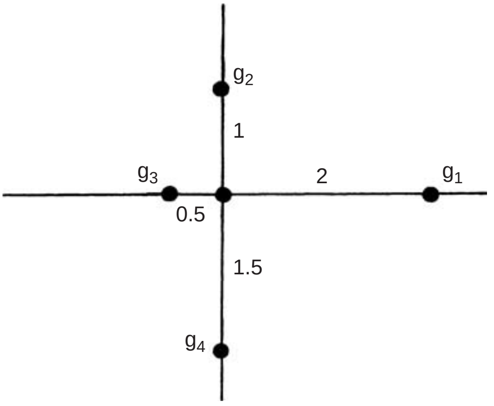


Figure 3.58. Simple example for applying the inverse distance squared (IDS) technique.

For linear dependence $m = 1$, quadratic (squared) dependence, $m = 2$, etc. The value $m = 2$ has been found to be suitable for a number of different kinds of deposits and is widely used. In practice a distribution of m values will be obtained from which a mean can be determined and a best value selected.

To this point in the discussion it has been assumed that the weighting function is independent of the angular position of the known grades with respect to the unknown. Such a function is called isotropic (independent of orientation). This is true for many deposits. For others however the variation of grade with distance does depend upon direction. Thus in one direction, say N-S, the best value for m is m_1 whereas in the E-W direction it would be m_2. Such a deposit would be termed anisotropic. Procedures are available for including these effects. Their discussion is beyond the scope of this book.

Before proceeding, let's take a closer look at the inverse distance squared formula.

$$g = \frac{\sum_{i=1}^{n} \frac{g_i}{d_i^2}}{\sum_{i=1}^{n} \frac{1}{d_i^2}}$$

It will be applied to the simple case shown in Figure 3.58.

Expanding the formula yields

$$g = \frac{\frac{g_1}{4} + \frac{g_2}{1} + \frac{g_3}{0.25} + \frac{g_4}{2.25}}{\frac{1}{4} + \frac{1}{1} + \frac{1}{0.25} + \frac{1}{2.25}}$$

The denominator becomes 5.694 and the equation can be written as

$$g = 0.044g_1 + 0.176g_2 + 0.702g_3 + 0.078g_4$$

This can be rewritten as

$$g = a_1g_1 + a_2g_2 + a_3g_3 + a_4g_4 \tag{3.21}$$

where

$$a_i = \frac{\frac{1}{d_i^2}}{\sum_{i=1}^{4} \frac{1}{d_i^2}} \tag{3.22}$$

The coefficients are:

$$a_1 = 0.044$$

$$a_2 = 0.176$$

$$a_3 = 0.702$$

$$a_4 = 0.078$$

The sum of the coefficients

$$\sum_{i=1}^{4} a_i = 0.044 + 0.176 + 0.703 + 0.078 = 1 \tag{3.23}$$

will always equal 1. Furthermore $0 \leq a_i \leq 1$.

In the following section, it will be shown that the geostatistical approach to grade estimation yields the same equation form. The coefficients a_i are simply determined in a different fashion. The constraints on a_i are the same.

To this point, it has been attempted to simply demonstrate how the method works. Some further words are required about the application in practice. Some rules in this regard are given by Hughes & Davey (1979) in Table 3.23. An example showing the application of the following rules:

(1) an angular exclusion of 18° (excludes G_3 and G_5),
(2) maximum of seven nearest holes (excludes G_1 and G_8), and
(3) power $m = 2$

is given in Figure 3.59.

Figure 3.60 shows an inverse distance squared ($m = 2$) computer evaluation of level 5140. The rules used for the simulation are:

$$\text{Radius of influence} = 250\,\text{ft}$$

$$m = 2$$

$$\text{Angular exclusion angle} = 18^\circ$$

The accumulation of blocks ≥0.6% Cu is calculated to be 2,003,000 st at an average grade of 0.91% Cu.

3.9 STATISTICAL BASIS FOR GRADE ASSIGNMENT

In the previous section one technique for assigning grades to blocks, based upon distance dependent weighting coefficients, was discussed. The application depended upon selecting the power m and a radius of influence for the samples. In some cases a value for m is just picked (often 2), and in others the data set is scanned. Little was mentioned as how to select a value for R. If $m = 2$ is used, the decrease of influence with distance is quite rapid and

Table 3.23. Example of inverse distance squared interpolation rules (Hughes & Davey, 1979).

1. Develop rock type distance factors. These factors are sets of A, B, and C coefficients for equations of the form $AX^2 + BX + C = Y$, where Y is the average standard deviation between grades and X is the distance between the sample points. This is done for all combinations of formations plus within each formation.
2. Develop geologic model with rock type code for each block being evaluated. Rock type codes are assigned to each composite value.
3. Radius of influence equals R ft and angle of exclusion equals $\alpha°$.
4. The block must pass one of the following in order to be assigned a grade:
   a) The block must be within R ft of a composite.
   b) The block is within R ft of a line connecting two composites which are within $3R$ ft of each other.
   c) The block is within R ft of a line connecting two composites that are within $3R$ ft of a third composite.
   d) The block is inside a triangle formed by three composites, any two legs of which are equal to or less than $3R$ ft long.
5. Collect all assay composites for the level that are within $5R$ ft of the center of the block.
6. Count the number of composites having the same rock type as the block. A rock type the same as the block is defined as:
   a) The rock type of the composite matches the rock type of the block.
   b) The rock type of the block is unknown or undefined.
7. If no composites are found to match the rock type of the block, extend the radius of search outward by R ft increments until one or more composites are found within an increment. Add these composites to the list of ones affecting the block grade assignment.
8. Compute distances from the block to each composite having a different rock type than the block, such that the new distance would be equivalent to the two points being in the same rock type.
   If the equivalent distance is less than the original distance, use the original distance. The original distance rather than the equivalent distance will be used by the minimum angle screening.
9. Compute the azimuths from the block to each composite influencing the assay assignment.
10. For each assay of the mineralization model, compute the angle between each pair of composites having data for the assay. Check to see if the angle is less than $\alpha°$. If the angle is less than $\alpha°$ and:
    a) the rock type of the closer composite matches the rock type of the block, the more distant composite is rejected.
    b) both composites match the rock type of the block and only two composites match the rock type of the block, both composites are retained.
    c) the rock type of the closer composite matches the rock type of the block, the more distant composite is rejected.
    d) the rock type of neither composite matches the rock type of the block, the more distant composite is rejected.
11. The grade assignment for the block is computed as:
    $$G = \sum_i \left(G_i/D_i^2\right) / \sum_i (1/D_i^2)$$
    where G_i is the sample assay value and D_i is the equivalent distance to the i-th composite.
    a) Unless there is a nonzero composite value within or on the boundary of the block, in which case that composite will be used directly.
    b) Unless there is only one composite, in which case the closest composite from the reject list having the same rock type is included. If no second composite can be found with the same rock type, the closest composite from the reject list is included.
12. If the resulting grade assignment is zero, it will be increased to the smallest nonzero number which can be represented in the model.

the use of a large value is not so serious. The minimum value of R is determined by the need to include a sufficient number of points for the calculations. This obviously varies with the drilling pattern. The field of geostatistics has contributed a number of techniques which can be used. Of particular importance is a way to evaluate the radius of influence R and

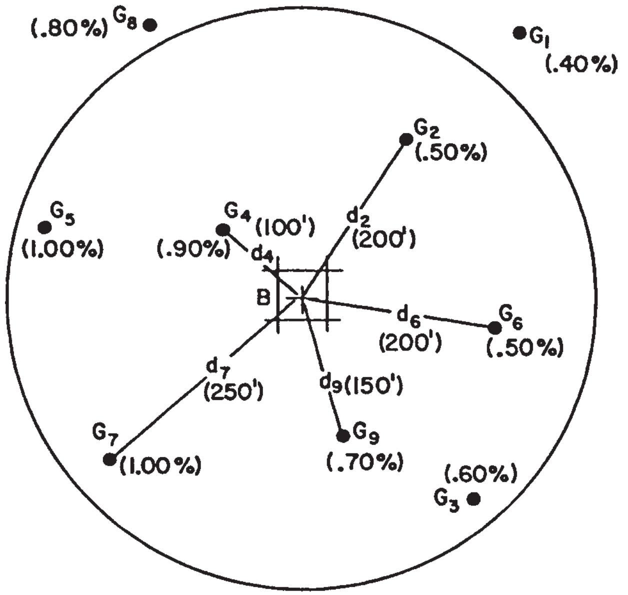


Figure 3.59. A more realistic IDS application (Hughes & Davey, 1979).

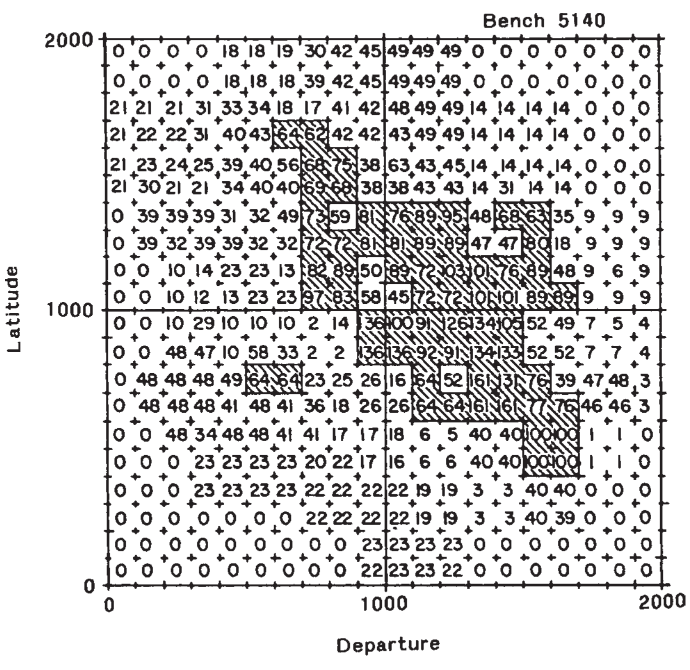


Figure 3.60. Inverse distance squared interpolation for the composited samples in Figure 3.45 (Hughes & Davey, 1979).

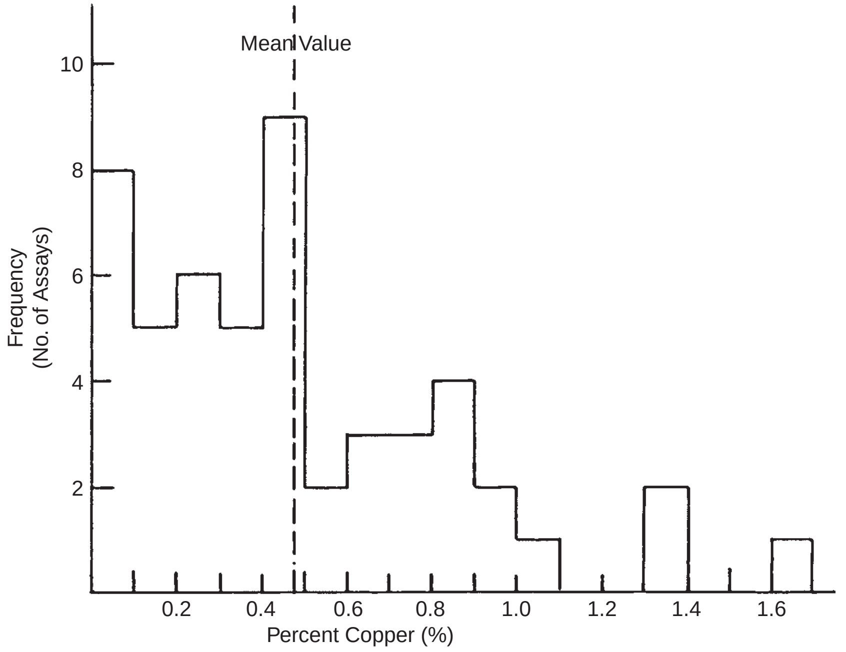


Figure 3.61. Histogram representation of the 5140 bench grades.

assigning grades to blocks. Unfortunately, due to the highly mathematical nature of their presentation, the level of understanding and appreciation of the technique within much of the mining industry is poor. In this section the authors have attempted to clarify some of these concepts.

3.9.1 *Some statistics on the orebody*

One of the first things which can and should be done is to see how the grades are distributed. It is most easily done by plotting a histogram of the data. This has been done in Figure 3.61 using the data from bench 5140 (Table 3.21).

The *average* grade $\bar{g}$ is calculated using

$$\bar{g} = \frac{1}{n}\sum_{i=1}^{n} g_i \tag{3.24}$$

where n is the number of samples and g_i is the individual grades. In this case the average grade is

$$\bar{g} = 0.477\%$$

It has been superimposed on Figure 3.61. If the grade distribution had been truly normal then a bell shaped curve centered around the average value would be expected. Here, this is not the case. There are many values clustered below the mean and a long tail into the high values. This is termed a positive skew and is quite common for low grade deposits.

Table 3.24. Grades from bench 5140 arranged in increasing order.

i	Grade (% Cu)	C_f (%)	i	Grade (% Cu)	C_f (%)
1	0.012	1.0	27	0.417	52.0
2	0.023	2.9	28	0.427	53.9
3	0.027	4.9	29	0.453	55.9
4	0.034	6.9	30	0.465	57.8
5	0.040	8.8	31	0.475	59.8
6	0.089	10.8	32	0.476	61.8
7	0.092	12.7	33	0.489	63.7
8	0.099	14.7	34	0.519	65.7
9	0.102	16.7	35	0.572	67.6
10	0.140	18.6	36	0.638	69.6
11	0.165	20.5	37	0.644	71.6
12	0.175	22.5	38	0.685	73.5
13	0.180	24.5	39	0.717	75.5
14	0.215	26.5	40	0.719	77.5
15	0.224	28.4	41	0.765	79.4
16	0.225	30.4	42	0.806	81.4
17	0.228	32.4	43	0.833	83.3
18	0.230	34.3	44	0.889	85.3
19	0.252	36.3	45	0.893	87.3
20	0.320	38.2	46	0.915	89.2
21	0.377	40.2	47	0.996	91.2
22	0.392	42.2	48	1.009	93.1
23	0.395	44.1	49	1.335	95.1
24	0.396	46.1	50	1.365	97.1
25	0.406	48.0	51	1.615	99.0
26	0.409	50.0			

The degree of departure from normality can be checked by plotting the values on standard probability paper. First one arranges the grades in order as in Table 3.24. Next the corresponding cumulative frequency of the grades are calculated using

$$C_f = \frac{100(i - ½)}{n} \tag{3.25}$$

where i is the i-th observation, n is the total number of observations, and C_f is the cumulative frequency.

If n is large it is not necessary to plot every point (every 5-th or 10-th point may be enough). The results are plotted in Figure 3.62. As can be seen, there are departures from a straight line particularly at the lower grades. For grades above 0.3% Cu, the fit is fairly good. If the entire distribution is to be represented, then measures must be taken to convert it into a normal distribution. Two types of logarithmic transformations may be applied to such skewed (whether negatively or positively) distributions. In the simplest case, one plots the natural logarithm of the grade (ln g_i) versus cumulative frequency on log probability paper (Fig. 3.63).

It is observed for grades greater than about 0.3% Cu, a straight line can be fitted. However for lower grades, the points fall below the curve. Hence the simple transformation of ln g_i

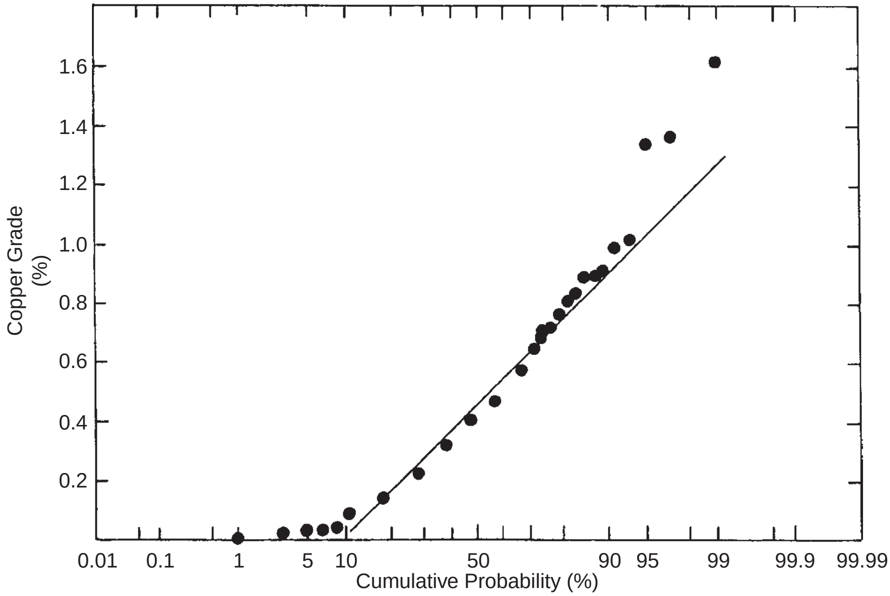


Figure 3.62. Copper grade versus cumulative probability for the 5140 bench grades.

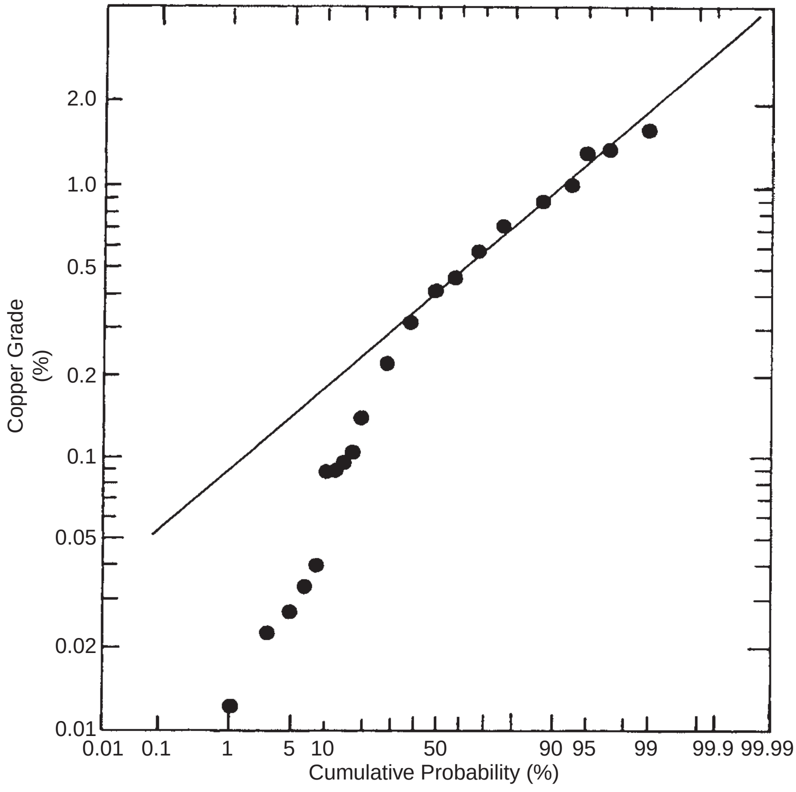


Figure 3.63. The logarithm of copper grade versus cumulative probability.

doesn't yield the desired normal distribution. The next step is to take the natural log of the grade plus an additive constant β and plot ln $(g_i + \beta)$ on probability paper.

If the number of samples is large enough, one can estimate β using the following formula and values from Figure 3.62:

$$\beta = \frac{m^2 - f_1 f_2}{f_1 + f_2 - 2m} \tag{3.26}$$

where m is the grade at 50% cumulative frequency, f_1 is the sample value corresponding to 15% cumulative frequency, and f_2 is the sample value corresponding to 85% cumulative frequency.

In general f_1 corresponds to frequency P and f_2 to frequency $1 - P$. In theory any value of P can be used but one between 5–20% gives the best results.

Applying this rule, one finds that

$$\beta = \frac{(0.409)^2 - 0.10 \times 0.81}{0.10 + 0.81 - 2 \times 0.409} = \frac{0.086}{0.092} = 0.935\%$$

The resulting value ln $(g_i + 0.935)$ provides a high degree of normalization to the grade distribution (Fig. 3.64).

The use of log-normal distributions introduces complexities which are beyond the scope of this book. For instance one should be aware that the grade at 50% probability on a log-normal distribution graph, would represent the median – also called the geometric mean – and not the true (arithmetic) mean of the distribution. It will be assumed that the grades from bench 5140 can be adequately represented by a normal distribution.

As can be seen in Table 3.24, there is a large spread or range in the grades.

The *range* is from 0.012% to 1.615%. The *variance* s^2, obtained using

$$s^2 = \frac{1}{n}\sum_{i=1}^{n}(g_i - \bar{g})^2 \tag{3.27}$$

is found equal to

$$s^2 = 0.1351(\%)^2$$

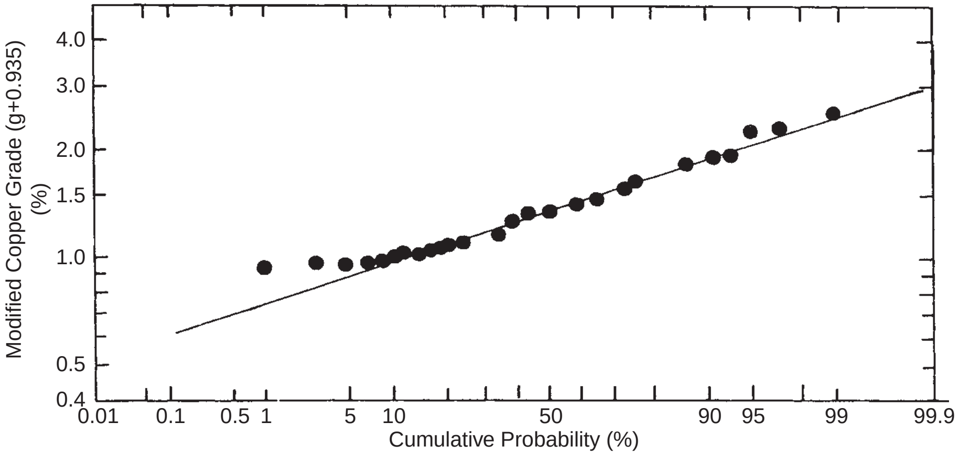


Figure 3.64. Plot of the logarithm $(g_i + \beta)$ versus cumulative probability for the grades of bench 5140.

The standard deviation (s), which is just the square root of the variance, becomes

$$s = 0.368\%$$

It will be recalled that about 68% of the grades should be contained within $\bar{g} \pm s$ and 95% within $\bar{g} \pm 2s$ given a normal distribution.

This traditional statistical approach has treated all the samples as a large group with no special notice being paid to their relative positions within the group. Such attention will be paid in the next section.

3.9.2 *Range of sample influence*

When using inverse distance weighting techniques, the range of influence of a sample is, in theory, infinite. In practice some finite range is assigned. The question arises as to whether a more quantitative way of determining the effective sample range could be devised? The geostatistical approach described in this section provides one way. The basic logic involved will first be described followed by an example.

If when sampling an orebody, the samples are collected close together, one might expect the resulting assay values to be similar. On the other hand, if they are collected far apart little similarity would be expected. In between these two extremes, one would expect some sort of functional relationship between grade difference and separation distance to apply. If the function could be determined, then the distance (influence range) at which samples first became independent of one another could be found. The basic procedure (Barnes, 1980) would be:

1. Decide on separation distances h into which sample pairs would be grouped. These distances are often called lags. For example, separation distances of 100 ft, 200 ft, 300 ft, etc.

Although each lag is thought of as a specific distance, in practice, the lag distance usually represents the mean of a distance class interval. In other words, the lag distance of 15 m (50 ft) may represent all pairs of samples falling between 11½ and 19½ m (37½ and 62½ ft) apart. Such a practice is necessitated by the uneven spacing of most samples, especially when computing directional variograms that are not parallel or normal to a roughly rectangular sampling pattern.

2. Identify the pairs falling within a particular group. Figure 3.65 illustrates a simple case of n samples separated by a constant lag distance h. Various pairs can be formed. There are $n-1$ pairs of distance h apart, $n-2$ pairs at separation distance $2h$, $n-3$ pairs separated by $3h$, etc.

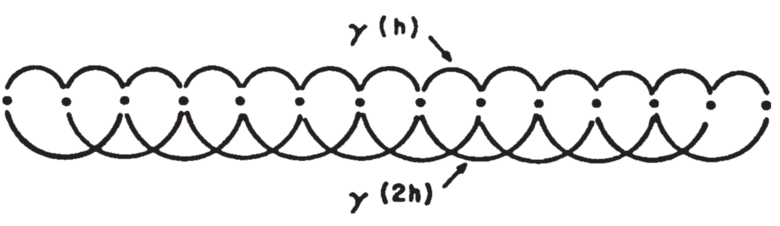


Figure 3.65. Variance computational scheme using sample pairs a given distance apart (Barnes, 1979).

3. Calculate the grade difference $g_i - g_j$ for each of the pairs within each group. It will be found that a distribution of differences exists. As was done before, the average, the variance and the standard deviation could all be calculated. A particular type of variance defined as

$$s^2(h) = \frac{1}{n(h)} \sum^{n} (g_i - g_j)^2 \tag{3.28}$$

where $n(h)$ is the number of pairs in the group of lag h, $s^2(h)$ is the variance for pairs with lag h, and g_i is the grade at point i of the pair, will be used. For mathematical convenience, one-half of $s^2(h)$, denoted by the symbol $\gamma(h)$ will be used:

$$\gamma(h) = \frac{1}{2n(h)} \sum^{n} (g_i - g_j)^2 \tag{3.29}$$

This is called the geostatistical variance or the semi-variance (half of the variance).

4. Once values of γ have been found for each of the different groups (called cells), the next step is to plot the results. The plot of γ versus average lag h is called a *variogram* or more properly a *semi-variogram*. In this book the term variogram is retained.

5. The final step is to express the relationship between γ and h in some type of useable form. The value of h beyond which little or no change in γ is observed is called the range of influence '*a*'.

3.9.3 *Illustrative example*

To illustrate these concepts an example using the N-S data pairs in Figure 3.45 will be worked. In viewing the plan map it is clear that most of the holes are spaced 200 ft apart. Table 3.25 summarizes the lags and the number of corresponding pairs.

It is found convenient to consider the data in thirteen cells (groups) incremented from each other by 100 ft. The cells are summarized in Table 3.26.

The location of the 19 pairs at a separation distance of 600 ft are shown in Figure 3.66. The calculation of $\gamma(600)$ is shown in Table 3.27.

Table 3.25. Pairs and distances used for computing a N-S variogram.

Distance (ft)	No. of pairs
200	31
300	1
350	1
400	26
500	1
600	19
700	2
800	12
900	1
1000	8
1100	1
1200	6

Table 3.26. Cells used in the example.

Cell	Separation distance (ft)	No. of pairs	Average separation distance (ft)
1	0 → 99	0	–
2	100 → 199	0	–
3	200 → 299	31	200
4	300 → 399	2	325
5	400 → 499	26	400
6	500 → 599	1	500
7	600 → 699	19	600
8	700 → 799	2	700
9	800 → 899	12	800
10	900 → 999	1	900
11	1000 → 1099	8	1000
12	1100 → 1199	1	1100
13	1200 → 1299	6	1200

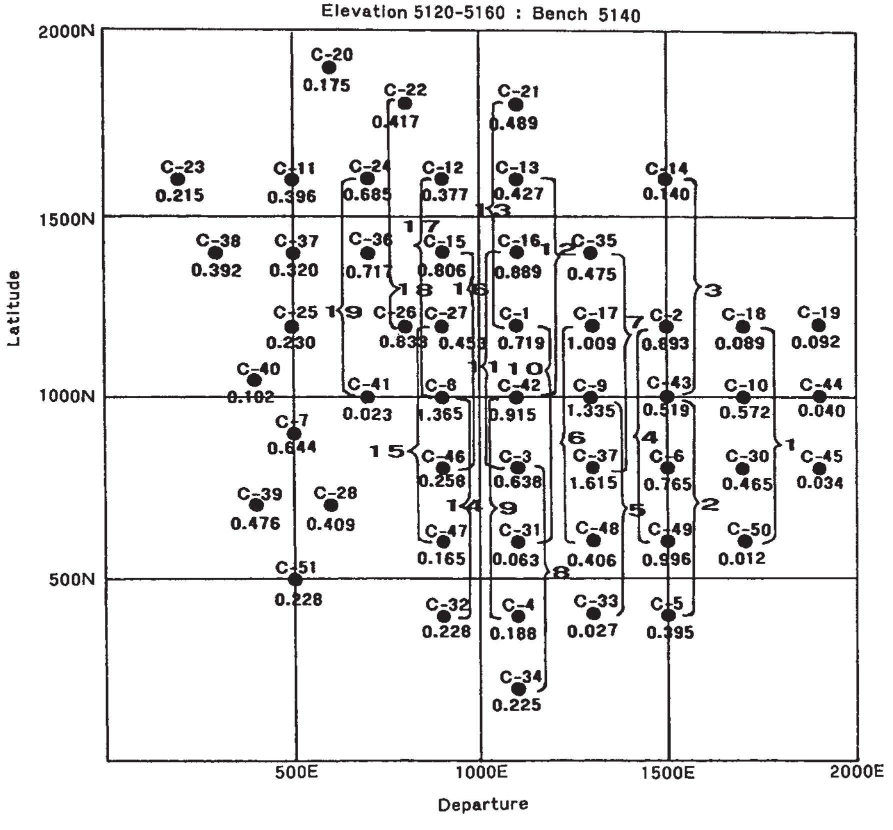


Figure 3.66. Sample calculation using the N-S pairs 600 feet apart.

Table 3.27. Steps in the determination of $\gamma(600)$ for use in the N-S variogram.

Pair	Grades $g_i - g_j$ ($\% \times 10^3$)	Grade difference ($\% \times 10^3$)	(Grade difference)^2 ($\% \times 10^3)^2$
1	89−12	77	5929
2	519−395	124	15,376
3	990−893	97	9409
4	519−140	379	143,641
5	1335−27	1308	1,710,864
6	1009−406	603	363,609
7	1615−475	1140	1,299,600
8	638−225	413	170,569
9	915−188	727	528,529
10	719−63	656	430,336
11	889−638	251	63,001
12	915−427	488	238,144
13	719−489	230	52,900
14	1365−224	1141	1,301,881
15	453−165	288	82,944
16	806−258	548	300,304
17	1365−377	988	976,144
18	833−417	416	173,056
19	685−23	662	438,244
		Avg GD = 555	$\Sigma(\mathrm{GD})^2 = 8{,}304{,}480$

$$\gamma(600) = \frac{8,304,480}{23 \times 19 \times 10^6} = 0.2185 \text{ percent}$$

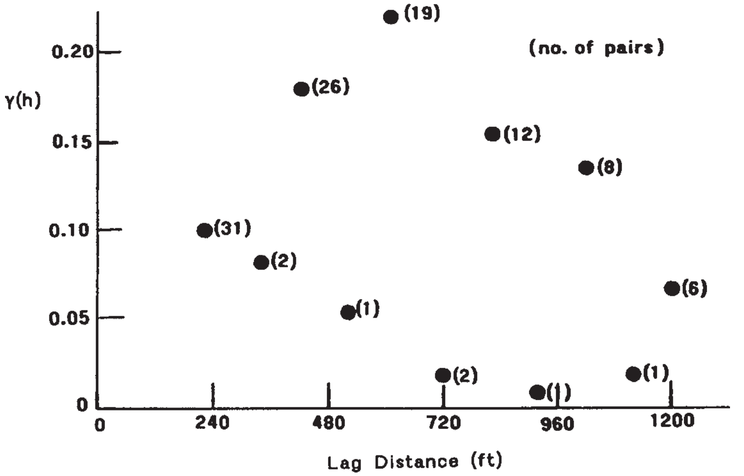


Figure 3.67. The resulting N-S variogram.

In Figure 3.67, the number of data pairs represented by each point is plotted. It is important that sufficient pairs are found at each lag in any direction to assure statistical significance. Ideally at least 30 such pairs are necessary to compute the variance for each lag in any given direction. Sometimes in the early sampling stage, it is difficult to find enough pairs

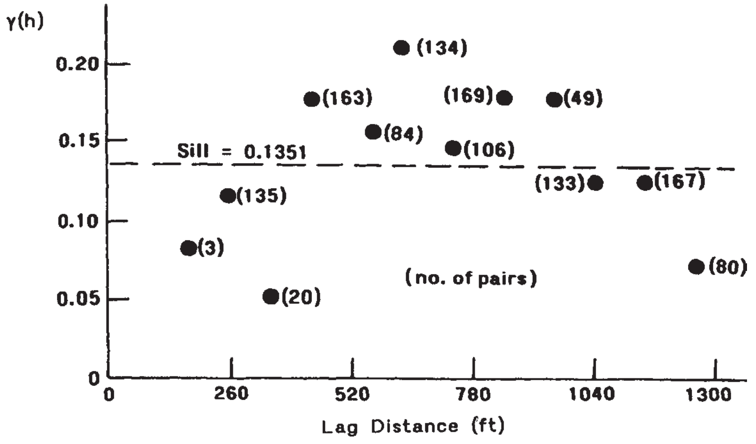


Figure 3.68. The variogram for bench 5140.

at certain lags to produce a viable variogram point, and lesser numbers may be used. A variogram program that will output a different symbol when plotting the $\gamma(h)$ value for all lags having less than 30 pairs is useful for quick recognition of less reliable points (Barnes, 1980b).

As indicated earlier, it is desired to find the value of h at which γ ceases to vary with distance. This is called the range of influence. Due to the small number of sample pairs at the higher separation distances, no particular plateau value is observed. If all directions are included (not just N-S), then a much larger number of pairs is obtained. The resulting figure is shown in Figure 3.68. A definite 'leveling off' is observed with distance although the magnitude of the plateau (termed the sill) is difficult to discern from these data. The variance for the entire set of samples was determined earlier to be 0.1351. It can be shown that this should equal the sill. Hence this line has been superimposed on the figure.

To complete the curve, the behavior in the region of the origin is needed. For samples taken very close together ($h \cong 0$), one would expect a difference in assay values due to

– lack of care in sample collection.
– poor analytical precision (limits of analytical precision),
– poor sample preparation, and
– highly erratic mineralization at low scale.

This type of variance would be expected to be present independent of the sampling distance. Its magnitude is given the symbol c_o and it is called the nugget effect. In some texts this part of the total variance is called the chaotic or unstructured variance. (Fig. 3.69).

That portion of the variogram lying between the nugget effect and the sill represents the true variability within the deposit for the given mineralization. It is called the structured variance.

A straight line extending from the Y axis to the sill has been drawn through the first few points in Figure 3.70. The value of c_o (the nugget effect) as read from the curve is 0.02. For the spherical model (of which this is an example), it has been found that the straight line

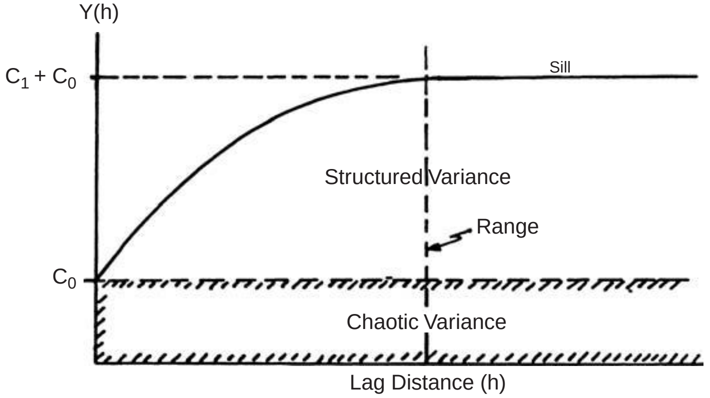


Figure 3.69. Diagrammatic representation of a spherical variogram.

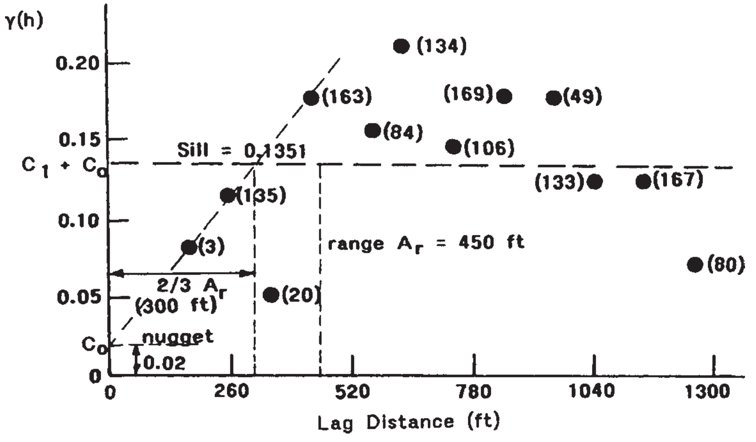


Figure 3.70. Determination of range, sill and nugget.

will intersect the sill at $h = 2/3\,a$, where a is the range. In this case, $a \cong 450$ ft. In summary:

$$c_0 = \text{nugget effect} = 0.02$$

$$c_1 + c_0 = \text{sill} = 0.135$$

$$c_1 = \text{structured variance} = 0.135 - 0.02 = 0.115$$

$$A_r = R = \text{range} = 450\,\text{ft}$$

It is clear that this process can be applied in particular directions to evaluate anisotropic behavior (or variation in range, with direction). The lag h would have an associated direction and thus become a vector quantity denoted by $\vec{h}$. Variograms are prepared for each type of mineralization within a deposit.

The evaluation process can be stopped at this point. The required value of $R = a$ can be used in polygon, inverse distance or other schemes for tonnage-grade calculation. On the other hand the variogram which reflects grade variability with distance can be used to

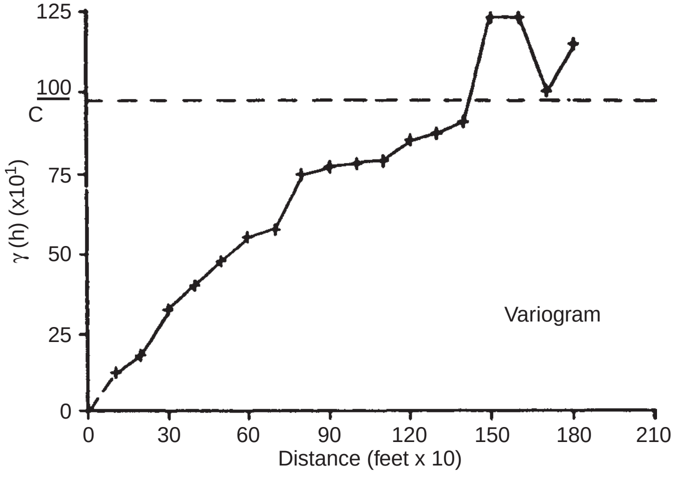


Figure 3.71. Typical variogram for a stratabound deposit (Barnes, 1979b).

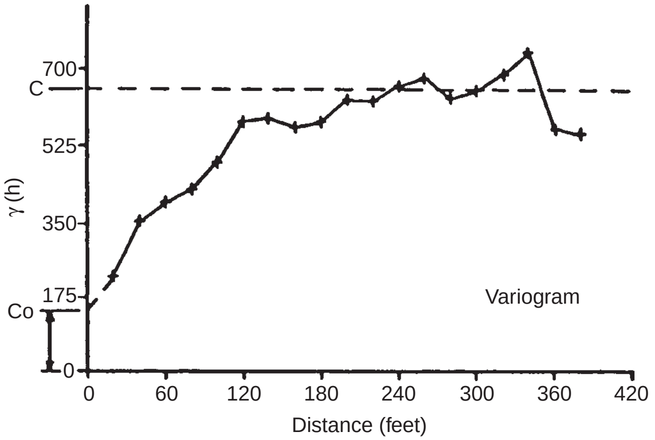


Figure 3.72. Typical variogram for a porphyry copper deposit (Barnes, 1979b).

develop weighting coefficients similar to the a_i's described earlier with respect to the inverse distance method. This process which is called kriging will be described in Section 3.10.

3.9.4 *Describing variograms by mathematical models*

Figures 3.71, 3.72, and 3.73 are real experimental variograms generated from three different types of deposits (Barnes, 1980). The slow steady growth of γ from zero in Figure 3.71 is characteristic of many stratigraphic and stratiform deposits with fairly uniform mineralization having a high degree of continuity. Figure 3.72 was generated from porphyry-copper deposit data where mineral veinlets, changes in structural intensity, and other discontinuous features created a significant nugget effect due to changes over very short distances. Beyond the short range effects, however, $\gamma(h)$ shows a fairly uniform growth curve and reaches a plateau at the sill of the variogram that is the overall variance of all the samples.

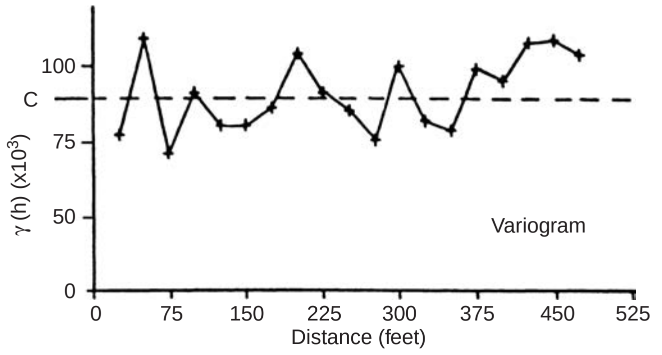


Figure 3.73. Typical variogram for a gold deposit (Barnes, 1979b).

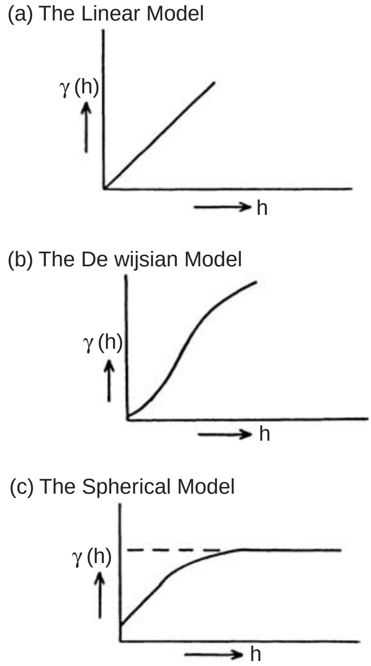


Figure 3.74. Diagrammatic representation of different variogram models (Barnes, 1979b).

The experimental variogram of Figure 3.73 is illustrative of a total random effect found in some gold deposits. The mineral continuity is nonexistent, and the samples appear to be completely independent no matter what the distance between them. Geostatistical ore reserve estimation techniques cannot make any contribution toward evaluating the deposit having a pure nugget effect since no regionalized element is present.

The change of variance with distance between samples can be read directly from the hand-drawn curve fitted through the experimental points. For computer calculations however, it is necessary to have an equation which describes the curve. The three models given in Figure 3.74, (as well as others) have been used to approximate the actual variograms.

Both the linear model and the De Wijsian model, which will produce a straight line when the lag h is plotted to log scale, imply that $\gamma(h)$ increases infinitely with increasing distances. Experience has shown that both models often accurately fit experimental variogram data near the origin, but break down when h becomes large.

The spherical model or Matheron model, as it is sometimes called, is one in which the variogram reaches a finite value as h increases indefinitely. This finite value, referred to as the sill of the spherical variogram, is the overall variance of the deposit and is reached when the grades are far enough apart to become independent of each other and act in a random manner. The spherical model has become the most important one and many practicing geostatisticians have adopted it as an almost universal model. The model has been found to adequately represent such diverse deposits as iron ore bodies, porphyry-copper deposits, stratibound lead-zinc deposits, bauxite and lateritic nickel, as well as uranium and phosphate deposits. This model will be the only one discussed further in this text.

The spherical scheme is defined by the formula

$$\gamma(h) = \begin{cases} c_1\left(\dfrac{3}{2}\dfrac{h}{a} - \dfrac{1}{2}\dfrac{h^3}{a^3}\right) + c_0 & \text{when } h \leq a \\ c_1 + c_0 & \text{when } h > a \end{cases} \tag{3.30}$$

where $c_1 + c_0 = \gamma(\infty)$ and is called the sill value, c_0 is the nugget effect (usually present), and a is the range ($a = A_r = R$) or maximum zone of influence.

3.9.5 *Quantification of a deposit through variograms*

Barnes (1979b, 1980) has summarized very nicely the types of quantitative information provided by variograms.

(a) A measure of continuity of the mineralization: A rate of increase of $\gamma(h)$ near the origin and for small values of h reflects the rate at which the influence of a sample decreases with increasing distance from the sample site. The growth curve demonstrates the regionalized element of the sample, and its smooth steady increase is indicative of the degree of continuity of mineralization.

The intersection of the curve with the origin provides a positive measure of the nugget effect of the samples from which the variogram has been generated and indicates the magnitude of the random element of the samples.

(b) A measure of the area of influence of a sample: The zone of influence of a sample is the distance or range in any direction over which the regionalized element is in effect. When samples reach a point far enough apart so as to have no influence upon each other, we have established the range or zone of influence of the sample. The quantification of the range or zone of influence in various directions has important applications in the design and spacing of development drill holes within a deposit. The total zone of influence is indicated by the point at which the $\gamma(h)$ growth curve reaches a plateau, referred to in the spherical scheme as the sill.

(c) A measure of mineral trend or mineral anisotropy of the deposit: The fact of mineral anisotropism in various types of deposits has long been recognized. The range of influence of a sample is greater along the strike or trend of the deposit than it is normal to trend. Most of the time, another anisotropism is evident in the vertical dimension. Prior to the variogram, there was no satisfactory way of determining the three-dimensional influence of a sample. With the simple process of computing variograms in different directions as well as vertically, one can readily determine not only the mineralogical trend but the magnitude of the directional changes in the zone of influence. Knowing quantitatively the mineralogical range in three dimensions, it is relatively simple to assign directional anisotropic factors that will give proper weighing to samples relative to their location from the point or block being evaluated. For example, if the range of influence along the trend is twice as great as the range normal to trend, one can multiply the distance in the normal direction by a factor of two to restore geometric isotropy in terms of the major trend direction.

3.10 KRIGING

3.10.1 *Introduction*

Prior to going into detail, it is perhaps worthwhile to review the objective and to summarize the approach to be taken. The overall problem (shown in Fig. 3.75) is that of assigning a grade g_0 to the point x_0 knowing the grades g_i at surrounding points x_i.

The objective can be simply expressed as that of determining coefficients a_i's which when multiplied by the known grades g_i and the resulting products summed will yield a best estimate of the grade g_0. The equation developed is called a linear estimator and has the form

$$g_0 = a_1 g_1 + a_2 g_2 + \cdots + a_n g_n \tag{3.21}$$

where g_0 is the grade to be estimated, g_i are the known grades, and a_i are the weighting functions.

As was discussed earlier, the inverse distance method is also of this form. The coefficients would be

$$a_i = \frac{\frac{1}{d_i^m}}{\sum_{i=1}^{n} \frac{1}{d_i^m}} \tag{3.20}$$

The distance weighting factor m is often chosen equal to 2.

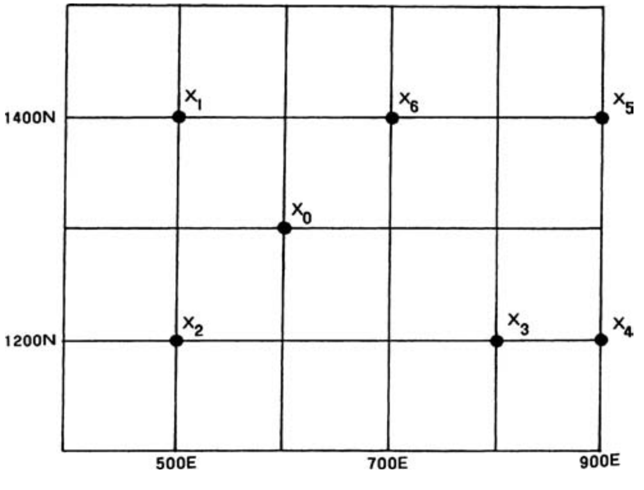


Figure 3.75. Sample locations for the point kriging example.

In the present case, we have a curve (a variogram) which expresses the variance as a function of distance. That curve can be used to calculate the total estimated variance of the grade g_0 for different combinations of the a_i coefficients. The best estimate of g_0 is that for which the variance is a minimum. A powerful advantage of this technique over that of other techniques, is that the variance is calculated as well as the estimated grade. The problem therefore boils down to finding the a_i coefficients. Such a set of coefficients must be calculated for each point requiring a grade assignment in the region under consideration. Therefore the use of a high speed computer is a definite requirement.

3.10.2 *Concept development*

For the example shown in Figure 3.75 there are six surrounding grades, hence an equation of the form

$$g_0 = a_1g_1 + a_2g_2 + a_3g_3 + a_4g_4 + a_5g_5 + a_6g_6 \tag{3.31}$$

is being sought.

Since six coefficients a_1, a_2, a_3, a_4, a_5, and a_6 must be found, at least six equations containing these six unknowns must be developed and solved. Although values of variances γ read directly from the variogram can be used directly in this process, it is found more convenient to use *covariances* σ. The covariance is related to the variance as shown in Figure 3.76.

Whereas γ (the variance) is the distance between the X axis and the curve for a given lag h, the covariance at h is the distance between the curve and the sill $c_0 + c_1$.

At a lag distance of h_0, $\gamma(h_0) = \gamma_0$ and the covariance is

$$\sigma(h_0) = c_1 + c_0 - \gamma_0 = \sigma_0 \tag{3.32}$$

For $h = 0$ (just at the location of the sample itself) the variance of the sample with itself γ is obviously equal to zero, $\gamma(0) = 0$. The corresponding covariance (σ) of the sample with itself is found either from the curve or using the following equation

$$\sigma(0) = c_0 + c_1 - \gamma(0) = c_0 + c_1 \tag{3.33}$$

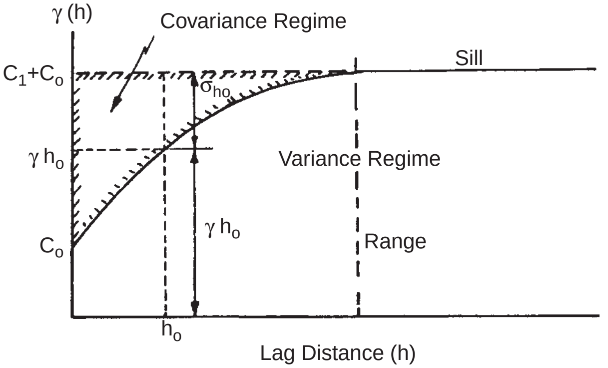


Figure 3.76. The relationship between variance and covariance.

Thus $\gamma(0)$ is just the sill value. For samples taken very close ($h=0+$) but not at the sample, the value of the variance $\gamma(0+)$ jumps to the nugget level c_0 and similarly the covariance becomes

$$\sigma(0+) = c_1 + c_0 - c_0 = c_1 \tag{3.34}$$

which is the true deposit variability. At the range of sample influence (a), $\gamma(a) = c_1 + c_0$ and the covariance becomes $\sigma(a) = 0$. The reason for using covariances in Formula (3.35) rather than the equivalent gamma values is due primarily to linear programming complexities (David, 1977). In any case, this is a very simple substitution and as such should not present understanding difficulties to the reader.

The equation for the total estimation variance σ_e^2 written in terms of the covariance is given below

$$\sigma_e^2 = \sigma_{x_0x_0} - 2\sum_{i=1}^{n} a_i\sigma_{x_0x_i} + \sum_{i=1}^{n}\sum_{j=1}^{n} a_ia_j\sigma_{x_ix_j} \tag{3.35}$$

where $\sigma_{x_0x_0}$ is the covariance between the grade at the point and itself, $\sigma_{x_0x_i}$ is the covariance between the point being considered (x_0) and the sample point x_i, a_i, a_j are the weighting coefficients, $\sigma_{x_ix_j}$ is the covariance between samples x_i and x_j, and σ_e^2 is the total variance.

As can be seen, the total variance consists of three parts:

– The covariance of the unknown grade with itself. As we have just seen, this is equal to ($c_0 + c_1$) which is the sill value and is constant.

– The weighted covariance between the unknown grade and each of the other samples. The covariance can be computed between the point and each of the samples from the variogram since the distance is known. The weighting coefficients a_i are unknown.

– The weighted covariances between each of the known samples. These can be computed from the variogram since the distances are known. The weighting coefficients a_i which apply in this region are unknown.

As stated earlier, the objective is to minimize σ_e^2 by a proper choice of the coefficients a_i. Similar to finding a minimum in many other types of engineering problems, to do this one takes a derivative, sets the resulting equation equal to zero, and solves for the unknown. For a system of equations such as this, partial derivatives with respect to each of the unknown coefficients are taken, the resulting linear equations set equal to zero, and solved for the coefficients.

Taking partial derivatives of Equation (3.35) with respect to a_i one finds that

$$-\sum_{i=1}^{n} \sigma_{x_0x_i} + \sum_{j=1}^{n} a_j\sigma_{x_ix_j} = 0 \tag{3.36}$$

In this case, a constraint is imposed on the a_i's to ensure that the grade estimation is unbiased. This means that on the average, the computed grade should be equal to the real grade and not systematically higher or lower. This constraint is written as

$$\sum_{i=1}^{n} a_i = 1 \tag{3.37}$$

It says simply, that the sum of the weighting factors should equal one. In this new problem of minimizing σ_e^2 in the light of a constraint, a special mathematical procedure involving

Lagrange multipliers is used. A treatment of this is beyond the scope of this book, and only the two resulting equations will be given:

$$\sum_{j=1}^{n} a_j \sigma_{x_i x_j} + \lambda = \sigma_{x_0 x_i}, \quad i = 1, \ldots, n \tag{3.38a}$$

$$\sum_{i=1}^{n} a_i = 1 \tag{3.38b}$$

where λ is the Lagrange multiplier. For n grades, there are $n+1$ unknowns $(a_1, \ldots a_n, \lambda)$. Equations (3.38) supply the needed $n+1$ equations. Once the a_i's have been found, the estimated grade is

$$g_0 = \sum_{i=1}^{n} a_i g_i \tag{3.39}$$

The estimated variance can then be found substituting the value of λ and a_i's into

$$\sigma_e^2 = \sigma_{x_0 x_0} - \sum_{i=1}^{n} a_i \sigma_{x_0 x_i} - \lambda \tag{3.40}$$

Thus this process provides what the inverse distance squared and other estimation procedures do not, a measure of the confidence associated with the assigned grade. It will be recalled that the actual grade would be expected to fall within the range of the average ± 1 standard deviation 68% of the time (and within ± 2 standard deviations 95% of the time) if the sample distribution is symmetric.

3.10.3 *Kriging example*

The concepts just described will be illustrated by an example. The grades corresponding to points $x_1, \ldots, x_6$ in Figure 3.75 are given in Table 3.28. For the sake of this example, only points x_1, x_2 and x_3 will be used. Hence it will be desired to find the equation

$$g_0 = a_1 g_1 + a_2 g_2 + a_3 g_3 \tag{3.41}$$

where a_1, a_2, a_3 are the weighting coefficients. A spherical variogram having the following values:

$$c_0 = 0.02 \quad c_0 + c_1 = 0.18 = \text{sill}$$

$$c_1 = 0.16$$

$$a = 450\,\text{ft}$$

has been found to describe the deposit (Hughes & Davey, 1979). This variogram is shown in Figure 3.77.

The required distances are first found (Table 3.29). Next the corresponding values of the variance γ are found using the general formulas

$$\gamma = \begin{cases} 0 & \text{if } h = 0 \\ c_1 \left[\dfrac{3}{2}\dfrac{h}{a} - \dfrac{1}{2}\dfrac{h^3}{a^3} \right] + c_0 & \text{if } 0 < h \leq a \\ c_1 + c_0 & \text{if } h > a \end{cases} \tag{3.42}$$

Table 3.28. Grade data for the kriging example. Hole designation after (Hughes & Davey, 1979).

Sample designation	Hole	Grade
x_1	C-37	$g_1 = 0.320$
x_2	C-25	$g_2 = 0.230$
x_3	C-26	$g_3 = 0.833$
x_4	C-27	$g_4 = 0.453$
x_5	C-15	$g_5 = 0.806$
x_6	C-36	$g_6 = 0.717$
x_0		$g_0 =$ (to be determined)

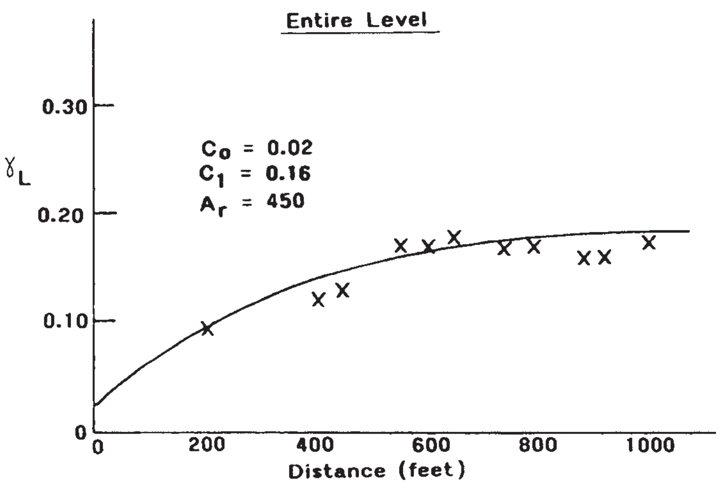


Figure 3.77. The experimental variogram (Hughes & Davey, 1979).

Table 3.29. Separation distance calculation.

From	To	Separation distance (ft)
x_0	x_1	$h_{01} = \sqrt{100^2 + 100^2} = 141$
x_0	x_2	$h_{02} = \sqrt{100^2 + 100^2} = 141$
x_0	x_3	$h_{03} = \sqrt{100^2 + 200^2} = 224$
x_1	x_2	$h_{12} = \sqrt{0^2 + 200^2} = 200$
x_1	x_3	$h_{13} = \sqrt{200^2 + 300^2} = 361$
x_2	x_3	$h_{23} = \sqrt{0^2 + 300^2} = 300$

Substituting the given values into the above equations yields

$$\gamma = \begin{cases} 0 & \text{if } h = 0 \\ 0.16\left[\dfrac{3}{2}\dfrac{h}{450} - \dfrac{1}{2}\left(\dfrac{h}{450}\right)^3\right] + 0.02 & \text{if } 0 \le h \le a \\ 0.18 & \text{if } h > a \end{cases}$$

Table 3.30. Variances for the example.

Lag	Distance (ft)	γ (% × %)
$h_{01} = h_{10}$	141	0.0927
$h_{02} = h_{20}$	141	0.0927
$h_{03} = h_{30}$	224	0.1296
$h_{12} = h_{21}$	200	0.1196
$h_{13} = h_{31}$	361	0.1712
$h_{23} = h_{32}$	300	0.1563
$h_{00} = h_{11} = h_{22} = h_{33}$	0	0.0

Table 3.31. Covariances for the example.

Lag	Distance (ft)	σ (% × %)
$h_{10} = h_{01}$	141	$\sigma_{01} = 0.0873 = \sigma_{10}$
$h_{20} = h_{02}$	141	$\sigma_{02} = 0.0873 = \sigma_{20}$
$h_{30} = h_{03}$	224	$\sigma_{03} = 0.0504 = \sigma_{30}$
$h_{12} = h_{21}$	200	$\sigma_{12} = 0.0604 = \sigma_{21}$
$h_{13} = h_{31}$	361	$\sigma_{13} = 0.0088 = \sigma_{31}$
$h_{23} = h_{32}$	300	$\sigma_{23} = 0.0237 = \sigma_{32}$
$h_{00} = h_{11} = h_{22} = h_{33}$	0	$\sigma_{11} = \sigma_{22} = \sigma_{33} = 0.18 = \sigma_{00}$

For $h_{01} = 141$, one finds that

$$\gamma = 0.16\left[\frac{3}{2}\left(\frac{141}{450}\right) - \frac{1}{2}\left(\frac{141}{450}\right)^3\right] + 0.02 = 0.0927$$

The resulting values are summarized in Table 3.30.

In the analysis, the covariances σ defined by

$$\sigma = \begin{cases} c_1 + c_0 & \text{if } h = 0 \\ c_1 + c_0 - \gamma & \text{if } 0 < h \le a \\ 0 & \text{if } h > a \end{cases} \tag{3.43}$$

are used.

The covariance σ_{01} corresponding to a lag h_{01} of 141 ft ($\gamma_{01} = 0.0927$) is

$$\sigma_{01} = 0.16 + 0.02 - 0.0927 = 0.0873$$

The covariances are summarized in Table 3.31.

The basic kriging equations are:

$$\sum_{j=1}^{n} a_j \sigma_{x_i x_j} + \lambda = \sigma_{x_0 x_i}, \qquad i = 1, \ldots, n$$

$$\sum_{i=1}^{n} a_i = 1$$

For this example, they become

$$\sum_{j=1}^{3} a_j \sigma_{x_i x_j} + \lambda = \sigma_{x_0 x_i}, \qquad i = 1, 3$$

$$\sum_{i=1}^{n} a_i = 1$$

Expanding one finds that

$$\begin{aligned}
a_1\sigma_{11} + a_2\sigma_{12} + a_3\sigma_{13} + \lambda &= \sigma_{01} \\
a_1\sigma_{21} + a_2\sigma_{22} + a_3\sigma_{23} + \lambda &= \sigma_{02} \\
a_1\sigma_{31} + a_2\sigma_{32} + a_3\sigma_{33} + \lambda &= \sigma_{03} \\
a_1 + a_2 + a_3 &= 1
\end{aligned}$$

The values of the covariances are now substituted from Table 3.31 into the above equations

$$\begin{aligned}
0.18a_1 + 0.0604a_2 + 0.0088a_3 + \lambda &= 0.0873 \\
0.0604a_1 + 0.18a_2 + 0.0237a_3 + \lambda &= 0.0873 \\
0.0088a_1 + 0.0237a_2 + 0.18a_3 + \lambda &= 0.0504 \\
a_1 + a_2 + a_3 &= 1
\end{aligned}$$

One is now faced with solving 4 equations with 4 unknowns. The answers are:

$$\begin{aligned}
a_1 &= 0.390 \\
a_2 &= 0.359 \\
a_3 &= 0.251 \\
\lambda &= -0.00677
\end{aligned}$$

The estimated grade is

$$g_0 = 0.390 \times 0.320 + 0.359 \times 0.230 + 0.251 \times 0.833 = 0.416\%\,\text{Cu}$$

The estimation variance is given by

$$\sigma_e^2 = \sigma_{x_0x_0} - \sum_{i=1}^{n} a_i\sigma_{x_0x_i} - \lambda$$

Since

$$\begin{aligned}
\sigma_{x_0x_0} &= c_0 + c_1 = 0.18 \\
\lambda &= -0.00677 \\
a_1\sigma_{01} + a_2\sigma_{02} + a_3\sigma_{03} &= 0.390 \times 0.0873 + 0.359 \times 0.0873 + 0.251 \times 0.0504 \\
&= 0.07804
\end{aligned}$$

Then

$$\sigma_e^2 = 0.18 - 0.07804 + 0.00677 = 0.10873$$

The standard deviation (SD) is the square root of the estimation variance or

$$\text{SD} = \sqrt{0.10873} = 0.330\%\,\text{Cu}$$

The interested reader is asked to estimate the grade at x_0 using all 6 surrounding points. The answers are:

$$a_1 = 0.263$$

$$a_2 = 0.326$$

$$a_3 = 0.167$$

$$a_4 = -0.0355$$

$$a_5 = -0.0372$$

$$a_6 = 0.318$$

$$\lambda = -0.00023$$

$$g_{x_0} = 0.480\%$$

$$\mathrm{SD} = 0.307\%$$

3.10.4 *Example of estimation for a level*

A kriging evaluation of level 5140 was performed by Hughes & Davey (1979). The variograms along strike (N 45° W to S 45° E) and perpendicular to strike (S 45° W to N 45° E) are shown in (Fig. 3.78).

The rules used by Hughes & Davey in the interpolation are:

– 250 ft radius of influence
– along strike

$$c_0 = 0.015 \qquad a = 450\,\text{ft}$$

$$c_1 = 0.20$$

– perpendicular to strike

$$c_0 = 0.01 \quad a = 400\,\text{ft}$$

$$c_1 = 0.16$$

The results are shown in Figure 3.79.

3.10.5 *Block kriging*

In the preceding sections the discussion has been on *point* kriging. *Block* values can be estimated by considering the blocks to be made up of several points. The values at the points are calculated as before and by averaging them, a block value is obtained. Parker & Sandefur (1976) have however shown that only a small error results when representing the grade of an entire block by a point. Hence the added effort required in block kriging may or may not be warranted.

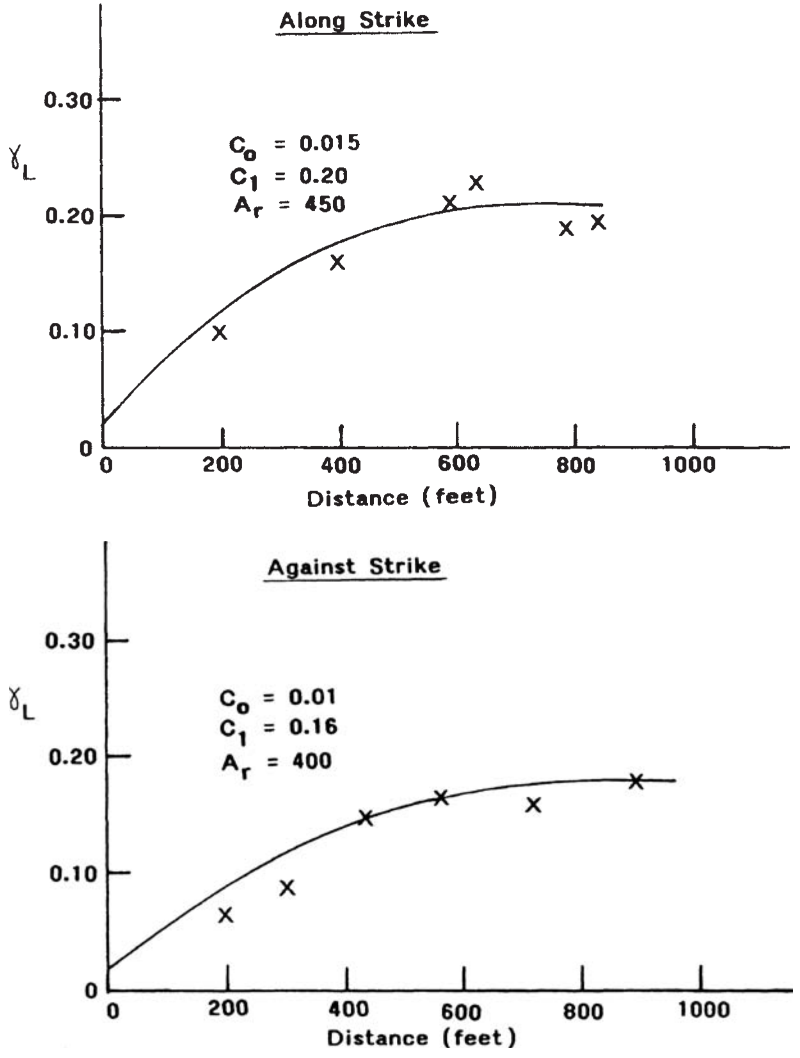


Figure 3.78. The experimental variograms for bench 5140 along strike and normal to strike (Hughes & Davey, 1979).

3.10.6 *Common problems associated with the use of the kriging technique*

Hughes & Davey (1979) have suggested the following problems are commonly associated with the use of the kriging technique.

1. Variograms do not accurately represent the mineralized zone because of inadequate data.
2. Mathematical models do not accurately fit the variogram data, or variograms have been improperly interpreted.
3. Kriging is insensitive to variogram coefficients.
4. Computational problems and expense are associated with repeatedly inverting large matrices.

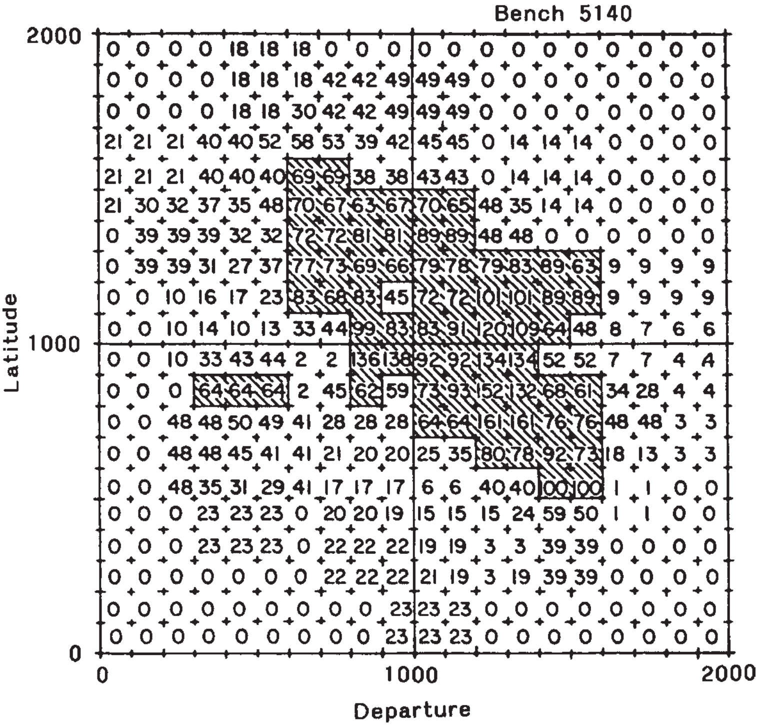


Figure 3.79. The kriged results for bench 5140 (Hughes & Davey, 1979).

Table 3.32. Comparison of the methods applied to the bench 5140 data (Hughes & Davey, 1979).

Method	Block size = 100 × 100 × 40 ft			
	Ore blocks	Tons ore	Avg ore grade (%)	Computer CPU sec.
Hand polygon		1,989,896	0.93	
Computer polygon	66	2,033,778	0.92	6.61
Inverse distance	65	2,002,963	0.91	7.08
Kriging	68	2,095,407	0.86	19.55

5. The matrix involved in finding the coefficients sometimes tends to be ill-conditioned. This means that for certain geometries, kriging doesn't work well.

6. There are problems associated with weighting coefficients.

3.10.7 *Comparison of results using several techniques*

Table 3.32 shows a comparison of some of the methods described in this chapter (Hughes & Davey, 1979). The decision as to which method to apply to any particular deposit evaluation is left to the user, and would certainly depend on the deposit, the data available, sample density, the type of results required, the required accuracy, and the amount of time, money, and energy that one is willing to expend on the evaluation of a specific deposit.

As indicated by Hughes & Davey (1979), there are pros and cons with each method. Many innovative designers have combined what they consider the best of various techniques to their particular application. The inverse distance weighting rules described by Hughes & Davey (1979) in Table 3.23 actually are a combination of inverse distance and geostatistical methods. Although the discussion herein has been restricted to two-dimensional examples, three-dimensional applications, especially with the aid of computers, are quite practical and generally result in a better interpolation of the deposit. There also is no question that the incorporation of geologic data (varying rock types, faults, etc.) into the grade assignment process is essential for generation of the most accurate model of the deposit.

REFERENCES AND BIBLIOGRAPHY

Adisoma, G.S., and Y.C. Kim. 1993. Jackknife kriging – A simple and robust new estimator. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(343–349). Montreal: CIMM.

Adisoma, G.S., and M.G. Hester. 1996. Grade estimation and its precision in mineral resources: the jackknife approach. Mining Engineering. 48(2): 84–88.

Ailleres, L., Finlay, J., Betts, P., Williams, H., and D. Milton. 2009. Three-dimensional combined gravity and magnetics inversion modelling as a guide to target haematitic iron ores – an example from the Koolanooka South (Western Australia) prospect. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 189–196.

Altman, K.A., Taylor, D.L., and R.W. Smith. 2000. How much testing should I do? Are the samples really representative? Pre-print Paper 00-71. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Axelson, A.H. 1963. A practical approach to computer utilization in mine planning. *Twenty-fourth annual mining symposium.* University of Minnesota: 151–165.

Bahaaddini, M., Sharrock, G., and B.K. Hebblewhite. 2011. Synthetic rock mass modelling of a rock mass with non-persistent joints. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 615–624.

Barnes, M.P. 1979a. Case study – ore-body modeling at Sacaton Mine, Arizona. In: *Computer Methods in the 80's* (A. Weiss, editor): 268–275. New York: SME-AIME.

Barnes, M.P. 1979b. Drill-hole interpolation: Estimating mineral inventory. In: *Open Pit Mine Planning and Design* (J.T. Crawford & W.A. Hustrulid, editors): 65–80. SME-AIME.

Barnes, M.P. 1980. *Computer-Assisted Mineral Appraisal and Feasibility.* New York: Society of Mining Engineers of the AIME.

Barnes, N.W. 1989. *The Limitations of Popular Techniques for Preproduction Reserve Estimation in Mining*. Salt Lake City, UT: M. Barnes and Associates, Inc.

Batterham, R.J., and R.A. Bearman. 2005. The role of science and technology in the future of the iron ore industry. Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 3–7.

Behn, R.G., Beniscelli, J., Carrasco, C.P., Cuadra, P., and G.A. Ferguson. 1998. Estimation of resources and engineering of reserves, Radomiro Tomic Project. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 241–264. London: IMM.

Berry, M., and P. McCarthy. 2006. Practical Consequences of Geological Uncertainty, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, NT, August 2006, pp. 253–258.

Best, J., and K. Gallant. 2004. Reconciliation wars – Who's data are right? Pre-print Paper 04-183. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Blackwell, G. 1998. Effective modeling of grades of mineral resources. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 265–280. London: IMM.

Blackwell, G.H., and A.J. Sinclair. 1993. QMIN, geostatistical software for mineral inventory using personal computers. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(350–357). Montreal: CIMM.

Blais, J. 1985. Personal Communication. Pueblo, Colorado: CF&I Steel Corporation.

Bodycoat, F.M. 2007. Stratigraphic and Structural Setting of Iron Mineralisation at E Deposit (East), Area C, Hamersley Province, Western Australia, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 51–56.

Boyle, C. 2007. Comparison of Multivariate Conditional Simulation Methods at the Yandicoogina Iron Ore Deposit, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 57–66.

Bye, A. 2006. The strategic and tactical value of a 3D geotechnical model for mining optimization, Anglo Platinum, Sandsloot open pit. JSAIMM. 106(2): 97–104.

Call, R.D. 1979. Development drilling. In: *Open Pit Mine Planning and Design* (J.T. Crawford & W.A. Hustrulid, editor): 29–40. New York: SME-AIME.

Carew, T.J. 2002. Unfolding – Getting the geometry right. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 103–112. Alaska: SME.

Carrasco, P.C. 2010. Nugget effect, artificial or natural? JSAIMM. 110(6): 299–305.

Case Histories and Methods in Mineral Resource Evaluation (A.E. Annels, editor). 1992. London: The Geological Society.

Cherrier, T.E. 1968. A report on the Ceresco Ridge extension of the Climax Molybdenite Deposit. M.Sc. Thesis. University of Minnesota.

Clark, A. 2011. Greenfields exploration and land access vital for pipeline of new projects. AusIMM Journal. Issue 5. Oct.

Clark, I. 1979a. Does geostatistics work? In: 16th *APCOM* (T.J. O'Neil, editor): 213–225. SME.

Clark, I. 1979b. *Practical Geostatistics.* London: Applied Science Publishers.

Clark, I. 1999. A case study in the application of geostatistics to lognormal and quasi-lognormal problems. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 407–416. Colorado School of Mines: CSM.

Clark, I. 2000. Erratic highs – A perennial problem in resource estimation. Pre-print Paper 00-110. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Clark, I. 2010. Statistics or geostatistics? Sampling error or nugget effect? JSAIMM. 110(6): 307–312.

Clark, I., and W. Harper. 2000. Practical geostatistics 2000. Ecosse North American Llc. Columbus, Ohio, USA, 342 pp.

Clark, J.W., Couzens, T.R., Dixon, W.C., and K.D. Wiley. 1984. Geological and geostatistical applications to mine production at Inspiration. Presented at the SME-AIME Fall Meeting Denver, Colorado Oct. 24–26, 1984. Preprint 84-381.

Clout, J.M.F. 2005. Iron Formation-Hosted Iron Ores in the Hamersley Province of Western Australia, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 9–19.

Colorado Department of Highways. Colorful Colorado Road Map.

Cortez, L.P., Durão, F.O., and A.J. Sousa. 1999. Mineral resources estimation methods: A comparative study. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 425–434. Colorado School of Mines: CSM.

Cortez, L.P., Sousa, A.J., and F.O. Durão. 1998. Mineral resources estimation using neural networks and geostatistical techniques. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 305–314. London: IMM.

Crawford, J.T. and Davey, R.K. 1979. Case study in open pit limit analysis. In: *Computer Methods for the 80's in the Mineral Industry* (A. Weiss, editor): 310–318. SME-AIME.

Crawford, D., Leevers, P., Parsons, S., Solly, G., and Nielsen. 2005. Grade control in the southern Middle-back Ranges: a case study. Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 271–280.

CRC Handbook of Chemistry and Physics, 1991–1992, 72nd Edition (D.R. Lide, editor-in-Chief): 4.1–4.34, 4.150–4.155. Boston: CRC Press.

Dagbert, M. 1982. Orebody modelling. Presented at the 1982 American Mining Congress International Mining Show, Las Vegas, NV, Oct. 11–14, 1982.

Dagbert, M., and M. David. 1977. Predicting vanishing tons before production starts or small blocks are no good for planning in porphyry type deposits. Presented at the 1977 AIME Annual Meeting, Atlanta, GA. Society of Mining Engineers of AIME: preprint number 77-AO-83.

Darragh, M., Clark, F., and B. Catto. 2009. Kicking the bucket – A case study of evolving domains and geological modeling, Sunrise Dam Gold mine, Western Australia. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 87–94.

Davey, R.K. 1979. Mineral block evaluation criteria. In: *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 83–96. NewYork: SME-AIME.

David, M. 1971. Geostatistical ore reserve calculations, a step by step case study. *Decision Making in the Mineral Industry,* CIM Special Volume 12: 185–191.

David, M. 1972. Grade-tonnage curve use and misuse in ore reserve estimation. *Trans. Inst. Min. Metall.* (Min. Sect. A): 129–132.

David, M. 1977. *Geostatistical Ore Reserve Estimation*: 283. Amsterdam: Elsevier.

David, M. 1988. *Handbook of Applied Advanced Geostatistical Ore Reserve Estimation*. Amsterdam: Elsevier.

David, M., and E. Toh. 1989. Grade control problems, dilution and geostatistics: Choosing the required quality and number of samples for grade control. *CIM Bulletin* (931) 82(11): 53–60.

Deraimse, J., and A.M. Thwaites. 1998. Use of non-linear geostatistical techniques for recoverable reserves estimation – a practical case study. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 315–326. London: IMM.

De Souza, L.E., Costa, J.F., and J.C. Koppe. 2002. Ageostatistical contribution to the use of resource/reserves classification systems. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 73–84. Alaska: SME.

Deutsch, C.V., and R.W. Lewis. 1992. Advances in the practical implementation of indicator geostatistics. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 169–180. Tucson: SME.

Deutsch, C.V., Magri, V., and Norrena. K. 1999. Optimal grade control using geostatistics and economics: Methodology and examples. Pre-print Paper 99-018. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

De-Vitry, C. 2005. Alternatives to Conditional Simulation for the Estimation of Iron Grade Distributions – A Case Study From BHP Billiton's Mt Whaleback Operations, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 183–192.

De-Vitry, C., Vann, J., and H. Arvidson. 2007. A Guide to Selecting the Optimal Method of Resource Estimation for Multivariate Iron Ore Deposits, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 67–77.

Dias, P.M., Costa, J.F.C.L., Koppe, J.C., and C. Diedrich. 2011. The effects of considering the spatial bulk density variability in mineral resources and reserves. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 81–88.

Dowd, P.A. 1998. Geostatistics in the United Kingdom – past, present and future. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 327–336. London: IMM.

Emery, X., and J.M. Ortiz. 2005. Estimation of mineral resources using grade domains: critical analysis and a suggested methodology. JSAIMM. 105(4): 247–256.

Emery, X., Ortiz, J.M., and A.M. Cáceres. 2008. Geostatistical modelling of rock type domains with spatially varying proportions: application to a porphyry copper deposit. JSAIMM. 108(5): 285–292.

Erarslan, K. 2001. Three dimensional variogram modeling and Kriging. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 51–56. Beijing: Balkema.

Francis, H.L., et al. 2000. Intelligent ore delineation: Integrating geosensing and geographical information systems. CIM Bulletin 93(1042): 96–99.

Fraser, S.J., Whitbourn, L., Yang, K., Ramanaidou, E., Connor, P., Poropat, G., Soole, P., Mason, P., Coward, D., and R. Phillips. 2006. Mineralogical Face-Mapping Using Hyperspectral Scanning for Mine Mapping and Control, AUSIMM Conference Proceedings, 6th International Mining Geology Conference, Darwin, August 2006, pp. 227–232

Frempong, P.K., and I. Clark. 1996. An indicator-based geostatistical assessment of the global vermiculite reserves at Palabora Mining Company. In Surface Mining 1996 (H.W. Glen, editor): 1–12. SAIMM.

Gertsch, L., Eloranta, J., and R. Gertsch. 2005. Ore characterization for process control: An update of a Mesabi Range project. Pre-print Paper 05-007. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.

González Jiménez, P., and S. Cossío Torres. 2008. Value Creation Through Sampling of a Gold Ore, AUSIMM Conference Proceedings, Sampling Conference Perth, WA, May 2008, pp. 69–71.

Green, C., and G. Dankett. 2011. Value creation through orebody knowledge innovations. AusIMM Journal. Issue 3. June.

Hanson, C., Thomas, D., and B. Gallagher. 2005. The Value of Early Geotechnical Assessment in Mine Planning. Coal 2005 Conference Brisbane, Proceedings, Australasian Institute of Mining and Metallurgy, pp. 17–30.

Hardwick, W.R. 1959. *Open-Pit Copper Mining Methods, Morenci Branch, Phelps Dodge Corp., Greenlee County, Ariz.* U.S. Bureau of Mines Information Circular 7911.

Hardwick, W.R. and E.L. Jones. III. 1959. *Open-Pit Copper Mining Methods and Costs at the Bagdad Mine, Bagdad Copper Corp., Yavapai County, Ariz.* USBM IC 7929.

Hardwick, W.R. and M.M. Stover. 1960. *Open-Pit Copper Mining Methods and Practices, Copper Cities Division, Miami Copper Co., Gila County, Ariz.* Bureau of Mines IC 7985.

Hatch, A.L. 1966. Factors affecting ore grade values. *Mining Engineering* 18(1): 72–75.

Hazen, S.W., Jr. 1967. *Assigning an Area of Influence for an Assay Obtained in Mine Sampling.* U.S. Bureau of Mines RI 6955.

Hekmat, A., Osanloo, M., and P. Moarefvand. 2011. Investigating the effect of different block dimensions on the economic life of open pit mines. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 287–295.

Henley, S., and D.F. Watson. 1998. Possible alternatives to geostatistics. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 337–354. London: IMM.

Hewlett, R.F. 1961. Application of computers to open pit ore reserve calculation. *Short Course on Computers and Computer Applications in the Mineral Industry, volume I.* University of Arizona.

Hewlett, R.F. 1962. *Computing Ore Reserves by the Polygonal Method Using a Medium-Size Digital Computer* RI 5952: 31 pp.

Hewlett, R.F. 1963. *Computing Ore Reserves by the Triangular Method Using a Medium-Size Digital Computer* RI 6176: 26 pp.

Hewlett, R.F. 1963. Computer methods in evaluation development, and operations of an ore deposit. Annual Meeting of the AIME, Dallas, Texas, Feb 24–28, 1963.

Hewlett, R.F. 1965. *Design of drill-hole grid spacings for evaluating low-grade copper deposits.* USBM R.I. 6634.

Hewlett, R.F. 1970. *Comparison of the Triangular, Polygonal, and a Statistical Method of Computing Grade and Tonnage of Ore for the Silver Bell Oxide Porphyry Copper Deposit* U.S. Bureau of Mines RI 7331: 33 pp.

Hodkiewicz, P., Guibal, D., and H. Arvidson. 2005. Structural Controls on Iron Mineralisation and Resource Domains in C Deposit, Area C, Hamersley Province, Australia Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 79–83.

Holmes, R.J. 2010. Sampling mineral commodities – the good, the bad, and the ugly. JSAIMM. 110(6): 269–276.

Hughes, W.E., and R.K. Davey. 1979. Drill hole interpolation: Mineralized interpolation techniques. In: *Open Pit Mine Planning and Design* (J. Crawford and W. Hustrulid, editors): 51–64. New York: Society of Mining Engineers of the AIME.

Hulse, D.E. 1992. The consequences of block size decisions in ore body modeling. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 225–232. Tucson: SME.

Hulse, D.E., and F. Davies. 1993. Using indicator kriging to analyze the impact of mining dilution. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(381–385). Montreal: CIMM.

Inoue, T., and K. Okaya. 1993. Modeling of ore grade variation and its application. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(386–393). Montreal: CIMM.

Jackson, C.F., and J.H. Hedges. 1939. Metal mining practice. *USBM Bulletin* 419: 58–79.

Jackson, C.F., and J.B. Knaebel. 1932. Sampling and estimation of ore deposits. *USBM Bulletin 356*: 114–149.

Jalkanen, G., and S. Vitton. 2009. Parameter estimation of the inverse distance estimation algorithm. Paper 09-116. SME Annual Meeting. Feb 22–25. Denver, CO.

Jalkanen, G., and S. Vitton. 2010. Dealing with trend in inverse distance estimation. Paper 10-045. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.

Jalkanen, G., Greuer, R., and H.S. Chen. 2002. Screening Kriging. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 95–102. Alaska: SME.

Johnson, K.R. 1995. Geological modeling – The way ahead (Concept modelling for exploration and mine planning). 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 1–4. Brisbane: AusIMM.

KCC (Scientific and Engineering Computer Center) 1967. *Isometric View of Bingham Pit* (5). Kennecott Copper Company.

Kenworthy, S., Noormohamed, K., Stuart, H., and P. Hodkiewicz. 2009. Mapping and three-dimensional modeling of structural controls in the Chirano gold deposits, Ghana – Keys to better near-mine exploration targeting. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 101–106.

Knudsen, H.P., Kim, Y.C., and E. Mueller. 1978. Comparative study of the geostatistical ore reserve estimation method over the conventional methods. *Mining Engineering* 30(1): 54–58.

Krige, D.G. 1984. Geostatistics and the definition of uncertainty. *Trans. Inst. Min. Metall (Sect A: Mining Ind.)* 93(4): A41–A47.

Krige, D.G. 1999. Conditional bias and uncertainty of estimation in geostatistics. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 3–14. Colorado School of Mines: CSM.

Krige, D.G., and P.G. Dunn. 1995. Some practical aspects of ore reserve estimation at Chuquicamata Copper Mine, Chile. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 125–134. Brisbane: AusIMM.

Krige, D.G., and W. Assibey-Bonsu. 2001. Valuation of recoverable resources by Kriging, direct condition or simulation. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 3–8. Beijing: Balkema.

Kuchta, M.E. 1986. Conceptual mine planning with interactive computer graphics. MSc Thesis T-3194. Colorado School of Mines.

Leevers, P., Gaughan, C., and G. Bubner. 2005. The Iron Magnet deposit. Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 85–97.

Lemieux, M. 2000. Parametric analysis in surface-mine reserve definition: the inherent error and its correction. Mining Engineering. 52(8): 57–64.

Lemieux, M. 2000. Surface mine reserve definition and the high grade fallacy. Mining Engineering. 52(2): 48–50.

Leonardson, R.W., Weakly, C.G., Lander, A.M., and P.B. Zohar. 2005. Exploring between drill holes yields new ounces at Goldstrike. Pre-print Paper 05-110. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.

Leuangthong, O., Hodson, T., Rolley, P., and Deutsch. C.V. 2006. Multivariate geostatistical simulation at Red Dog mine, Alaska, USA. CIM Magazine. May. pp. 115–124.

Leuangthong, O., Lyall, G., and C.V. Deutsch. 2002. Multivariate geostatistical simulation of a nickel laterite deposit. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 261–274. Alaska: SME.

Lilly, P.A. 2006. Geology and the Engineering of Safe and Efficient Mines, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 7–9.

Lopez, W.A. 1990. Cokriging: A multivariate geostatistical analysis to enhance ore reserve estimation. MS Thesis T-3577, Colorado School of Mines.

Martinez, L., Leiva, O., and J. Araya. 2011. What do geostatistics and cut-off grade optimisation have to do with real options applied to mine project evaluation? Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30.

McIntosh, C., and L.J. Reynolds. 2006. A New Geological Model for the Globe Progress Mine, New Zealand, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 51–55.

McWilliams, J.R. 1959. *Mining Methods and Costs at the Anaconda Company Berkeley Pit, Butte, Mont.* U.S. Bureau of Mines Information Circular 7888.

Merks, J.W. 1992. Geostatistics or voodoo statistics. *E&MJ* 193(9): 45–49.

Mohanta, H.K., and A.K. Sen. 1995. Computerized orebody modeling and mine design for the Chira Iron Ore deposit. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 21–32. Brisbane: AusIMM.

Murthy, P.S.N., and M. Makki. 1999. Possible alternatives to geostatistics – some views. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 435–444. Colorado School of Mines: CSM.

Nelson, M.G., and N.K. Riddle. 2003. Sampling and analysis for optimal ore-grade control. Pre-print Paper 03-128. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.

O'Donnell, N. 1950. Mine and surface maps. *Mining Engineering* 2(9): 944–946.

Ortiz, J.M., and X. Emery. 2006. Geostatistical estimation of mineral resources with soft geological boundaries: a comparative study. JSAIMM. 106(8): 577–584.

Page, P.I. 2005. Mining grade and sampling theory – the logical way to evaluate ore reserves. JSAIMM. 105(5): 311–316.

Parker, H.M. and R.L. Sandefur. 1976. A review of recent developments in geostatistics. AIME Annual Meeting, Las Vegas, NV.

Parker, H.M., Stearley, R.D., Swenson, A.J., Harrison, E.D., Maughan, J.R., Switzer, J.E., Mudge, K.L., and T.W. Smith. 2001. Evolution of computerized resource estimation at Bingham Canyon, Utah. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 25–30. Beijing: Balkema.

Parr, C.J. and N. Ely. 1973. Chapter 2. Mining law. *SME Mining Engineering Handbook, Volume 1* (A.B. Cummins and I.A. Given, editors): 2-2–2-4. SME-AIME.

Parrish, I.S. 1993. Tonnage factor – a matter of some gravity. Mining Engineering. 45(10):1268–1271.

Patterson, J.A. 1959. Estimating ore reserves follows logical steps. *E&MJ* 160(9): 111–115.

Pelley, C.W., and O.A. Satti. 1993. Geostatistics for risk analysis in orebody evaluation and ore related mine decisions. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(302–309). Montreal: CIMM.

Peters, W.C. 1978. *Exploration and Mining Geology*: 434. New York: John Wiley & Sons.

Pfleider, E.P. 1962. Mine valuation problem. University of Minnesota.

Phelps, E.R. 1968. Correlation of development data and preliminary evaluation. *Surface Mining* (E.P. Pfleider, editor): 122–137. New York: AIME.

Pilger, G.G., Costa, J.F., and J.C. Koppe. 2002. Optimizing the value of a sample. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 85–94. Alaska: SME.

Popoff, C.C. 1966. *Computing Reserves of Mineral Deposits: Principles and Conventional Methods*. Bureau of Mines IC 8283.

Ramsey, R.H. 1944. How Cananea develops newest porphyry copper. *E&MJ* 145(12): 74–83.

Ranta, D.E., A.C. Noble and M.W. Ganster. 1984. Geology and geostatistics in ore reserve estimation and mine evaluation. *Mine Feasibility – Concepts to Completion* (G.E. McKelvey, compiler). Northwest Mining Association.

Rattenbury, M., Lane, R., van Dam, M., McIntosh, C., Whetter, N., and M. McCulloch. 2009. Rapid three-dimensional modeling of mine geology – Structural applications to the Globe-Progress mine, West Coast, New Zealand. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 13–17.

Reich, H. 1961. Density of rocks. *Mining Engineers' Handbook, Third Edition* (R. Peele, editor) 1: 10-A-30. New York: Wiley.

Rendu, J.-M. 1980. A case study: Kriging for ore valuation and mine planning. *E&MJ* 181(1): 114–120.

Rendu, J.-M. 1984. Pocket geostatistics for field geologists. Presented at the SME-AIME Fall Meeting, Denver, Colorado, Oct. 24–26, 1984. Preprint 84-408.

Rendu, J.M. 1994. 1994 Jackling Lecture: Mining geostatistics – forty years passed. What lies ahead. Mining Engineering. 46(6): 557–558.

Rendu, J.M. 1998. Practical geostatistics at Newmont Gold: A story of adaptation. Mining Engineering. 50(2): 40–45.

Ripke, S. 2007. Mystery of missing grade and recovery. Paper 07-018. SME Annual Meeting. Feb 25–28. Denver, CO.

Robson, S.D., and M.J. Nugus. 2006. Multiple Geological Models, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 185–190.

Rossi, M. 2004. Comparing simulated and interpreted geologic models. Pre-print Paper 04-29. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Rossi, M.E. 1999. Improving the estimates of recoverable reserves. Mining Engineering. 51(1): 50–54.

Rossi, M.E., and J. Camacho Vidakovich. 1999. Using meaningful reconciliation information to evaluate predictive models. Pre-print Paper 99-020. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Roy, I., and B.C. Sarkar. 2002. Geostatistical orebody modeling and inventory of Gua Iron Ore Deposit, Jharkhand, India. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 243–250. Alaska: SME.

Samanta, B., Bandopadhyay, S., Ganguli, R., and S. Dutta. 2005. A comparative study of the performance of single neural network vs. Adaboost algorithm based combination of multiple neural networks for mineral resource estimation. JSAIMM. 105(4): 237–246.

Schneider, S., Murphy, R.J., Melkumyan, A., and E. Nettleton. 2011. Autonomous mapping of mine face geology using hyperspectral data. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24-30. pp. 865–876.

Schofield, D., and B. Denby. 1993. Genetic algorithms: A new approach to pit optimization. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(126–133). Montreal: CIMM.

Schurtz, R.F. 1999. A rigorous integral relation between kriging and inverse distance with certain of its implications. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 417–424. Colorado School of Mines: CSM.

Scofield, N.A. 2011. Geological domaining and resource estimation – A discussion. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 99–112.

Seigel, H.O., Gingerich, J.C., and E.O. Köstlin. 2002. Explore or acquire? The dilemma. CIM Bulletin 95(1058): 62–69.

Sikka, D.B., and R.B. Bhappu. 1994. Economic potential of the Malanjkhand Proterozoic porphyry copper deposit, M.P. India. Mining Engineering. 46(3): 221–229.

Stanley, B.T. 1979. Mineral model construction: Principles of ore-body modelling. *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 43–50. New York: SME-AIME.

Stanley, D. 1993. Geologic modeling at Newmont Gold Company. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(445–452). Montreal: CIMM.

Storkey, A., Doecke, A., and A. Whaanga. 2007. Stratigraphy of the Western Channel Iron Deposits of the BHP Billiton Iron Ore Marillana Creek Operations, Western Australia, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 187–195.

Tang, Y., and Y. Lan. 2001. A new calculation method of mineral reserves and examination of mineral reserves. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 35–38. Beijing: Balkema.

Thompson, I.S. 2002. A critique of valuation methods for exploration properties and undeveloped mineral resources. CIM Bulletin 95(1061): 57–62.

Thwaites, A.M. 1998. Assessment of geological uncertainty for a mining project. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 391–406. London: IMM.

USGS (United States Geological Survey). 1976. Jefferson County, Colorado – County Map Series (Topographic). Scale 1:50000.

Vallée, M. 2000. Mineral resources + engineering, economy and legal feasibility = ore reserves. CIM Bulletin 93(1033): 53–61.

Vorster, A.P., and W.L. Schöning. 1995. A revision of ore evaluation techniques at Sischen Iron Ore Mine, South Africa. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 139–148. Brisbane: AusIMM.

Vorster, A.P., and J. Smith. 1996. Developments in ore evaluation and grade control at Sishen Iron Ore mine. In Surface Mining 1996 (H.W. Glen, editor): 105–115. SAIMM.

Wade, E.J. 1967. Computerization of ore reserve calculations at Tasu. *Canadian Mining Journal* 88(3).

Waterman, G.C. and S. Hazen. 1968. Chapter 3.1. Development drilling and bulk sampling. In: *Surface Mining* (E.P. Pfleider, editor): 69. New York: AIME.

Weaton, G.F. 1972. Valuation of a mineral property. Personal Communication. Minneapolis, Minn.

Weaton, G.F. 1973. Valuation of a mineral property. *Mining Engineering* 25(5).

Westerfelt, W.Y. 1961. Weights of minerals and rocks. Section 25. Mine Examinations. In: *Mining Engineers Handbook – Third Edition* (R. Peele, editor) II: 25–21.

Whitworth, D. 2009. Geology, Grade Control and Reconciliation at the Douglas Mineral Sands Deposit. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 155–166.

Williams, W.R. 1983. *Mine Mapping & Layout*. New Jersey: Prentice-Hall.

Wu, X., and P.D. Williams. 2007. Indicator Kriging Model and Revenue-Based Mine Planning System Implemented at Phoenix Mine, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 159–165.

Yalcin, E., and A. Ünal. 1992. The effect of variogram estimation on pit-limit design. *CIM Bulletin* (961)85(6): 47–50.

Yamamoto, J.K. 1996. Ore reserve estimation: A new method of block calculation using the inverse of weighted distance. 197(9): 69–72.

Zhu, D., Qiu, G., and J. Pan, 2009. Present situation and future trends in the Chinese steel industry and utilisation of lowgrade iron ore resources. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 35–42.

REVIEW QUESTIONS AND EXERCISES

1. Briefly discuss the steps in the development of a geologic data base.
2. What is meant by core 'logging'?
3. In a surface mine feasibility study, what role do maps serve?
4. What is meant by the 'scale' of a map?
5. In the English system of units, what is meant by a scale of 1:50?
6. In the metric system, what is meant by a scale of 1:1000?
7. Is a scale of 1:50 larger or smaller than a scale of 1:20? Explain your answer.
8. What different types of maps might be involved in a feasibility study? What type(s) of information might be included/shown on each?
9. In the U.S., what process might be followed to determine the ownership status of a certain parcel of land?
10. What are the three common map types used for mine planning and design? What would be shown on each?
11. In plan, in which quadrant is the map normally positioned?
12. For sections, which is the normal viewing direction chosen?
13. What are some of the different possibilities for the choice of the north direction?
14. What are some of the possible choices for the coordinate system?
15. Practical guidelines for the preparation of maps are provided in Table 3.1. What is meant by 'scale: graphic and numerical'? Why is this important?
16. What does the nomenclature DDH stand for?
17. Mining operations are commonly divided into four stages. What are they? Today, what is an important fifth stage?
18. What types of geologic information should be developed in the exploration stage?
19. What are the two basic types of diamond core drilling? Describe each.
20. What types of information might be collected during the logging of core?
21. Summarize the characteristics of each of the different exploratory techniques.
22. Why is it important that the core and the information gathered there from be treated with utmost respect?
23. In Table 3.4, assay values are given for both core and sludge samples. What are sludge samples? How do the values compare? What is a possible reason for the differences?
24. Where is the ore located based upon the drill hole log from the Comstock mine? What is the mineral? What is the nature of the overburden?
25. What is meant by oriented core? What is the purpose?
26. What is meant by the RQD? How is it calculated? What information does it provide?
27. What is the purpose(s) of compositing?
28. What is the difference between a simple average and a weighted average?

29. What is the difference between an ore zone composite and a bench composite?
30. What are some of the benefits of compositing?
31. Using the drill-hole log given in Table 3.8 and assuming a bench height of 18 ft, what would be the composite grades assigned to the first three benches?
32. How does the choice of bench height affect the compositing?
33. How does the choice of bench elevation affect the compositing?
34. What is the purpose of a tonnage factor?
35. Assuming that an ore has a specific gravity of 2.9, what would be the tonnage factor expressed in the English and metric systems?
36. Assuming that the ore density is 3.1 g/cm3, what is the tonnage factor in the English system?
37. Assuming that the ore density is 190 lbs/ft3, what is the tonnage factor in the metric system?
38. What techniques are available to determine the density of the material to be mined? Why is this determination important?
39. In the example given in section 3.4.2, assume that the mining company has a contract to sell 7000 tons of metal X per year. The mined material contains 1.5% of the contained metal and the processing plant recovery is 75%. How large a plan area must be exposed per year? Assume the other factors in the example are the same.
40. Begin with Figure 3.22 and complete the steps required to arrive at the final section shown in Figure 3.27.
41. Repeat Problem 40 but first draw Figure 3.22 in AutoCad. Then use the technique to complete the steps. Compare the areas with those shown in Figure 3.28.
42. What geologic features are evident on the Mesabi iron range of Minnesota? Why is it helpful to know this?
43. In Figure 3.30 the pit slope angles are indicated to be 27°. On the drawing, however, they appear to be at 45°. What is the reason?
44. Describe the process by which the 2D sections are combined to form the 3D pit.
45. In Table 3.13 the ore area and the average grade for a section have been determined. Repeat the process assuming that Hole 6 is missing.
46. How is the overall pit ore tonnage and grade determined?
47. How is the pit end tonnage determined?
48. Discuss the method of vertical sections based upon the use of grade contours.
49. In section 3.6 equivalent influence distances have been chosen for sections 4 and 20 to incorporate end volumes. Show how the numbers were chosen.
50. Discuss the process used to obtain the ore tonnage, the average in-place ore grade, and the average grade of the mined ore in the example given in section 3.6.
51. What is the significance of the 0.12% grade used in the example?
52. In Table 3.18 an average stripping ratio has been determined. What definition of stripping ratio has been used here?
53. Today, horizontal rather than vertical sections are commonly used when describing ore bodies. What has been the primary reason for the change?
54. Determine the average grade for the triangular solid defined by holes C-51, C-28 and C-47 in Figure 3.45? If the thickness of the slice is 40 ft, what is the volume of the solid? If the density is 2.6 g/cm^3, how many tons are involved?
55. Repeat problem 54 assuming that the ore intercept is 40 ft for hole C-51, 20 ft for hole C-28, and 30 ft for hole C-47.

56. In section 3.7.3 an example involving a polygon formed around hole C-41 has been presented. Apply the process to hole C-35.
57. In Figure 3.49 the process involving the construction of a polygon at the boundary of a hole array is demonstrated. Apply the technique to hole C-51. Assume R = 250 ft.
58. Summarize the steps in developing hand-generated polygons.
59. What are some special conditions where the radius of sample influence does not apply?
60. Today, typically block models are used to represent ore bodies. Summarize the important guidelines regarding block size.
61. What 'special case' situation will be used in this section to assign grades to the blocks?
62. What is meant by the rule of nearest points? Assign grades in the region defined by 500N–1000N and 500E–1000E using this rule. Compare your results to what is shown in Figure 3.54.
63. Describe the inverse distance weighting technique.
64. Assign a grade to the block having center coordinates (650E, 850N) in Figure 3.57 using the inverse distance weighting technique.
65. Describe the inverse squared weighting technique. Assign a grade to the block in problem 64 using this technique.
66. How might you determine the most appropriate distance weighting power to be used?
67. What is meant by the angle of exclusion?
68. How do you select the radius of influence?
69. What is meant by the term 'isotropic'? What is meant by the term 'anisotropic'? How would this property be included?
70. Carefully read the rules provided in Table 3.23. How does one take into account the presence of different rock types?
71. Using the grades for bench 5140 as given in Table 3.21, check the given average grade value.
72. Plot a histogram of the values using an interval of 0.2% Cu. Superimpose the mean value.
73. From viewing the histogram developed in problem 72, what type of distribution is shown?
74. Discuss the procedure used to calculate the % cumulative frequency.
75. What is the finesse provided by standard probability plots? Locate this paper on the Web and download an example. Plot the data provided in Table 3.24. Compare your result with that given in Figure 3.62.
76. Repeat the process described in the book to determine the additive constant ß. Develop the plot shown in Figure 3.64.
77. What is meant by the following terms:
    - range
    - variance
    - standard deviation
78. Determine the variance for the data set given in Table 3.24.
79. Describe the logic for determining the range of sample influence using geostatistical techniques.
80. What is meant by:
    - the lag
    - the geostatistical variance
    - the semi-variance
    - the semi-variogram

81. In section 3.9.3 an example is provided of the calculation of $\gamma(600)$ using N-S pairs. Repeat the procedure for $\gamma(800)$ using N-S pairs.
82. In figure 3.68 all of the pairs have been used independent of direction. To be able to do this, what must be true?
83. What is meant by
    – the sill
    – the nugget effect
    – the range
84. How does the 'sill' relate to the usual statistical values as determined in problem 77?
85. What is the reason for the occurrence of the 'nugget effect'?
86. What is meant by
    – the chaotic variance
    – the structured variance
    – the total variance
87. What might an experimental variogram look like for
    – a strataform deposit
    – a porphyry copper deposit
    – a placer gold deposit
88. Why do you need a mathematical expression to represent the experimental variogram?
89. What are the characteristics of the spherical model?
90. What is the equation for the spherical model? What parameters must be extracted from the experimental variogram?
91. What type of information concerning the deposit is provided by a variogram?
92. Discuss the basic idea behind the use of kriging.
93. What is meant by covariance?
94. An example of the application of kriging has been presented in section 3.10.3. Follow through the example. Check the answers provided for a_1, a_2, a_3 and γ. Determine the estimated grade value.
95. Assign a grade to point x_0 using the inverse distance technique and the same holes as were used in problem 93.
96. Repeat the kriging example in section 3.10.3 but now use all six of the surrounding holes.
97. Assign the grade to point x_0 using the inverse distance technique and the same holes as in problem 95.
98. Figure 3.78 shows the variograms determined along strike and against strike. What is the conclusion?
99. What is meant by block kriging?
100. What are some of the common problems associated with the use of the kriging technique?
101. What are the advantages of kriging? How is the estimation variance used?

CHAPTER 4

Geometrical considerations

4.1 INTRODUCTION

The ore deposits being mined by open pit techniques today vary considerably in size, shape, orientation and depth below the surface. The initial surface topographies can vary from mountain tops to valley floors. In spite of this, there are a number of geometry based design and planning considerations fundamental to them all. These are the focus of this chapter. By way of introduction consider Figure 4.1 which is a diagrammatic representation of a volume at the earth's surface prior to and after the development of an open pit mine.

The orebody is mined from the top down in a series of horizontal layers of uniform thickness called benches. Mining starts with the top bench and after a sufficient floor area has been exposed, mining of the next layer can begin. The process continues until the bottom bench elevation is reached and the final pit outline achieved. To access the different benches a road or ramp must be created. The width and steepness of this ramp depends upon the type of equipment to be accommodated. Stable slopes must be created and maintained during the creation and operation of the pit. Slope angle is an important geometric parameter which has significant economic impact. Open pit mining is very highly mechanized. Each piece of mining machinery has an associated geometry both related to its own physical size, but also with the space it requires to operate efficiently. There is a complementary set of drilling, loading and hauling equipment which requires a certain amount of working space. This space requirement is taken into account when dimensioning the so-called working benches. From both operating and economic viewpoints certain volumes must or should, at least, be removed before others. These volumes have a certain minimum size and an optimum size.

It is not possible in this short chapter to try and fully cover all of the different geometrical aspects involved in open pit mine planning and design. However, the general principles associated with the primary design components will be presented and whenever possible illustrated by examples.

4.2 BASIC BENCH GEOMETRY

The basic extraction component in an open pit mine is the bench. Bench nomenclature is shown in Figure 4.2.

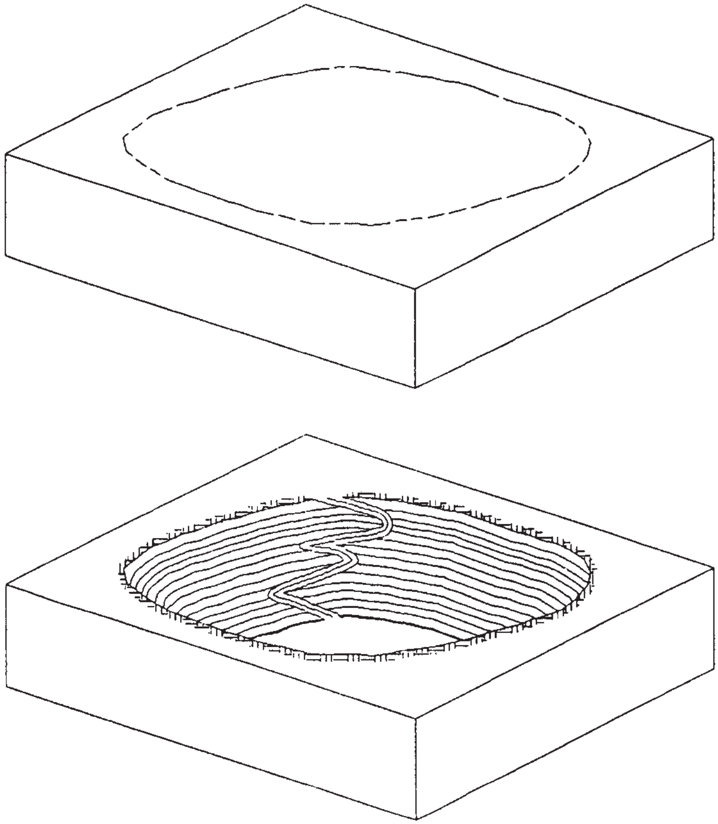


Figure 4.1. Geometry change in pit creation.

Each bench has an upper and lower surface separated by a distance H equal to the bench height. The exposed subvertical surfaces are called the bench faces. They are described by the toe, the crest and the face angle α (the average angle the face makes with the horizontal). The bench face angle can vary considerably with rock characteristics, face orientation and blasting practices. In most hard rock pits it varies from about 55° to 80°. A typical initial design value might be 65°. This should be used with care since the bench face angle can have a major effect on the overall slope angle.

Normally bench faces are mined as steeply as possible. However, due to a variety of causes there is a certain amount of back break. This is defined as the distance the actual bench crest is back of the designed crest. A cumulative frequency distribution plot of measured average bench face angles is shown in Figure 4.3.

The exposed bench lower surface is called the bench floor. The bench width is the distance between the crest and the toe measured along the upper surface. The bank width is the horizontal projection of the bench face.

There are several types of benches. A working bench is one that is in the process of being mined. The width being extracted from the working bench is called the cut. The width of the working bench W_B is defined as the distance from the crest of the bench floor to the new

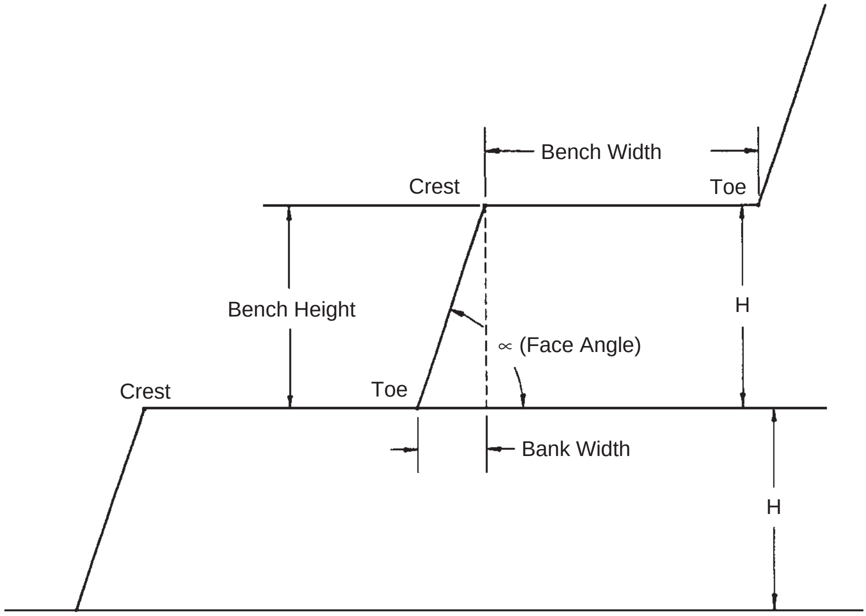


Figure 4.2. Parts of a bench.

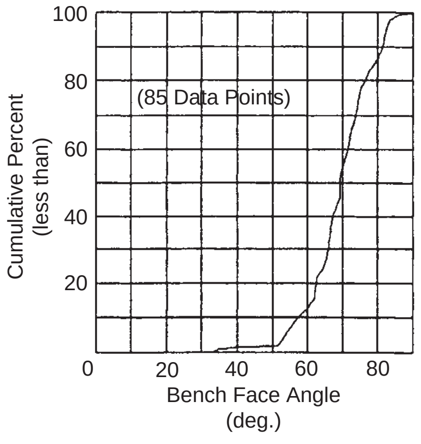


Figure 4.3. Cumulative frequency distribution of measured bench face angles (Call, 1986).

toe position after the cut has been extracted (see Fig. 4.4). A detailed calculation of cut and working bench dimensions is found in Subsection 4.4.5. After the cut has been removed, a safety bench or catch bench of width S_B remains.

The purpose of these benches is to:

(a) collect the material which slides down from benches above,
(b) stop the downward progress of boulders.

During primary extraction, a safety bench is generally left on every level. The width varies with the bench height. Generally the width of the safety bench is of the order of $^2/_3$ of the

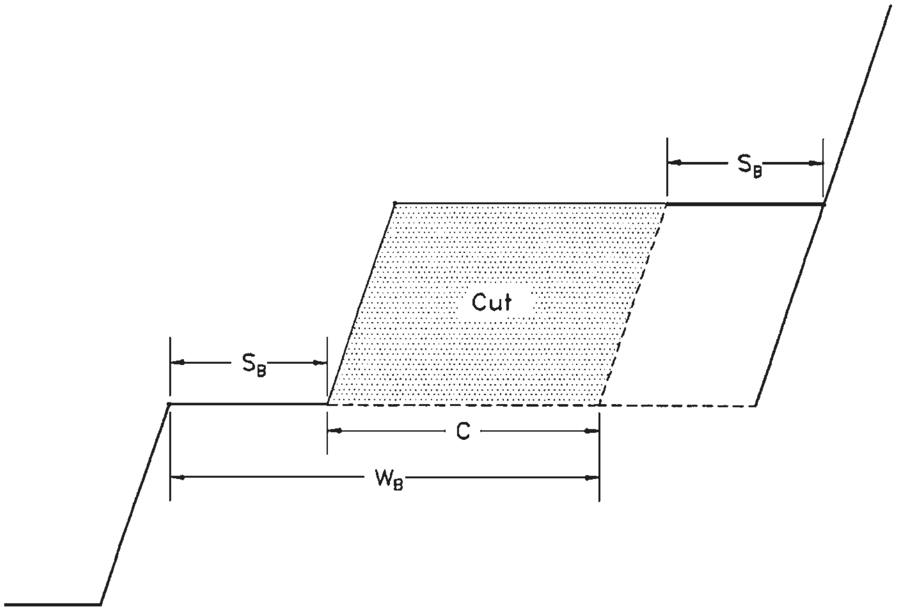


Figure 4.4. Section through a working bench.

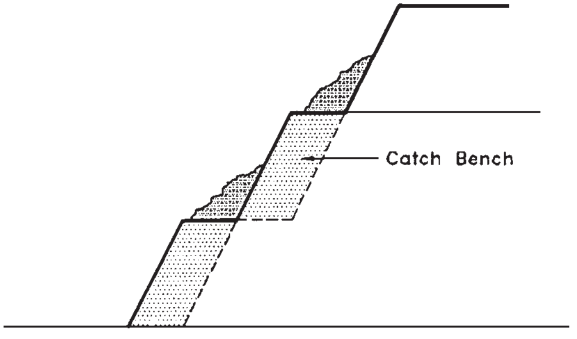


Figure 4.5. Functioning of catch benches.

bench height. At the end of mine life, the safety benches are sometimes reduced in width to about $\frac{1}{3}$ of the bench height.

Sometimes double benches are left along the final pit wall (Fig. 4.6). These are benches of double height which consequently permit, at a given overall slope angle, a single catch bench of double width (and hence greater catching capability). Along the final pit contour careful blasting is done to maintain the rock mass strength characteristics.

In addition to leaving the safety benches, berms (piles) of broken materials are often constructed along the crest. These serve the function of forming a 'ditch' between the berm and the toe of the slope to catch falling rocks. Based upon studies of rock falls made by Ritchie (1963), Call (1986) has made the design catch bench geometry recommendations given in Table 4.1 and illustrated in Figure 4.7.

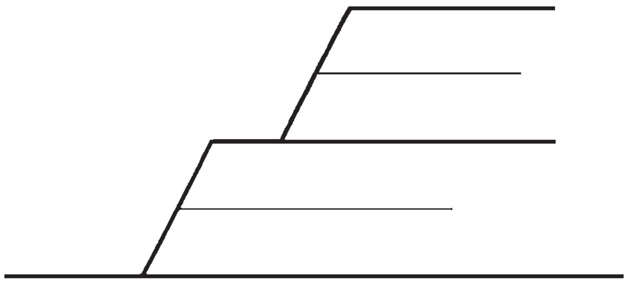


Figure 4.6. Double benches at final pit limits.

Table 4.1. Typical catch bench design dimensions (Call, 1986).

Bench height (m)	Impact zone (m)	Berm height (m)	Berm width (m)	Minimum bench width (m)
15	3.5	1.5	4	7.5
30	4.5	2	5.5	10
45	5	3	8	13

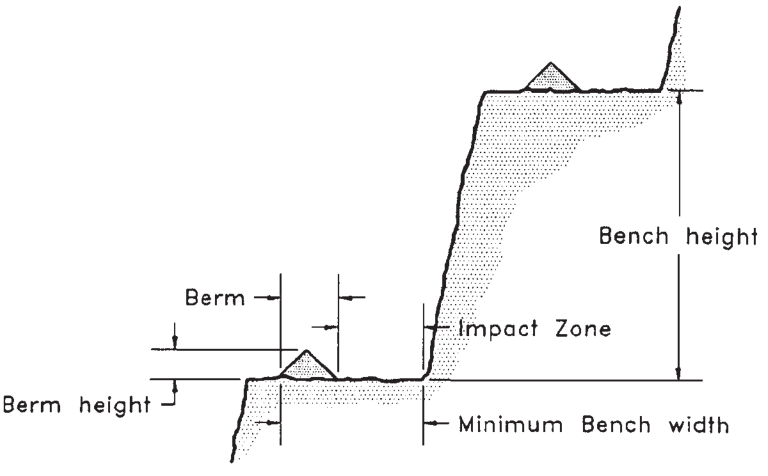


Figure 4.7. Catch bench geometry (Call, 1986).

A safety berm is also left (Fig. 4.8) along the outer edge of a bench to prevent trucks and other machines from backing over. It serves much the same function as a guard rail on bridges and elevated highways. Normally the pile has a height greater than or equal to the tire radius. The berm slope is taken to be about 35° (the angle of repose).

In some large open pits today median berms are also created in the center of haulage roads. In this book the word 'berm' is used to refer to the piles of rock materials used to improve mine safety. Others have used the word 'berm' as being synonymous with bench.

In the extraction of a cut, the drills operate on the upper bench surface. The loaders and trucks work off of the bench floor level.

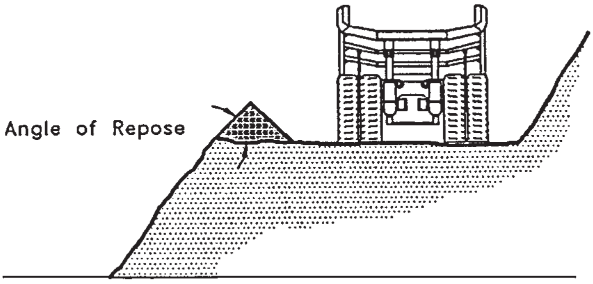


Figure 4.8. Safety berms at bench edge.

A number of different factors influence the selection of bench dimensions. Bench height becomes the basic decision since once this is fixed the rest of the dimensions follow directly. A common bench height in today's large open pits is 50 ft (15 m). For smaller pits the value might be 40 ft (12 m). For small gold deposits a typical value could be 25 ft (7.5 m). A general guideline is that the bench height should be matched to the loading equipment. When using shovels, the bench height should be well within the maximum digging height. For the 9 yd capacity shovel shown in Figure 4.9, it is seen that the maximum cutting height is 43′6″. Hence it could be used with 40 ft benches. A general rule of thumb is that the bench height should not be greater than that of the sheave wheel. Operating in benches with heights greater than this sometimes result in overhangs which endanger the loading and other operations.

Figure 4.10 shows typical reach heights for shovels and front end loaders as a function of bucket size.

At one time, bench heights were limited by drilling depth. Modern drills have largely removed such restrictions. However, in large open pit mines, at least, it is desirable to drill the holes in one pass. This means that the drill must have a mast height sufficient to accommodate the bench height plus the required subdrill.

A deposit of thickness T can be extracted in many ways. Two possibilities are shown in Figure 4.11:

(a) 3 benches of height 50 ft,

(b) 6 benches of height 25 ft.

Higher and wider benches yield:

– less selectivity (mixing of high and low grade and ores of different types);

– more dilution (mixing of waste and ore);

– fewer working places hence less flexibility;

– flatter working slopes; large machines require significant working space to operate efficiently.

On the other hand, such benches provide:

– fewer equipment setups, thus a lower proportion of fixed set up time;

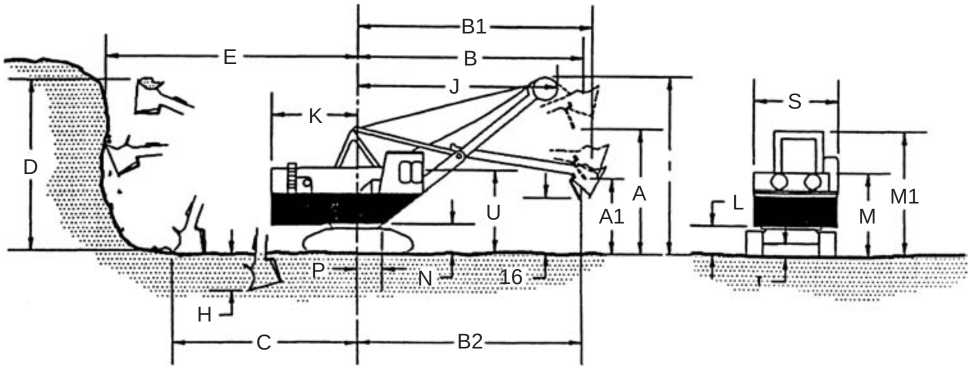
– improved supervision possibilities;

– higher mining momentum; larger blasts mean that more material can be handled at a given time;

– efficiencies and high productivities associated with larger machines.

The steps which are followed when considering bench geometry are:

(1) Deposit characteristics (total tonnage, grade distribution, value, etc.) dictate a certain geometrical approach and production strategy.



Shovel Working Range

Dipper Capacity (Nominal) cu.yds	9
Dipper Capacities (Range) cu.yds	6 ½–6
Length of Boom	41'–6"
Effective length of dipper handle	25'–6"
Overall length of dipper handle	30'–9"

These dimensions will vary slightly depending upon dipper selection.

	Angle of boom	45°	
A	Dumping height – maximum	28'–0"	A
A1	Dumping height at maximum radius – B1	20'–6"	A1
B	Dumping radius at maximum height – A	45'–6"	B
B1	Dumping radius – maximum	47'–6"	B1
B2	Dumping radius at 16'0" dumping height	47–0"	B2
D	Cutting height – maximum	43'–6"	D
E	Cutting radius – maximum	54'–6"	E
G	Radius of level floor	35'–3"	G
H	Digging depth below ground level – maximum	8'-6"	H
I	Clearance height – boom point sheaves	42'–3"	I
J	Clearance radius – boom point sheaves	40'–0"	J
K	Clearance radius – revolving frame	19'–9"	K
L	Clearance Under frame – to ground	6'–2"	L
M	Clearance height top of house	18'–10"	M
M1	Height of A-frame	31'–2"	M1
N	Height of boom foot above ground level	9'–11"	N
P	Distance – boom foot to center of rotation	7'–9"	P
S	Overall width of machinery house & operating cab	22'–6"	S
T	Clearance under lowest point in truck frame	14"	T
U	Operator's eye level	18'–0"	U

Figure 4.9. Diagrammatic representation of a 9 yd^3 shovel (Riese, 1993).

(2) The production strategy yields daily ore-waste production rates, selective mining and blending requirements, numbers of working places.

(3) The production requirements lead to a certain equipment set (fleet type and size).

(4) Each equipment set has a certain optimum associated geometry.

(5) Each piece of equipment in the set has an associated operating geometry.

(6) A range of suitable bench geometries results.

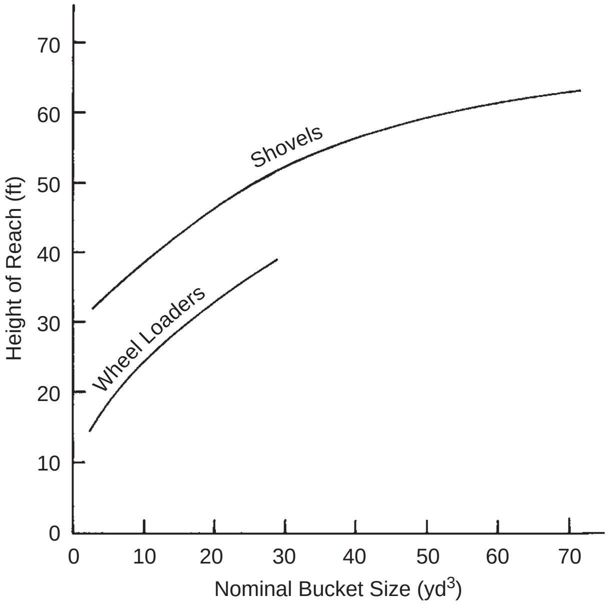


Figure 4.10. Height of reach as a function of bucket size.

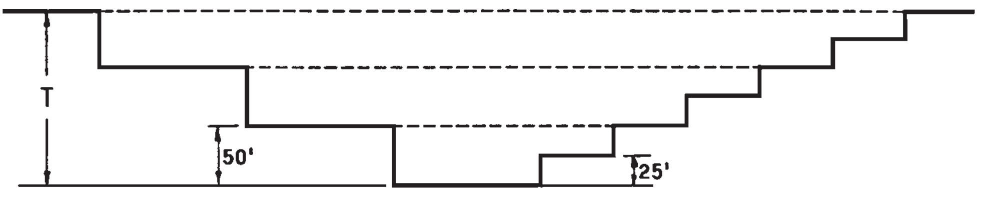


Figure 4.11. Two different bench height scenarios.

(7) Consequences regarding stripping ratios, operating vs. capital costs, slope stability aspects, etc. are evaluated.

(8) The 'best' of the various alternatives is selected.

In the past when rail bound equipment was being extensively used, great attention was paid to bench geometry. Today highly mobile rubber tired/ crawler mounted equipment has reduced the detailed evaluation requirements somewhat.

4.3 ORE ACCESS

One of the topics which is little written about in the mining literature is gaining initial physical access to the orebody. How does one actually begin the process of mining? Obviously the approach depends on the topography of the surrounding ground. To introduce the topic it will be assumed that the ground surface is flat. The overlying vegetation has been removed

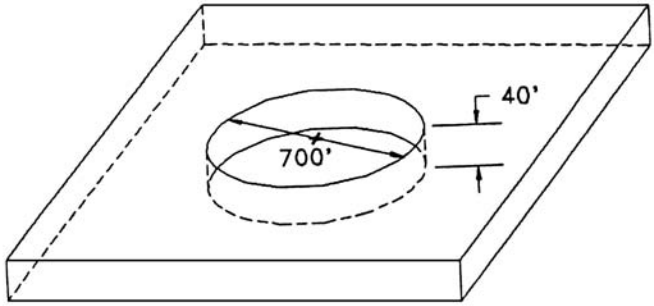


Figure 4.12. Example orebody geometry.

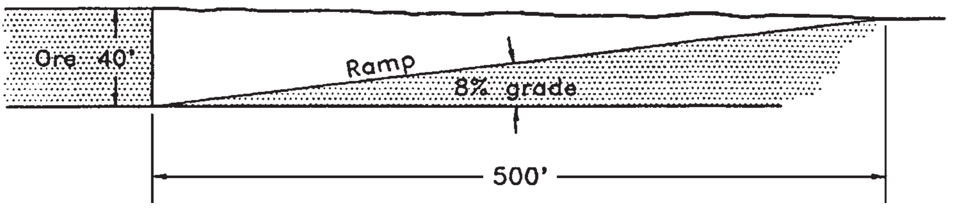


Figure 4.13. Ramp access for the example orebody.

as has the soil/sand/gravel overburden. In this case it will be assumed that the orebody is 700 ft in diameter, 40 ft thick, flat dipping and is exposed by removing the soil overburden. The ore is hard so that drilling and blasting is required. The bench mining situation is shown in Figure 4.12.

A vertical digging face must be established in the orebody before major production can begin. Furthermore a ramp must be created to allow truck and loader access. A drop cut is used to create the vertical breaking face and the ramp access at the same time. Because vertical blastholes are being fired without a vertical free face, the blast conditions are highly constrained. Rock movement is primarily vertically upwards with only very limited sideways motion. To create satisfactory digging conditions the blastholes are normally rather closely spaced. Here only the geometry aspects will be emphasized. To access the orebody, the ramp shown in Figure 4.13 will be driven. It has an 8% grade and a width of 65 ft. Although not generally the case, the walls will be assumed vertical. To reach the 40 ft desired depth the ramp in horizontal projection will be 500 ft in length. There is no general agreement on how the drop cut should be drilled and blasted. Some companies drill the entire cut with holes of the same length. The early part of the ramp then overlies blasted rock while the final portion is at grade. In the design shown in Figure 4.14 the drop cut has been split into three portions. Each is blasted and loaded out before the succeeding one is shot. Rotary drilled holes $9\frac{7}{8}''$ in diameter are used. The minimum hole depth is 15 ft. This is maintained over the first 90 ft of the ramp. The hole depth is then maintained at 7 ft below the desired final cut bottom. A staggered pattern of holes is used.

The minimum width of the notch is controlled largely by the dimensions of the loading machine being used. In this example, it will be assumed that the loading machine is the 9 yd^3 capacity shovel shown diagrammatically in Figure 4.9.

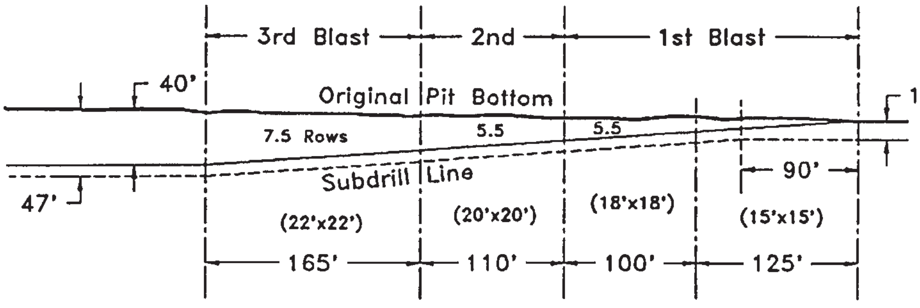


Figure 4.14. Blast design for the ramp excavation.

In the generally tight confines of the drop cut the following shovel dimensions:
K, the clearance radius of the revolving frame,
J, the clearance radius of the boom point sheaves,
G, the maximum digging radius of the level floor, and
E, the maximum cutting radius
are of importance. As can be seen from Figure 4.9, these are:

$$K = 19'9''$$

$$J = 40'0''$$

$$G = 35'3''$$

$$E = 54'6''$$

The minimum width of the drop cut is given by

$$\text{Minimum width} = K + J$$

In this case it is

$$\text{Minimum width} = 19'9'' + 40'0'' = 59'9''$$

This is such that both the front and rear portions of the machine can clear the banks on the two sides as it revolves in the digging and dumping modes.

The maximum digging radius of the level floor is used to indicate the maximum drop cut width for the shovel working along one cutting path. The maximum value is that which the shovel dipper (bucket) can be moved horizontally outward, thereby accomplishing floor cleanup.

The maximum width of the cut at floor level would be

$$\text{Maximum cut width (floor)} = 2 \times 35'3'' = 70'6''$$

The maximum width of the cut at crest level would be

$$\text{Maximum cut width (crest)} = 2 \times 54'6'' = 109'$$

In practice the cutting width for the shovel moving along one path is relatively tightly constrained by the shovel dimensions. In this case:

$$\text{Minimum cut width (crest)} \cong 60\ \text{ft}$$

$$\text{Maximum cut width (floor)} \cong 71\ \text{ft}$$

$$\text{Maximum cut width (crest)} = 109\ \text{ft}$$

For typical cut slope angles of 60 to 80°, the maximum cut width (floor) is the controlling dimension. When the cutting path is down the center of the cut and the shovel is digging to both sides the maximum floor and minimum crest radii would be

$$\text{Maximum floor radius} = 35'3''$$

$$\text{Minimum crest radius} = 40'0''$$

In any case, for laying out the blasting round and evaluating minimum pit bottom dimensions one wants to exceed the minimum working space requirements.

Figures 4.15A through 4.15D show the minimum floor bottom geometry when the shovel moves along the two cutting paths. The loading would first be from one bank. The shovel would then move over and load from the other. This would be considered very tight operating conditions and would be used to create a final cut at the pit bottom.

The usual drop cut is shown in Figures 4.16A through 4.16C where the shovel moves along the cut centerline and can dig to both sides. It will be noted that the shovel must swing through large angles in order to reach the truck.

In both cases the working bench geometry at this stage is characterized by cramped operating conditions.

Two locations for the drop cut/ramp will be considered. The first (case A) is entirely in the waste surrounding the pit. It is desired to have the floor of the ramp at the bottom of ore just as it reaches the ore-waste contact. This is shown diagrammatically in Figure 4.17. The volume of waste rock mined in excavating the ramp is

$$\text{Ramp volume} = \frac{1}{2}H\frac{100H}{g}R_W$$

where R_W is the average ramp width, H is the bench height, and g is the road grade (%). In this case it becomes

$$\text{Ramp volume} = \frac{1}{2}\frac{(40)^2 \times 100 \times 65}{8} = 650,000\ \text{ft}^3$$

This waste must be excavated and paid for before any ore can be removed. However in this arrangement all of the ore can be removed. If it is assumed that the orebody can be extracted with vertical walls, then the ore volume extracted is

$$\text{Ore volume} = \frac{\pi D^2 H}{4} = \frac{\pi}{4}(700)^2 \times 40 = 15,400,000\ \text{ft}^3$$

Upon entering the orebody mining proceeds on an ever expanding front (Fig. 4.18).

As the front expands the number of loading machines which can effectively operate at the same time increases. Hence the production capacity for the level varies with time.

In summary for this ramp placement (case A):

Waste removed (road) = 650,000 ft^3
Ore extracted = 15,400,000 ft^3
% ore extracted = 100%

Another possibility (case B) as is shown in Figure 4.19 is to place the ramp in ore rather than to place the ramp in waste rock. This would be driven as a drop cut in the same way as discussed earlier. The volume excavated is obviously the same as before but now it is ore. Since the material is ore it can be processed and thereby profits are realized earlier.

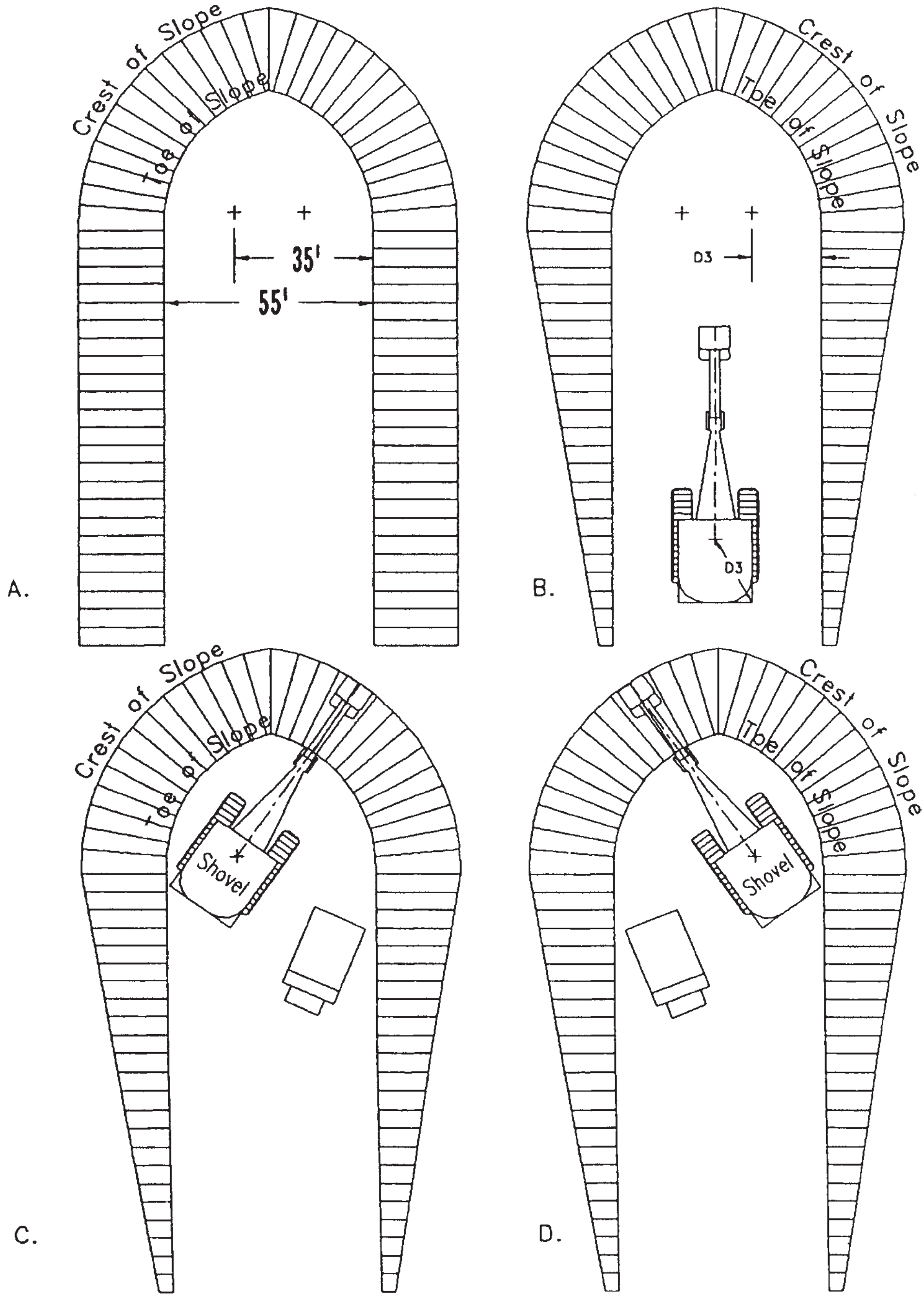


Figure 4.15. Minimum width drop cut geometry with shovel alternating from side to side.

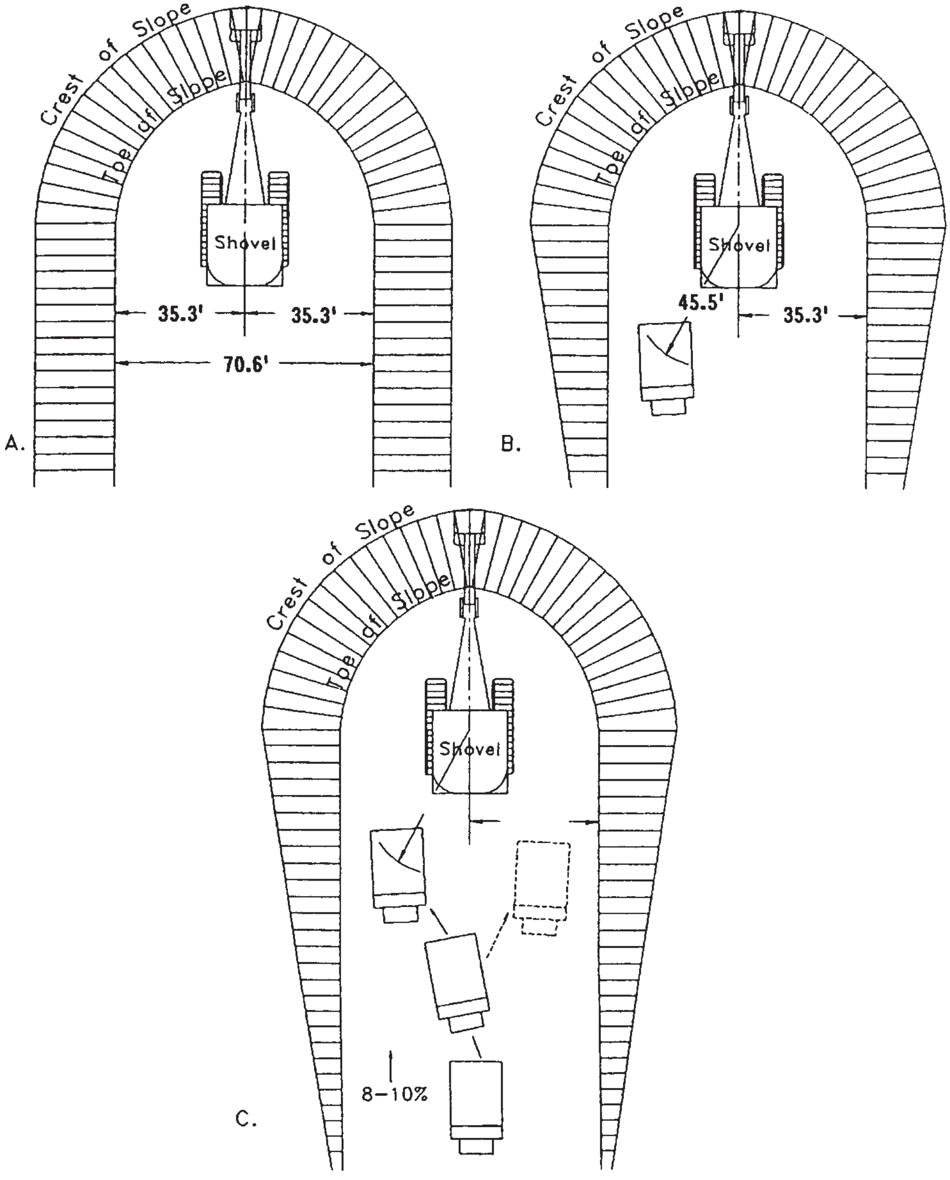


Figure 4.16. Minimum width drop cut geometry with shovel moving along centerline.

From the ramp bottom, the extraction front is gradually increased in length (Fig. 4.20). Obviously the disadvantage is that when mining is completed a quantity of ore remains locked up in the ramp. This quantity is equal to the amount of waste extracted in case A.

Thus the two important points to be made are:

– If the haul road is added external to the planned pit boundaries, then an additional quantity of material equal to the volume of the road must be extracted.

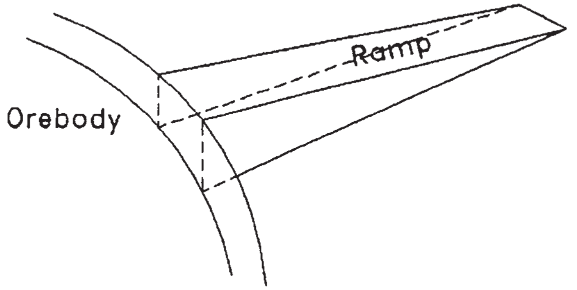


Figure 4.17. Isometric view of the ramp in waste approaching the orebody.

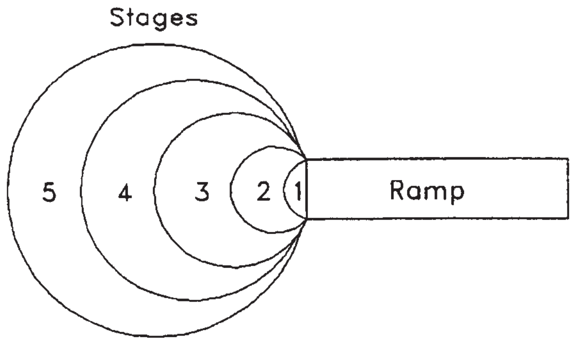
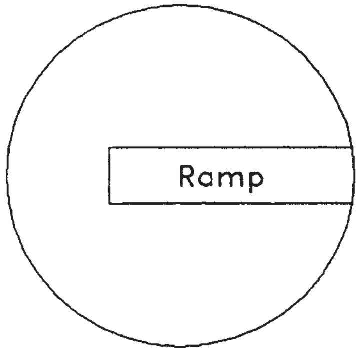


Figure 4.18. Diagrammatic representation of the expanding mining front.



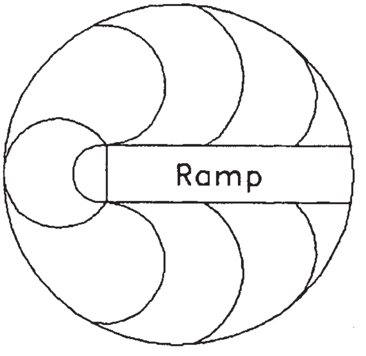


Figure 4.19. Dropcut/ramp placement in ore.

Figure 4.20. Expansion of the mining front.

– If the haul road is added internal to the original planned boundaries, then a quantity of material equal to the road volume must be left in place.

Rather than a straight road such as shown in case A, one might have considered a curved road such as shown in plan in Figure 4.21. With the exception of the final portion, the road is entirely driven in waste. The road could be placed so that the 'ore' left is in the poorest grade.

Assume that the pit is not 1 bench high but instead consists of 2 benches such as is shown in Figure 4.22. The idea is obviously to drive the ramp down to the ore level and establish

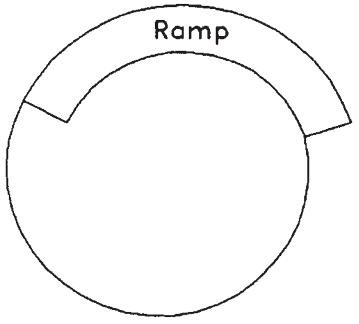


Figure 4.21. Ramp starting in waste and ending in ore.

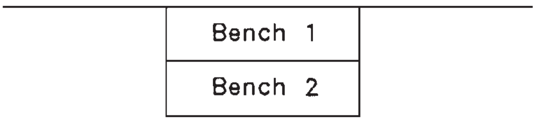


Figure 4.22. Section through a two bench mine.

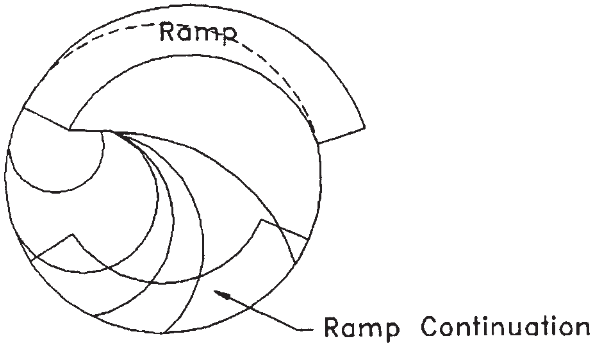


Figure 4.23. Two ramp sections with pit expansion.

the desired production rate. Then while mining is underway on level 1, the ramp would be extended in ore to the lower level as shown in Figure 4.23 through the use of a drop cut. All of the ore lying below the ramp is obviously sterilized. For a multi-bench operation, the procedure continues as shown in Figure 4.24. Note that a flat section having a length of 200 ft has been left in this example between the decline segments. The ramp has a corkscrew shape and the coils get tighter and tighter as the pit is deepened. Rather soon in this example, the pit would reach a final depth simply because the ramp absorbed all of the available working space.

A vertical section taken through the final pit with the orebody superimposed is shown in Figure 4.25. For this particular design where only the initial segment of the ramp is in waste, a large portion of the orebody is sterilized. The amount of waste removed is minimized, however.

An alternative design is one where the ramp is underlain by waste and all of the ore is removed. To make this construction one starts the road design at the lowest bench and works back out. This exercise is left to the reader.

The actual design will generally be somewhere in between these two alternatives with the upper part of the ramp underlain by waste and the lower part by 'ore'.

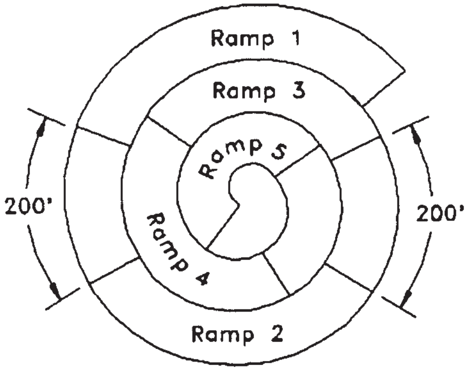


Figure 4.24. Plan view showing ramp locations for a five bench mine.

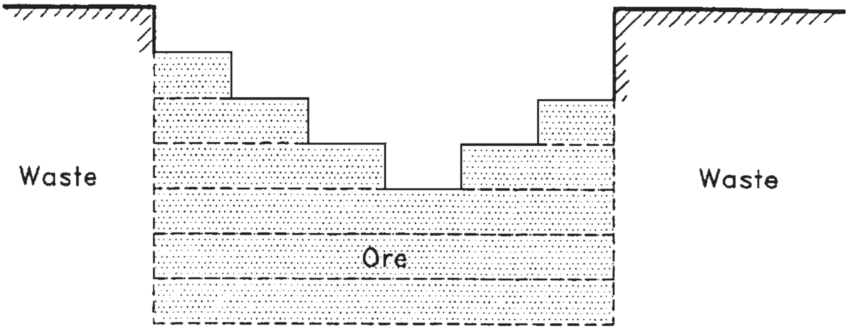


Figure 4.25. Section view showing the sterilization of reserves by ramp.

The excavation may start with attacking the ore first so that the cash flow is improved. Later during the mine life, the waste will be stripped as the main access is gradually moved outward.

In summary:

– there can be considerable volumes associated with the main ramp system;

– the location of the ramp changes with time;

– in the upper levels of the pit, the ramp is underlain by waste; in the lower ranges it is underlain by mineral;

– cash flow considerations are significantly affected by ramp timing;

– the stripping ratio, the percent extraction and the overall extraction are strongly affected simply by the haul road geometry (road width and road grade).

Drop cuts are used on every level to create a new bench. Figures 4.26A through 4.26D show the steps going from the current pit bottom through the mining out of the level. Often the ramp is extended directly off of the current ramp and close to the existing pit wall. This is shown in Figure 4.27. A two level loading operation is shown isometrically in Figure 4.28. The ramp access to both levels in this relatively simple example is easily seen.

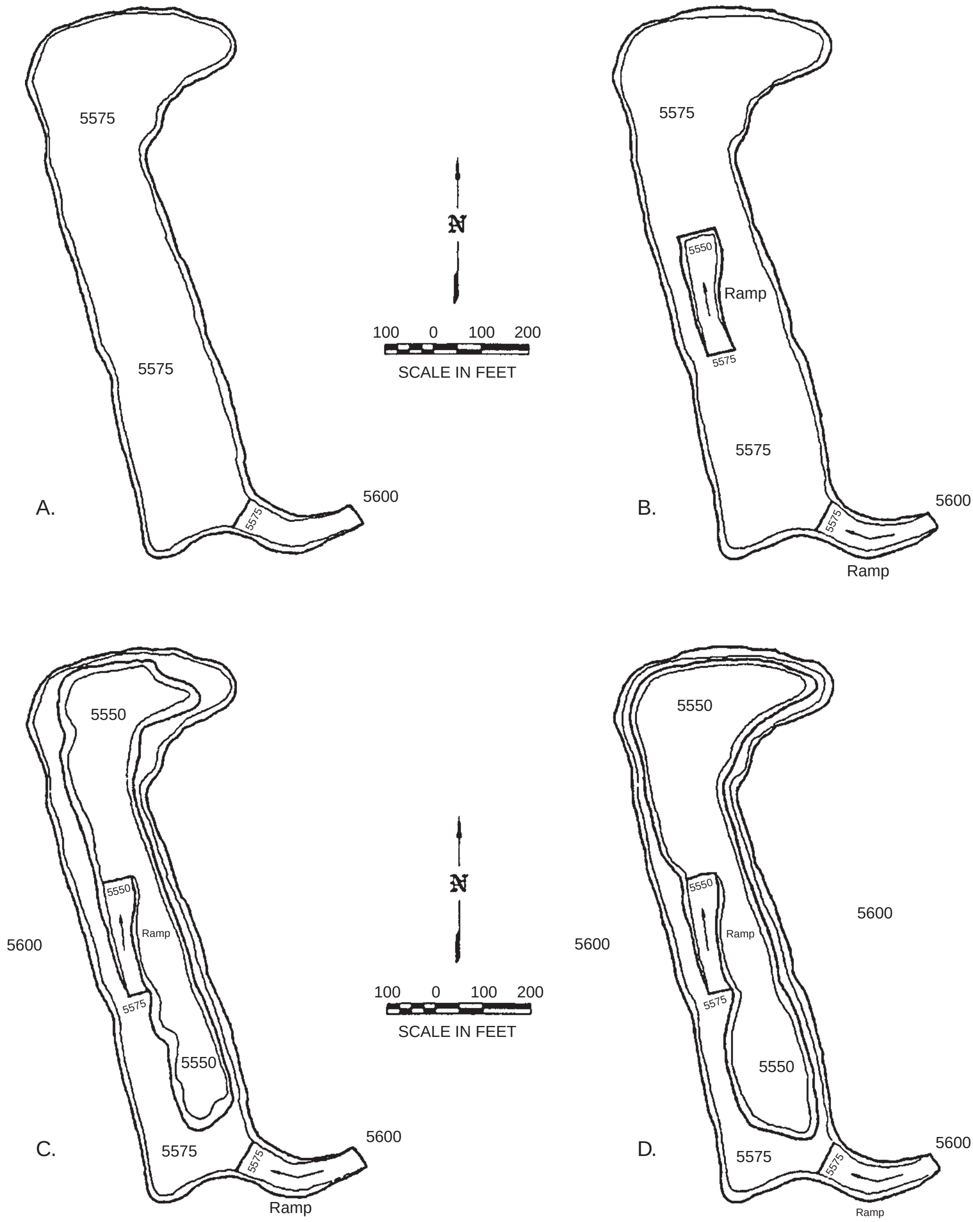


Figure 4.26. Plan view of an actual pit bottom showing drop cut and mining expansion (McWilliams, 1959).

There are many examples where the orebody lies in very rugged terrain. Figure 4.29 shows diagrammatically one possible case. Here the entry to the orebody is made by pushing back the hillside. Bench elevations are first established as shown on the figure. In this case the bench height is 50 ft. Initial benches are established by making pioneering cuts along the surface at convenient bench elevations.

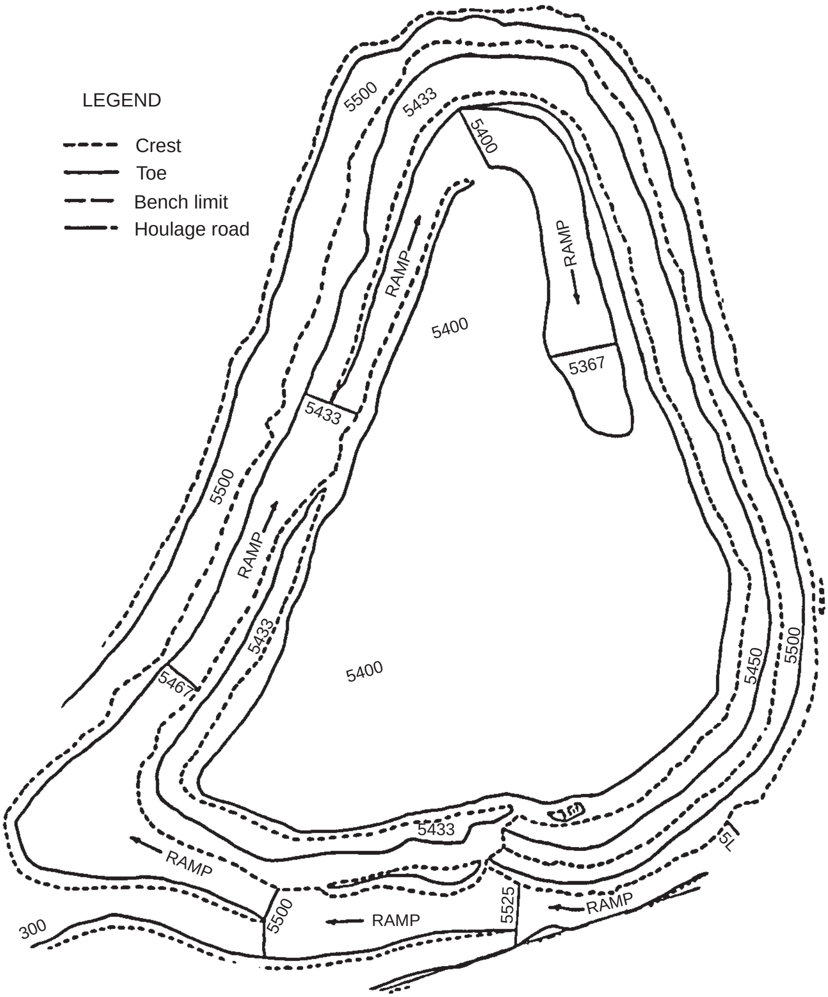


Figure 4.27. Extension of the current ramp close to the pit wall (McWilliams, 1959).

If the slope is composed of softer material, then a dozer can notch it without further assistance (Fig. 4.30). For harder rock types, ripping prior to dozing may be enough. However if the rock is hard or the slope is steep, drilling and blasting will probably be necessary for the pioneer cut. Generally air track types of drills are used. They can reach and drill in very difficult places and can tow their own air compressors/generators.

As shown in Figure 4.31 a shovel can be used instead of a dozer for notching a slope. The notch is enlarged by taking successive cuts until the full bench height is achieved.

Once these initial benches are established, mining of the full faces with vertical blast holes proceeds. Obviously the upper benches have to be advanced before the lower ones.

The final pit outline for this section is shown in Figure 4.33. The reader is encouraged to consider the pit development sequence and the point where drop cuts would be used.

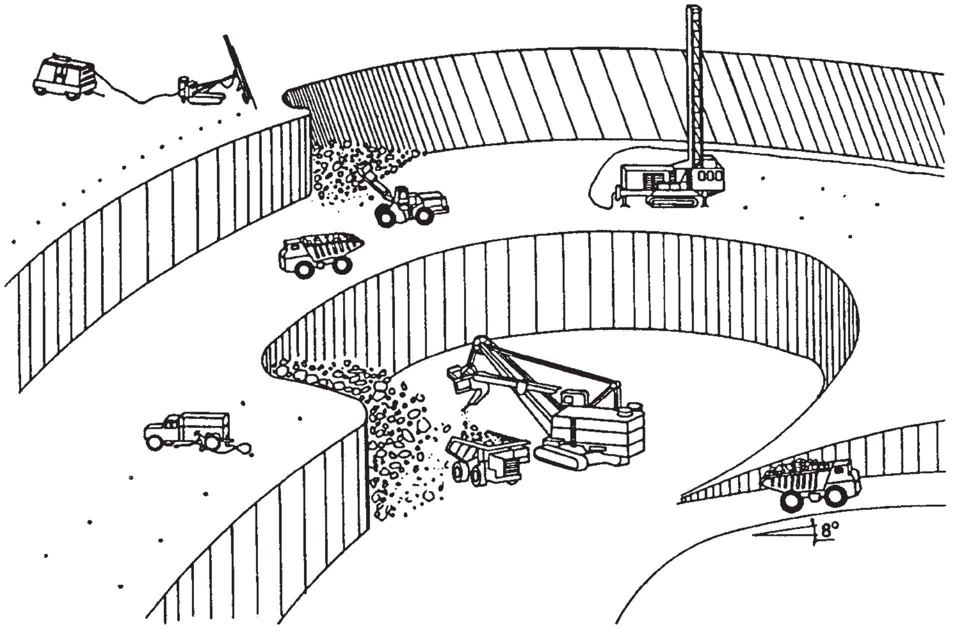


Figure 4.28. Isometric view of simultaneous mining on several levels (Tamrock, 1978).

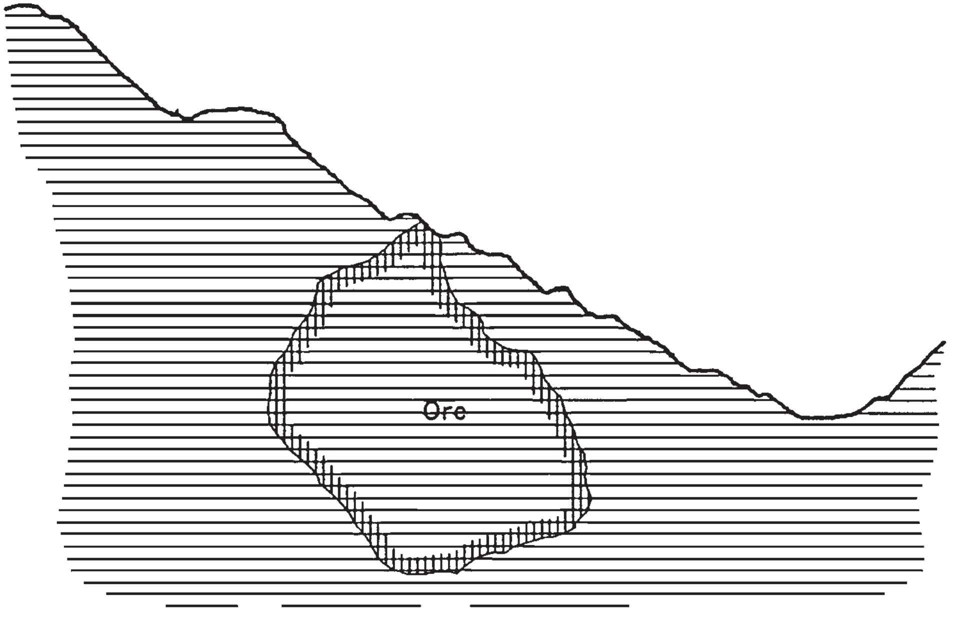


Figure 4.29. Deposit located in mountainous terrain.

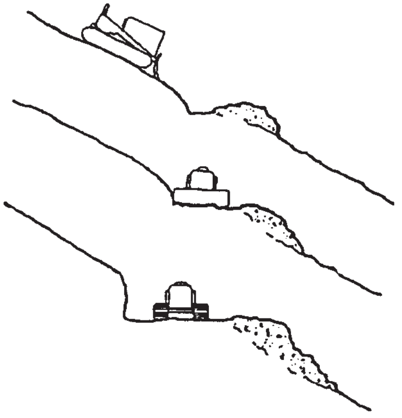


Figure 4.30. Creating initial access/benches (Nichols, 1956).

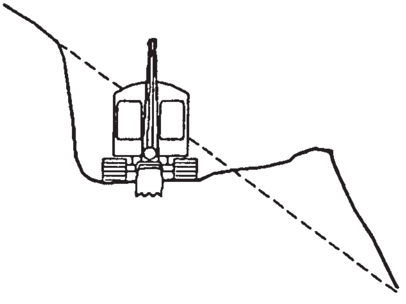


Figure 4.31. Sidehill cut with a shovel (Nichols, 1956).

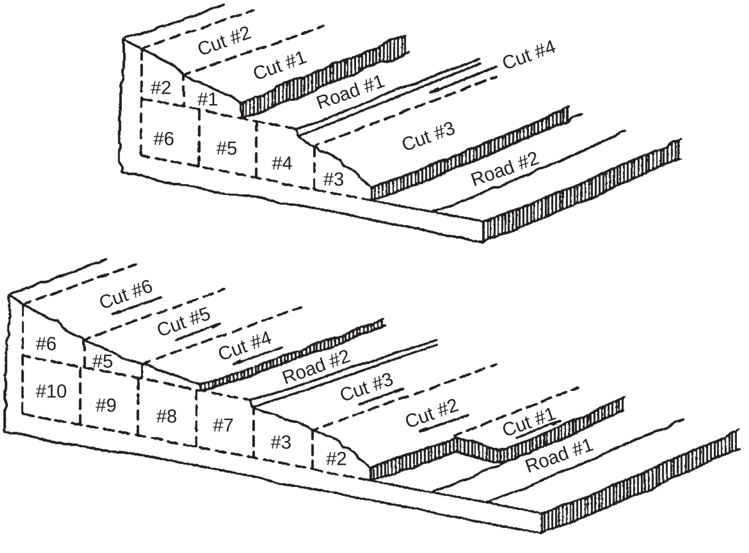


Figure 4.32. Shovel cut sequence when initiating benching in a hilly terrain (Nichols, 1956).

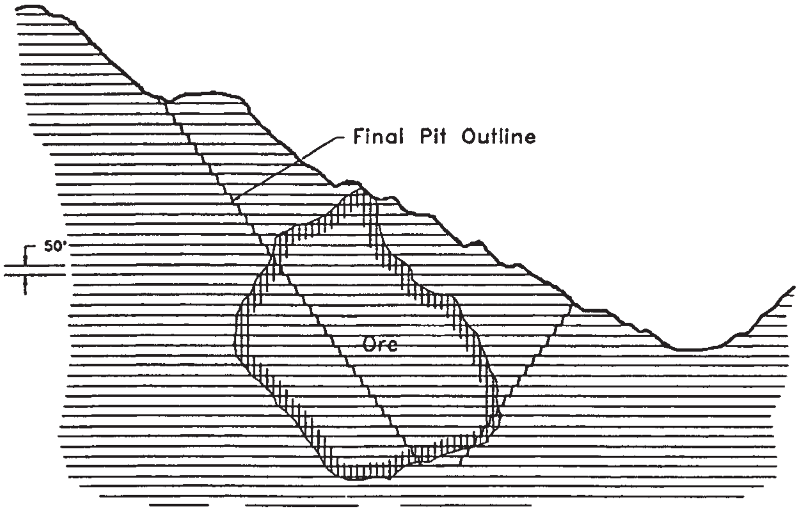


Figure 4.33. Final pit outline superimposed on a section.

4.4 THE PIT EXPANSION PROCESS

4.4.1 *Introduction*

When the drop cut has reached the desired grade, the cut is expanded laterally. Figure 4.34 shows the steps. Initially (Fig. 4.34A) the operating space is very limited. The trucks must turn and stop at the top of the ramp and then back down the ramp towards the loader. When the pit bottom has been expanded sufficiently (Fig. 4.34B), the truck can turn around on the pit bottom. Later as the working area becomes quite large (Fig. 4.34C) several loaders can be used at the same time. The optimum face length assigned to a machine varies with the size and type. It is of the range 200 to 500 ft.

Once access has been established the cut is widened until the entire bench/level has been extended to the bench limits. There are three approaches which will be discussed here:

1. Frontal cuts.
2. Parallel cuts – drive by.
3. Parallel cuts – turn and back.

The first two apply when there is a great deal of working area available, for example at the pit bottom. The mining of more narrow benches on the sides of the pit is covered under number three.

4.4.2 *Frontal cuts*

The frontal cut is shown diagrammatically in Figure 4.35.

The shovel faces the bench face and begins digging forward (straight ahead) and to the side. A niche is cut in the bank wall. For the case shown, double spotting of the trucks is used. The shovel first loads to the left and when the truck is full, he proceeds with the truck on the right. The swing angle varies from a maximum of about 110° to a minimum of 10°. The average swing angle is about 60° hence the loading operation is quite efficient. There

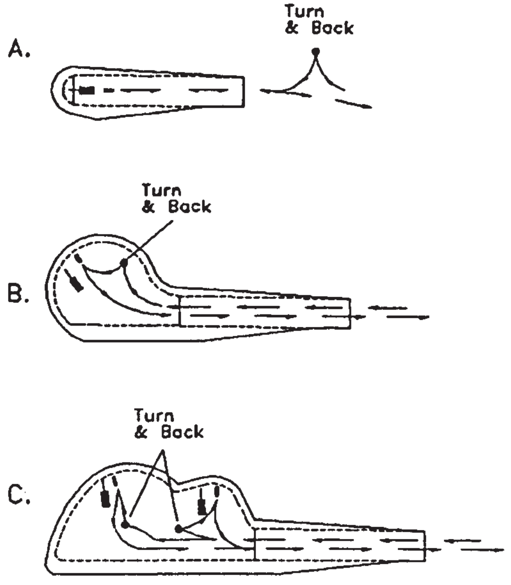


Figure 4.34. Detailed steps in the development of a new production level (Carson, 1961).

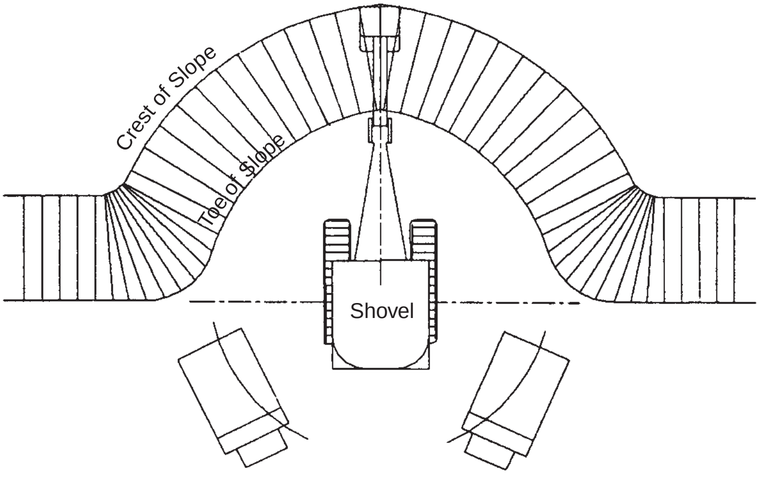


Figure 4.35. Diagrammatic representation of a frontal cutting operation.

must be room for the trucks to position themselves around the shovel. The shovel penetrates to the point that the center of swing is in line with the face. It then moves parallel to itself and takes another frontal cut (Fig. 4.36).

With a long face and sufficient bench width, more than one shovel can work the same face (Fig. 4.37). A fill-in cut is taken between the individual face positions (Fig. 4.38). From the shovels view point this is a highly efficient loading operation. The trucks must however stop and back into position.

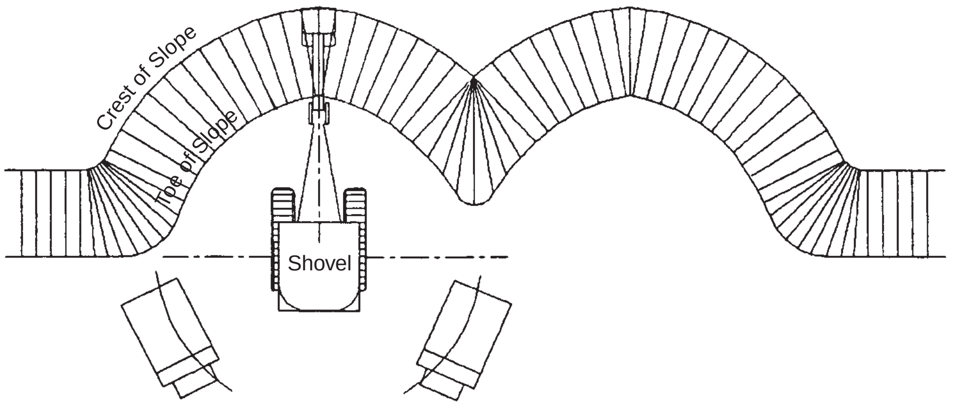


Figure 4.36. Shovel move to adjacent cutting position.

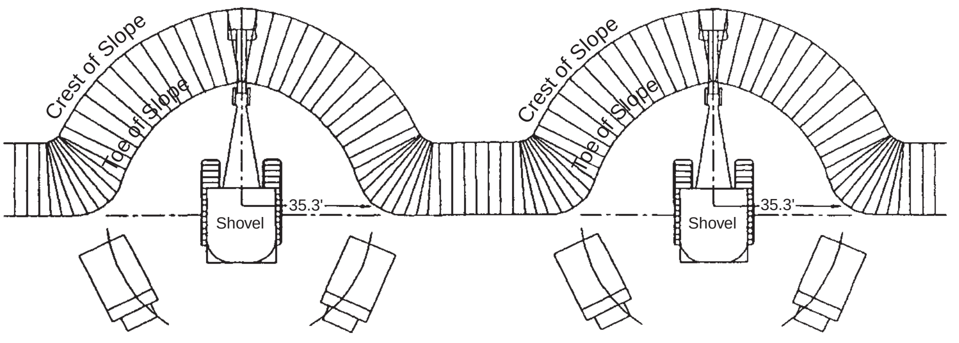


Figure 4.37. Two shovels working the same face.

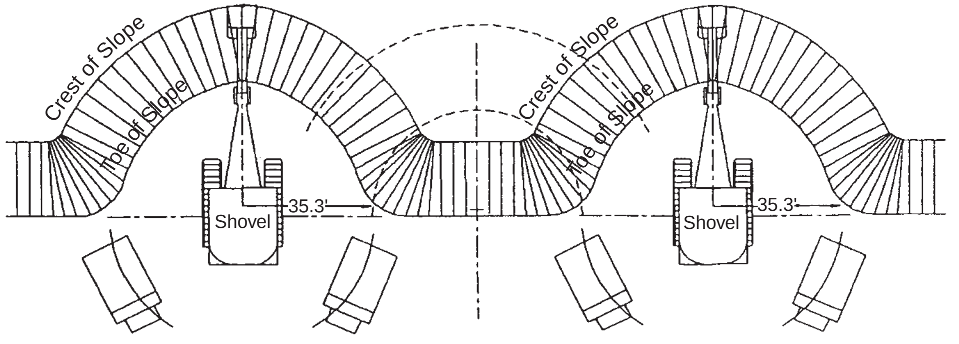


Figure 4.38. Fill-in cutting to complete the face.

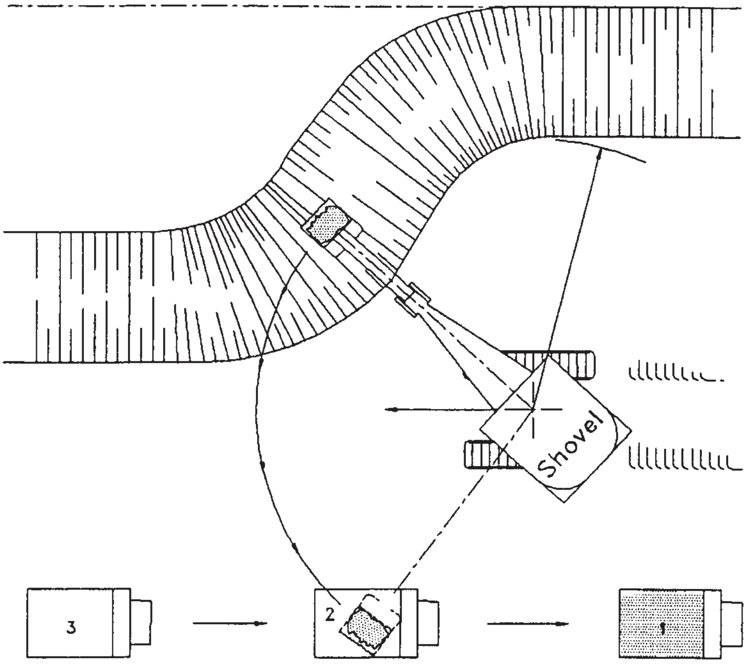


Figure 4.39. Parallel cut with drive-by.

4.4.3 *Drive-by cuts*

Another possibility when the mine geometry allows is the parallel cut with drive-by. This is shown diagrammatically in Figure 4.39. The shovel moves across and parallel to the digging face. For this case, bench access for the haul units must be available from both directions. It is highly efficient for both the trucks and the loader. Although the average swing angle is greater than for the frontal cut, the trucks do not have to back up to the shovel and spotting is simplified.

4.4.4 *Parallel cuts*

The expansion of the pit at the upper levels is generally accomplished using parallel cuts. Due to space limitations there is only access to the ramp from one side of the shovel. This means that the trucks approach the shovel from the rear. They then stop, turn and back into load position. Sometimes there is room for the double spotting of trucks (Fig. 4.40) and sometimes for only single spotting (Fig. 4.41).

Pit geometry is made up of a series of trade-offs. Steeper slopes result in a savings of stripping costs. On the other hand they can, by reducing operating space, produce an increase in operating costs.

Figure 4.42 shows the single spotting sequence. Truck 2 (Fig. 4.42B) waits while the shovel completes the loading of truck 1. After truck 1 has departed (Fig. 4.42C), truck 2 turns and stops (Fig. 4.42D) and backs into position (Fig. 4.42E). While truck 2 is being

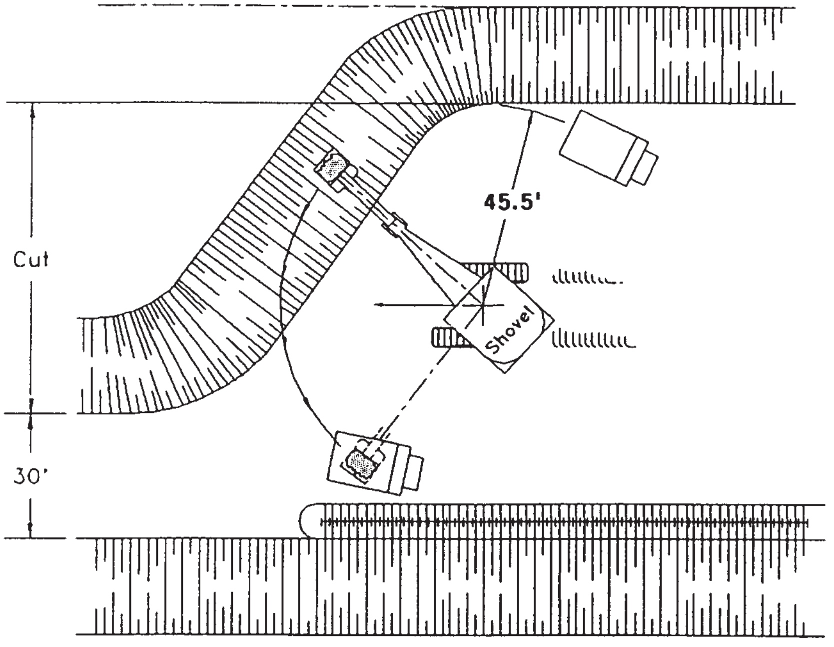


Figure 4.40. Parallel cut with the double spotting of trucks.

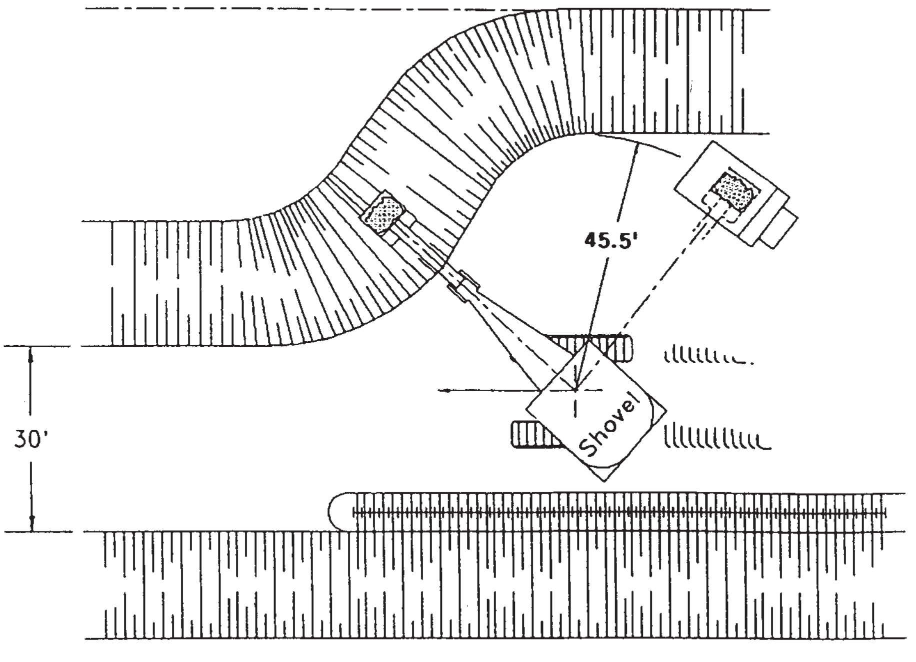


Figure 4.41. Parallel cut with the single spotting of trucks.

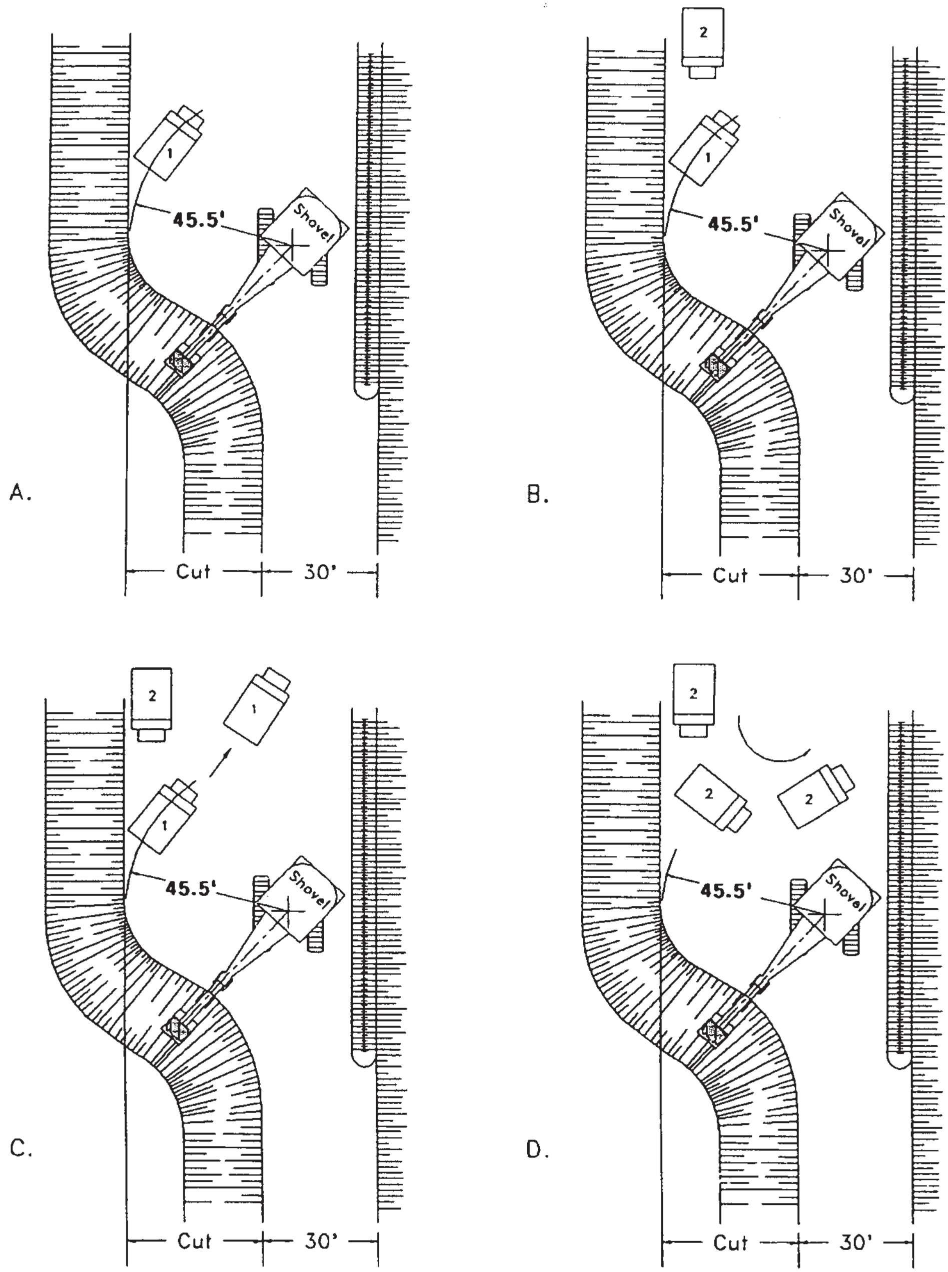


Figure 4.42. Time sequence showing shovel loading with single spotting.

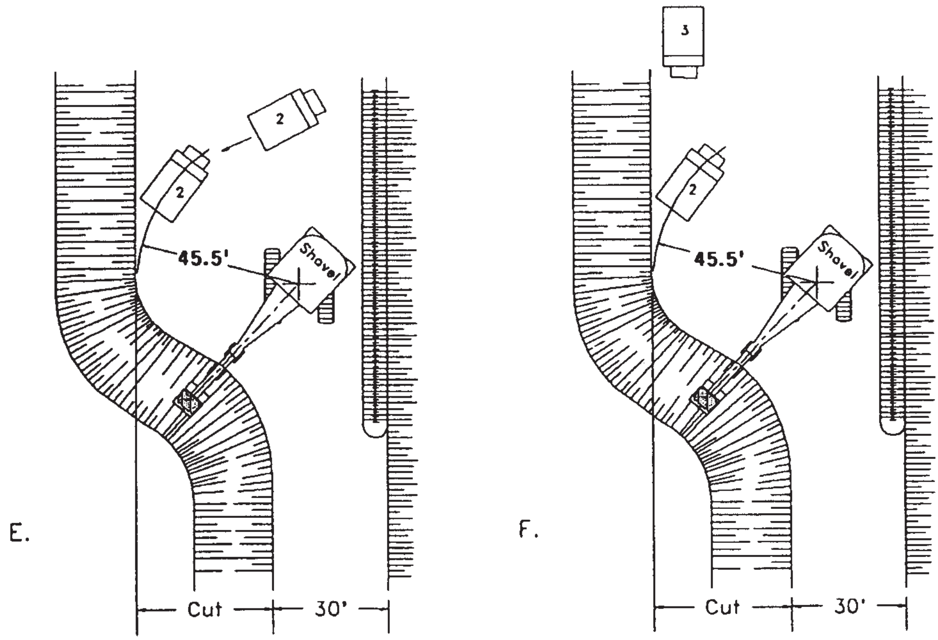


Figure 4.42. (Continued).

loaded truck 3 arrives (Fig. 4.42F). The process then repeats. In this situation both the trucks and the shovel must wait causing a reduction in the overall productivity.

The double spotting situation is shown in Figure 4.43. Truck 1 is first to be loaded (Fig. 4.43A).

Truck 2 arrives (Fig. 4.43B) and backs into position (Fig. 4.43C). When it is just in position the shovel has completed the loading of truck 1. As truck 1 departs (Fig. 4.43D) the shovel begins the loading of truck 2. As truck 2 is being loaded truck 3 arrives. It turns (Fig. 4.43E) and backs into position (Fig. 4.43F). As truck 2 leaves the shovel begins loading truck 3 (Fig. 4.43G). With this type of arrangement there is no waiting by the shovel and less waiting by the trucks. Thus the overall productivity of this system is higher than that for single spotting. The sequencing is unfortunately quite often not as the theory would suggest. Figures 4.43H and 4.43I show two rather typical situations. Both of these can be minimized through the use of an effective communications/dispatching system.

4.4.5 *Minimum required operating room for parallel cuts*

In the previous section the physical process by which a pit is expanded using parallel cuts was described. In this section, the focus will be on determining the amount of operating room required to accommodate the large trucks and shovels involved in the loading operation.

The dimension being sought is the width of the working bench. The working bench is that bench in the process of being mined. This width (which is synonymous with the term 'operating room') is defined as the distance from the crest of the bench providing the floor

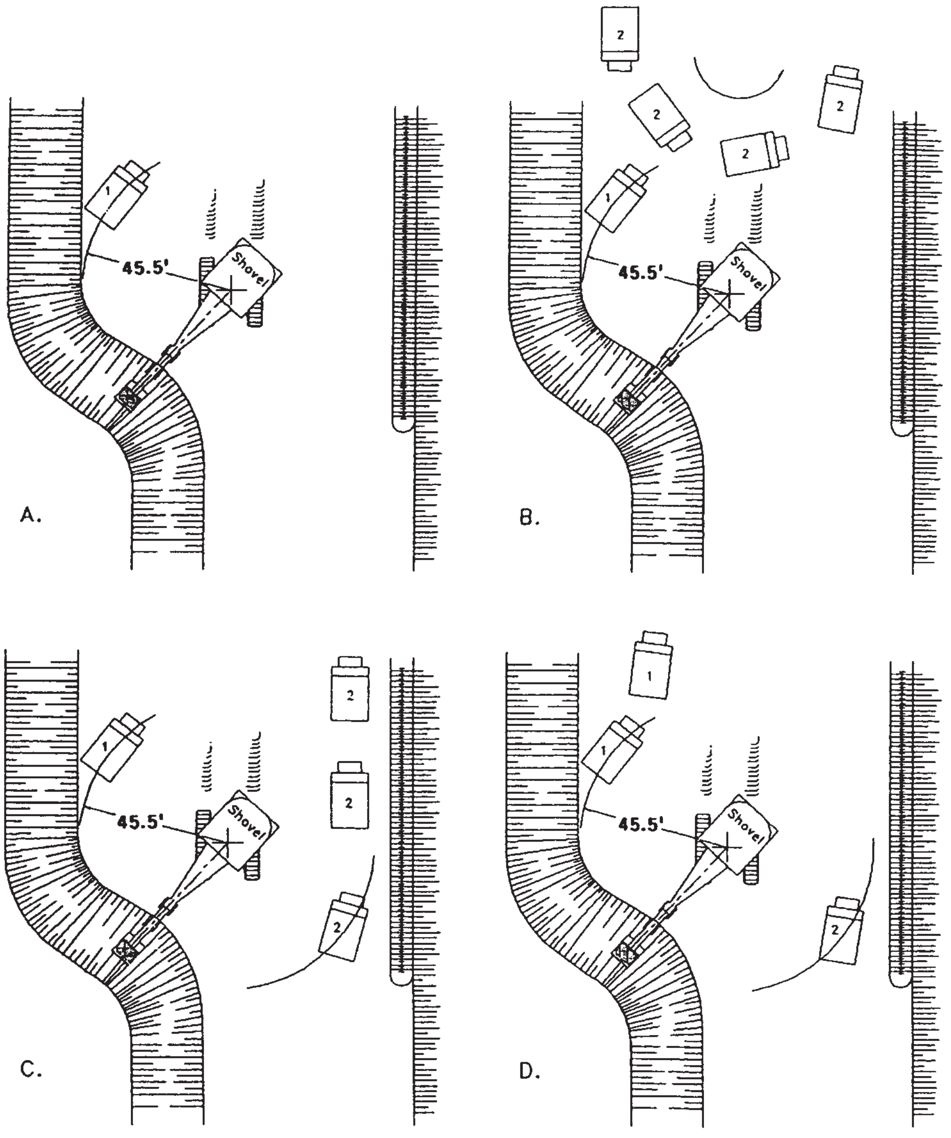


Figure 4.43. Time sequence showing shovel loading with double spotting.

for the loading operations to the bench toe being created as the parallel cut is being advanced. The minimum amount of operating room varies depending upon whether single or double spotting of trucks is used, with the latter obviously requiring somewhat more. The minimum width (W_B) is equal to the width of the minimum required safety bench (S_B) plus the width of the cut (W_C) being taken. This is expressed as

$$W_B = S_B + W_C$$

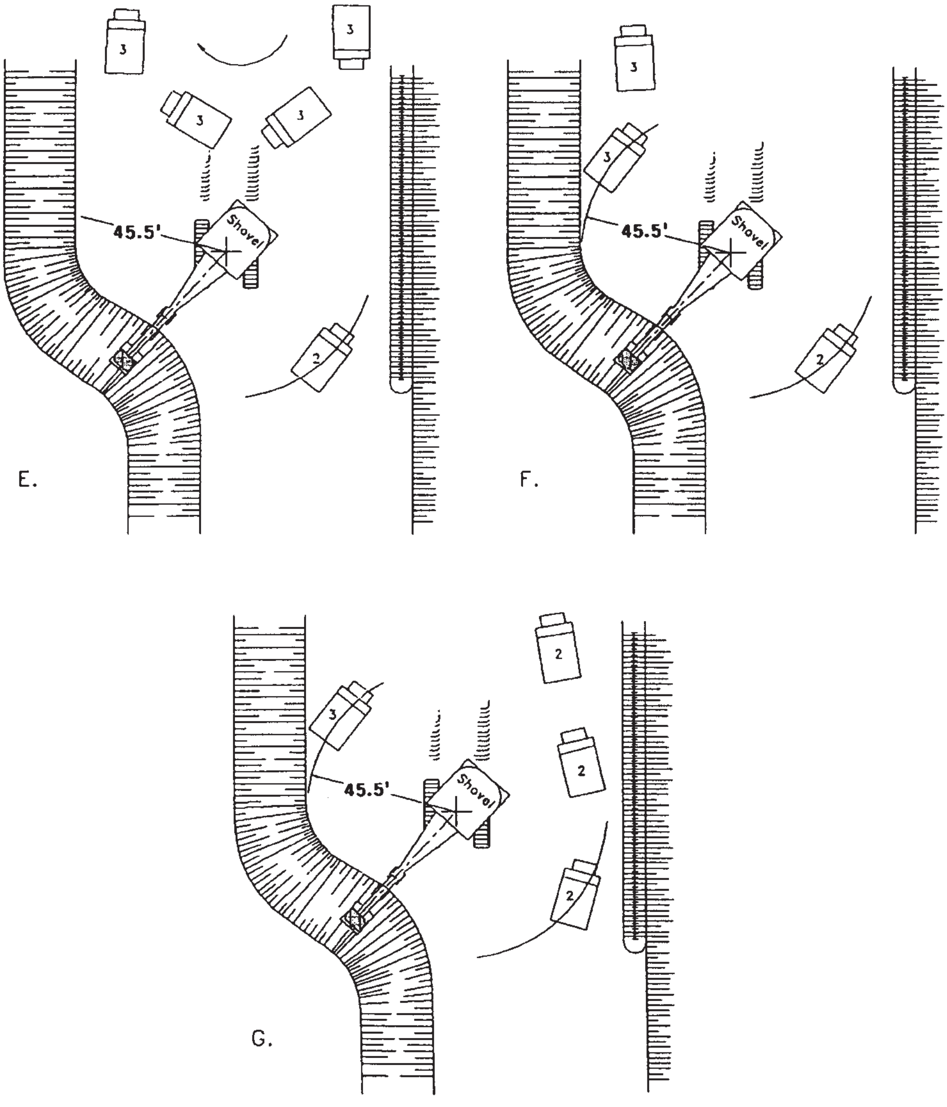


Figure 4.43. (Continued).

The easiest way of demonstrating the principles involved is by way of example. For this, the following assumptions will be made:

- Bench height = 40 ft.
- A safety berm is required.
- The minimum clearance between the outer truck tire and the safety berm = 5 ft.
- Single spotting is used.
- Bench face angle = 70°.
- Loading is done with a 9 yd^3 BE 155 shovel (specifications given in Fig. 4.9).
- Haulage is by 85 ton capacity trucks.

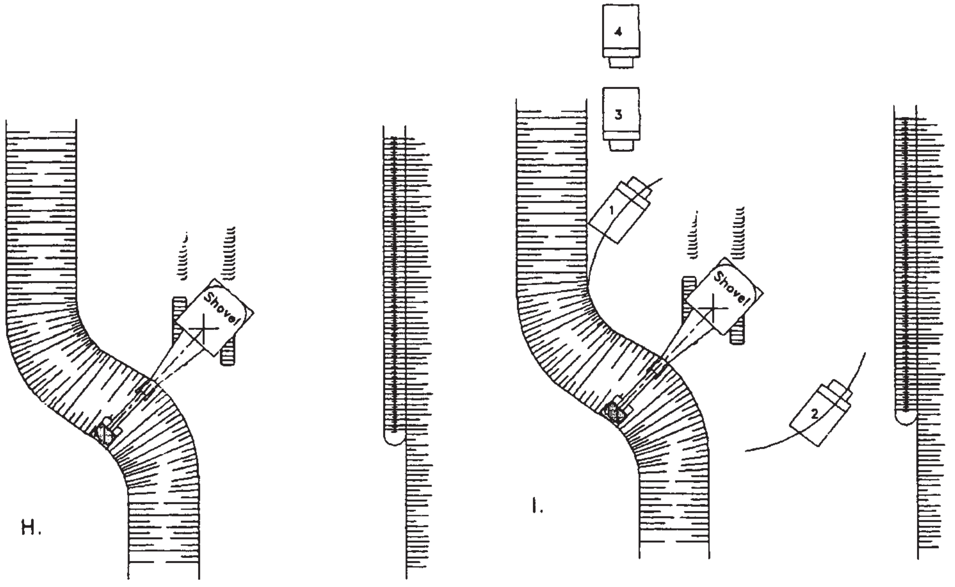


Figure 4.43. (Continued).

– Truck width = 16 ft.
– Tire rolling radius = 4 ft.

The general arrangement in plan and section is shown in Figure 4.44. The design shows that:

$$\text{Working bench width} = 102\,\text{ft}$$

$$\text{Cut width} = 60\,\text{ft}$$

$$\text{Safety bench width} = 42\,\text{ft}$$

The basic calculations (justification) behind these numbers will now be presented.

Step 1. A safety berm is required along the edge of this bench. As will be discussed in Subsection 4.9.5, the height of the berm should be of the order of the tire rolling radius. For this truck, the berm height would be about 4 ft. Assuming that the material has an angle of repose of 45°, the width of the safety berm is 8 ft (see Fig. 4.45). It is assumed that this berm is located with the outer edge at the crest.

Step 2. The distance from the crest to the truck centerline is determined assuming parallel alignment. A 5′ clearance distance between the safety berm and the wheels has been used. Since the truck is 16 ft wide, the centerline to crest distance (T_C) is 21 ft.

Step 3. The appropriate shovel dimensions are read from the specification sheet (Fig. 4.9):

(a) Shovel centerline to truck centerline. This is assumed to be the dumping radius (B) at maximum height,

$$B = 45'6'' = 45.5\,\text{ft}$$

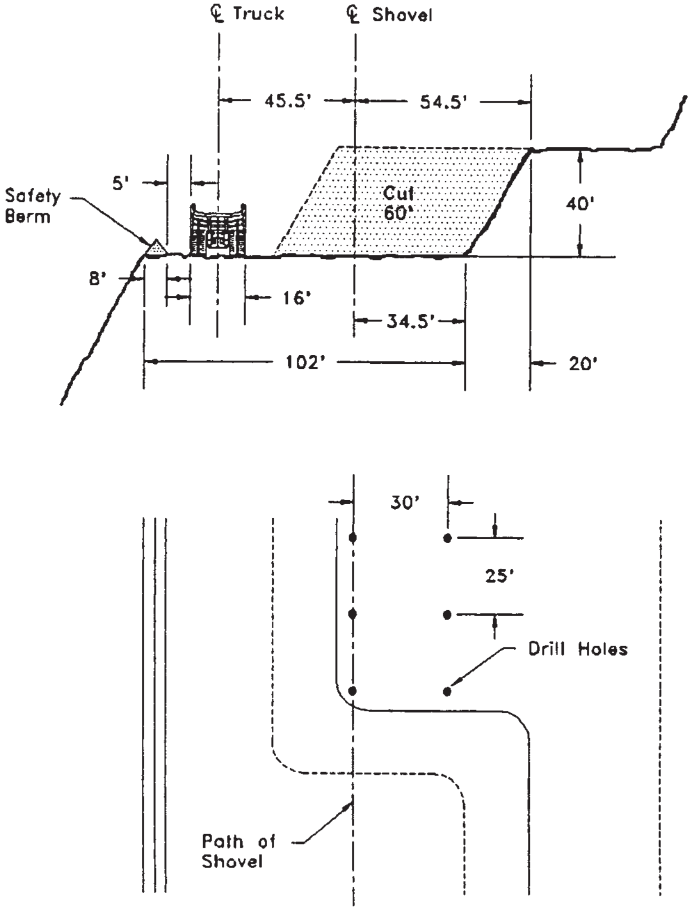


Figure 4.44. Section and plan views through a working bench.

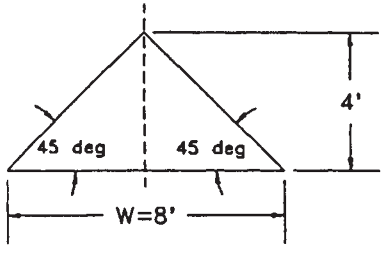


Figure 4.45. Simplified representation of a safety berm.

(b) The maximum dumping height (A) is more than sufficient to clear the truck,

$$A = 28\,\text{ft}$$

(c) The level floor radius dimension (G) is the maximum distance from the shovel centerline which the floor can be cleaned. In this case

$$G = 35'3'' = 35.25\,\text{ft}$$

This will be used as the maximum shovel centerline to toe distance.

Step 4. The desired working bench dimension becomes

$$W_B = T_C + B + G = 21 + 45.5 + 35.25 \cong 102\,\text{ft}$$

Step 5. The corresponding width of cut is now calculated. In this case it has been assumed that the shovel moves along a single path parallel to the crest. Information from the shovel manufacturer suggests that the maximum cutting width (W_C) may be estimated by

$$W_C = 0.90 \times 2 \times G = 0.90 \times 2 \times 35.25 \cong 63.5\,\text{ft}$$

This applies to the width of the pile of broken material. Therefore, to allow for swell and throw of the material during blasting, the design cut width should be less than this value. Here a value of 60 ft has been assumed.

Step 6. Knowing the width of the working bench and the cut width, the resulting safety bench has a width

$$S_B = 102 - 60 = 42\,\text{ft}$$

This is of the order of the bench height (40 ft) which is a rule of thumb sometimes employed.

Step 7. Some check calculations are made with regard to other dimensions.

a) The maximum cutting height of the shovel

$$D = 43'6'' = 43.5\,\text{ft}$$

is greater than the 40′ bench height. Thus the shovel can reach to the top of the bench face for scaling.

b) The maximum shovel cutting radius (E) is

$$E = 54'6'' = 54.5\,\text{ft}$$

Since the maximum radius of the level floor (G) is

$$G = 35'3'' = 35.25\,\text{ft}$$

the flattest bench face angle which could be scaled (Fig. 4.46) is

$$\text{Slope} = \tan^{-1} \frac{40}{54.50 - 35.25} = 64.3°$$

Thus the shovel can easily scale the 70° bench face.

Step 8. The cut dimension should be compared to the drilling and blasting pattern being used. In this particular case the holes are 12¼ ins. in diameter (D_e) and ANFO is the explosive. Using a common rule of thumb, the burden (B) is given by

$$B = 25\frac{D_e}{12} \cong 25\,\text{ft}$$

The hole spacing (S) is equal to the burden

$$S = 25\,\text{ft}$$

Thus two rows of holes are appropriate for this cut width.

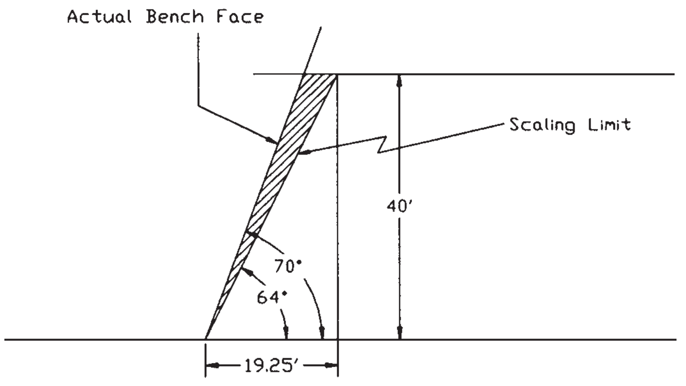


Figure 4.46. The bench/bench face geometry for the example.

A somewhat simplified approach has been applied to the matter of determining working bench width. The complications arise when one examines the best width from an overall economic viewpoint.

As will be discussed in Section 4.5, the working bench is generally one of a set of 3 to 5 benches being mined as a group. The others in the set each have a width equal to that of a safety bench. As the cut is extracted, the remaining portion of the working bench is reduced to a safety bench width. Since the width of the working bench is approximately equal to the combined widths of the others in the set, it has a major impact on the overall slope angle. A wider working bench means that the slope angle is flatter with the extra costs related to earlier/more stripping, but the equipment operating efficiency is higher (with lower related costs). On the other hand a more narrow working bench would provide a steeper overall slope at the cost of operating efficiency. Thus, there are other factors, beside those related to equipment geometries, which must be considered.

4.4.6 *Cut sequencing*

In the previous section the terms ‘working bench,’ ‘cut’ and ‘safety bench’ were introduced. These will now be applied to a simple example in which a 90 ft wide cut 1000 ft long will be taken from the right hand wall of the pit shown in Figure 4.47. As can be seen the wall consists of 4 benches. The entire bench 1 (B1) is exposed at the surface. Benches B2, B3 and B4 are safety benches, 35 ft wide. The process begins with the drill working off the upper surface of B1. The holes forming the cut to be taken from B1 are drilled and blasted (Fig. 4.48). The shovel then moves along the floor of bench B1 (upper surface of B2) and loads the trucks which also travel on this surface. The working bench has a width of 125 ft. When the cut is completed the geometry is as shown in Figure 4.49. The cut to be taken from bench 2 is now drilled and blasted. The shovel moves along the top of bench 3 taking a cut width (W_C). A portion of bench 2 remains as a safety bench. The process is repeated until the bottom of the pit is reached. The shovel then moves back up to bench 1 and the process is repeated. If it is assumed that the shovel can produce 10,000 tons/day, then the overall production from these 4 benches is 10,000 tons/day. The four benches associated with this shovel are referred to as a mine production unit.

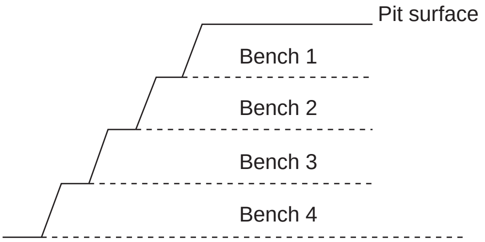


Figure 4.47. Initial geometry for the push back example.

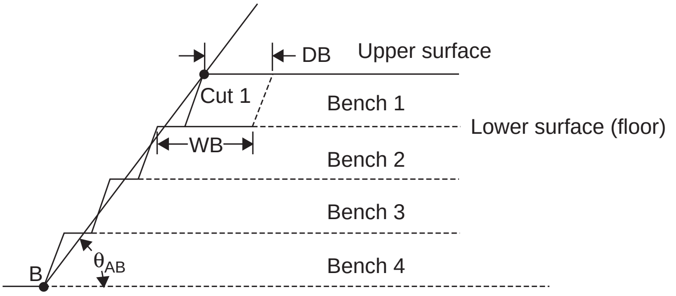


Figure 4.48. Cut mining from bench 1.

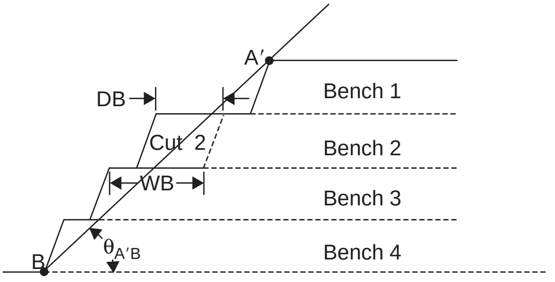


Figure 4.49. Cut mining from bench 2.

4.5 PIT SLOPE GEOMETRY

There are a number of 'slopes' which enter into pit design. Care is needed so that there is no confusion as to how they are calculated and what they mean. One slope has already been introduced. That is the bench face angle (Fig. 4.50). It is defined as the angle made with the horizontal of the line connecting the toe to the crest. This definition of the slope going from the toe to the crest will be maintained throughout this book.

Now consider the slope consisting of 5 such benches (Fig. 4.51). The angle made with the horizontal of the line connecting the lowest most toe to the upper most crest is defined as the overall pit slope,

$$\Theta(\text{overall}) = \tan^{-1} \frac{5 \times 50}{4 \times 35 + \frac{5\times 50}{\tan 75^\circ}} = 50.4^\circ$$

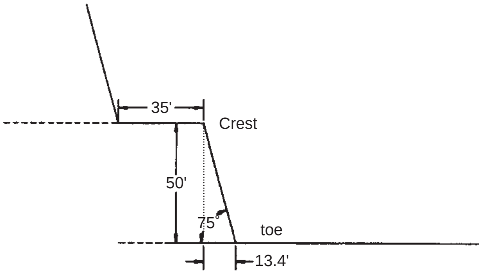


Figure 4.50. Safety bench geometry showing bench face angle.

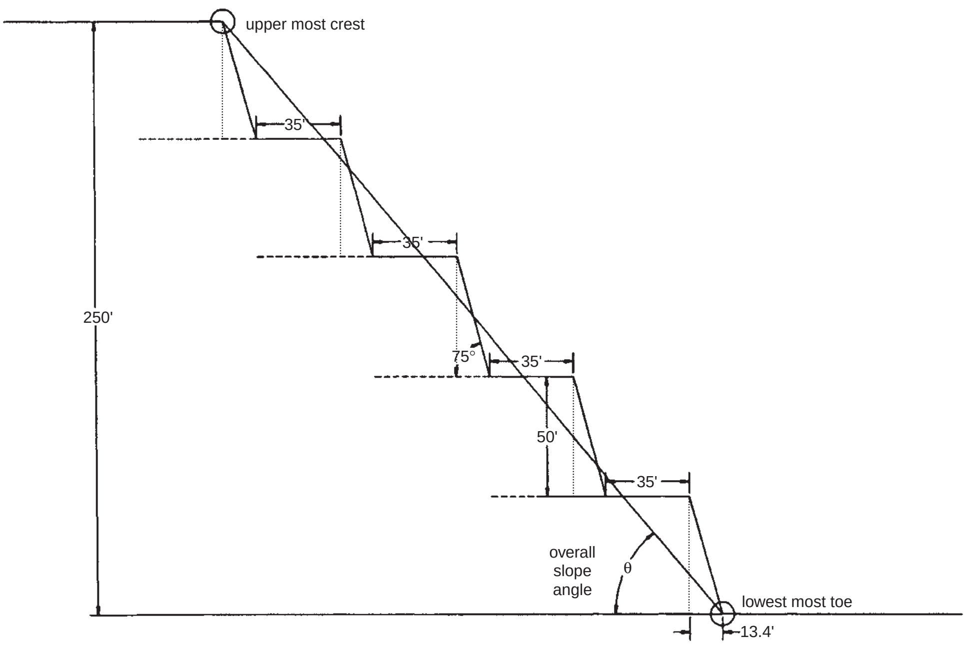


Figure 4.51. Overall slope angle.

If as is shown in Figure 4.52 an access ramp with a width of 100 ft is located half way up bench 3, the overall pit slope becomes

$$\Theta(\text{overall}) = \tan^{-1}\frac{5\times 50}{4\times 35+\frac{5\times 50}{\tan 75^\circ}+100} = 39.2^\circ$$

As can be seen, the presence of the ramp on a given section has an enormous impact on the overall slope angle.

The ramp breaks the overall slope into two portions (Fig. 4.53) which can each be described by slope angles. These angles are called interramp angles (between-the-ramp angles). In this case

$$\mathrm{IR}_1 = \mathrm{IR}_2 = \tan^{-1}\frac{125}{2\times 35+\frac{2\times 50}{\tan 75^\circ}+\frac{25}{\tan 75^\circ}} = 50.4^\circ$$

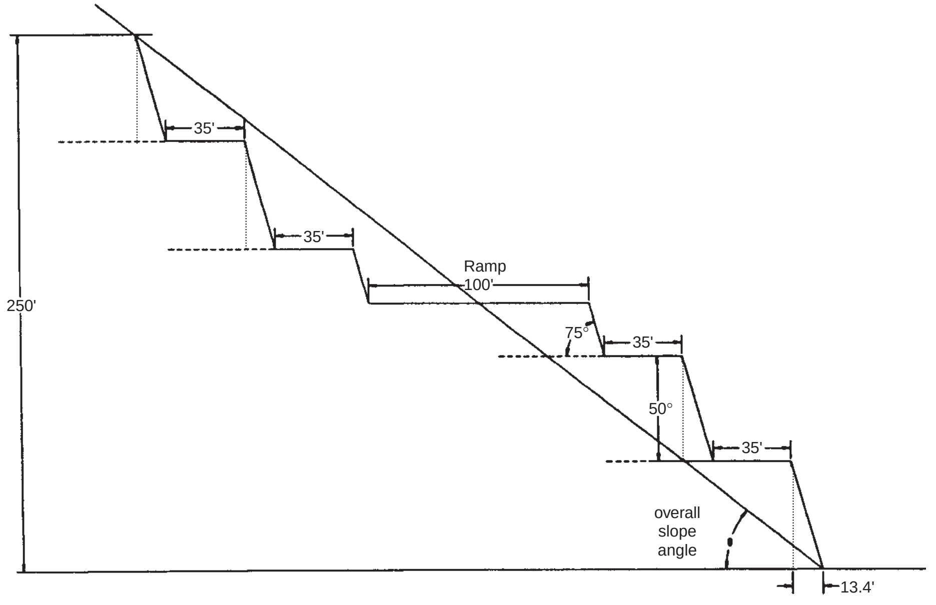


Figure 4.52. Overall slope angle with ramp included.

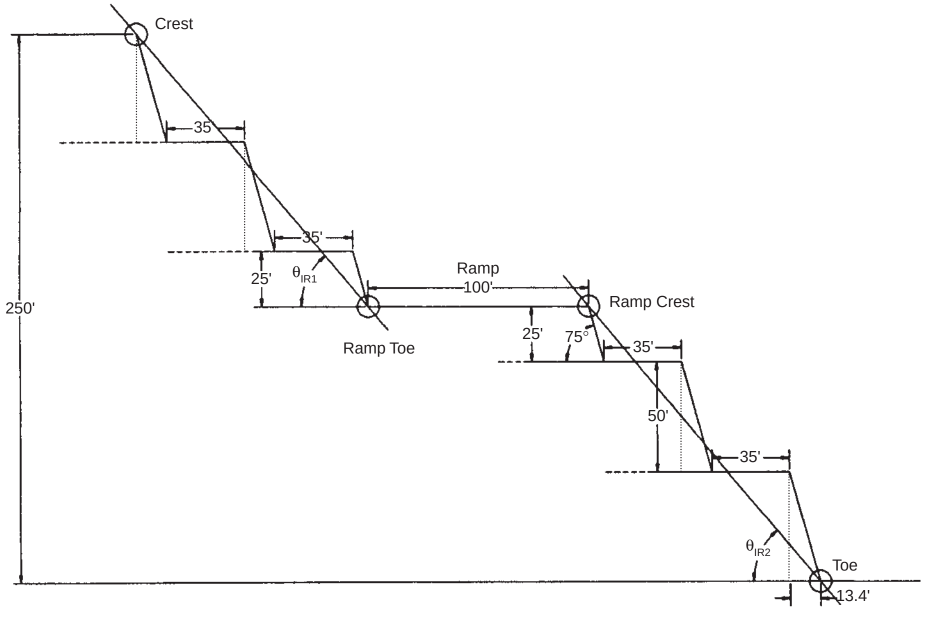


Figure 4.53. Interramp slope angles for Figure 4.52.

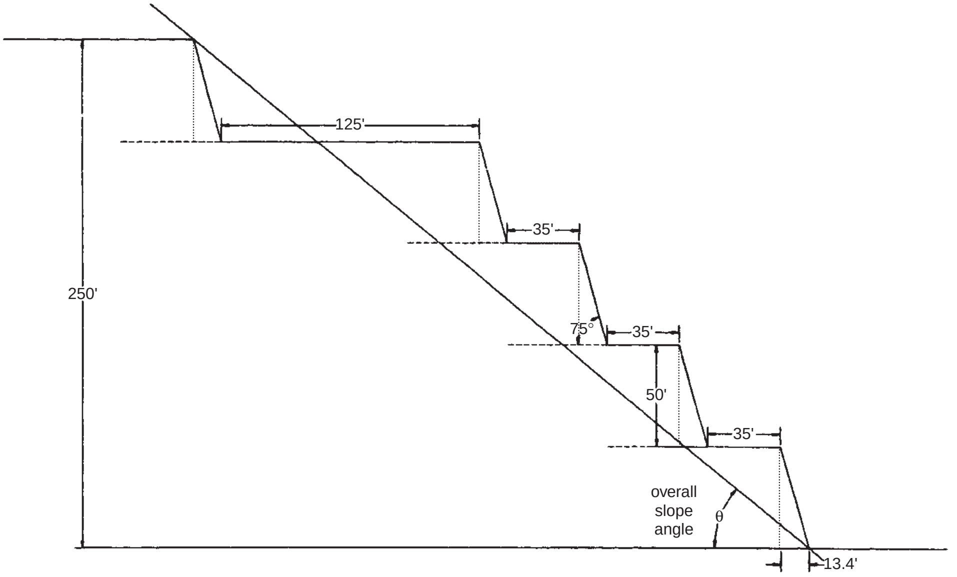


Figure 4.54. Overall slope angle with working bench included.

The interramp wall height is 125 ft for each segment. Generally the interramp wall heights and angles for the different slope segments would not be the same. From a slope stability viewpoint each interramp segment would be examined separately.

While active mining is underway, some working benches would be included in the overall slope. Figure 4.54 shows a working bench 125 ft in width included as bench 2. The overall slope angle is now

$$\Theta = \tan^{-1} \frac{5 \times 50}{125 + 4 \times 35 + \frac{5 \times 50}{\tan 75°}} = 37.0°$$

The working bench is treated in the same way as a ramp in terms of interrupting the slope. The two interramp angles are shown in Figure 4.55. In this case

$$\Theta_{IR_1} = 75°$$

$$\Theta_{IR_2} = \tan^{-1} \frac{200}{3 \times 35 + \frac{4 \times 50}{\tan 75°}} = 51.6°$$

The interramp wall heights are

$$H_1 = 50'$$

$$H_2 = 200'$$

For this section, it is possible that the ramp cuts bench 3 as before. This situation is shown in Figure 4.56.

The overall slope angle has now decreased to

$$\Theta = \tan^{-1} \frac{250}{125 + 3 \times 35 + 100 + \frac{5 \times 50}{\tan 75°}} = 32.2°$$

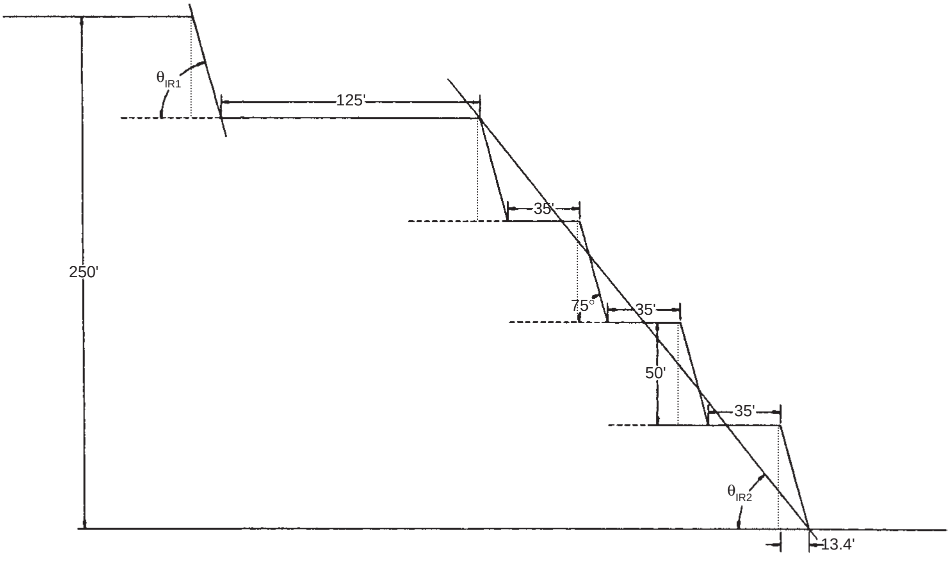


Figure 4.55. Interramp angles associated with the working bench.

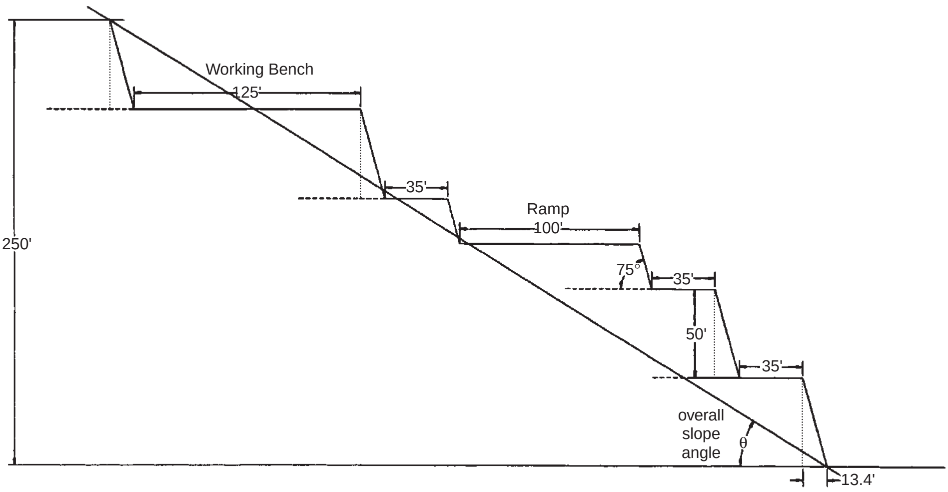


Figure 4.56. Overall slope angle with one working bench and a ramp section.

As shown in Figure 4.57, there are now three interramp portions of the slope. The interramp wall heights and angles are:

Segment 1:

$$\Theta_{IR_1} = 75^\circ$$
$$H = 50'$$

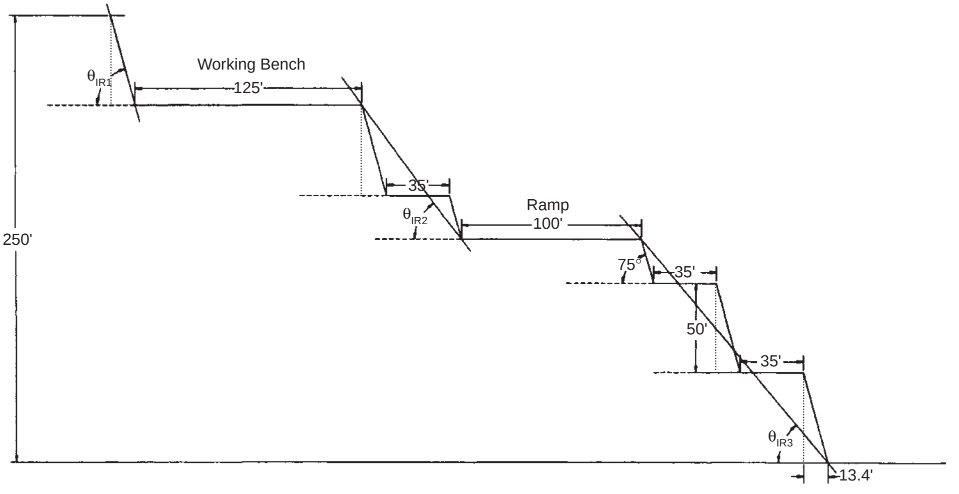


Figure 4.57. Interramp slope angles for a slope containing a working bench and a ramp.

Segment 2:

$$\Theta_{IR_2} = \tan^{-1}\frac{75}{35 + \frac{75}{\tan 75°}} = 35.7°$$

$$H = 75'$$

Segment 3:

$$\Theta_{IR_3} = \tan^{-1}\frac{125}{2 \times 35 + \frac{125}{\tan 75°}} = 50.4°$$

$$H = 125'$$

In Figure 4.57, the overall slope is shown to contain one working bench. Under some circumstances there may be several working benches involved in the mining of the slope. Figure 4.58 shows the case of a slope with 6 benches of which two are working benches 125 ft in width.

The overall (working slope) is given by

$$\Theta = \tan^{-1}\frac{300}{3 \times 35 + 2 \times 125 + \frac{300}{\tan 75°}} = 34.6°$$

The slope associated with each shovel working group is shown in Figure 4.59. In this case it is

$$\Theta = \tan^{-1}\frac{150}{125 + 35 + \frac{150}{\tan 75°}} = 36.8°$$

If the number of working benches is increased to 3 for the slope containing 6 benches, the overall slope would be further reduced. Thus to maintain reasonable slope angles, most mines have one working bench for a group of 4 to 5 benches.

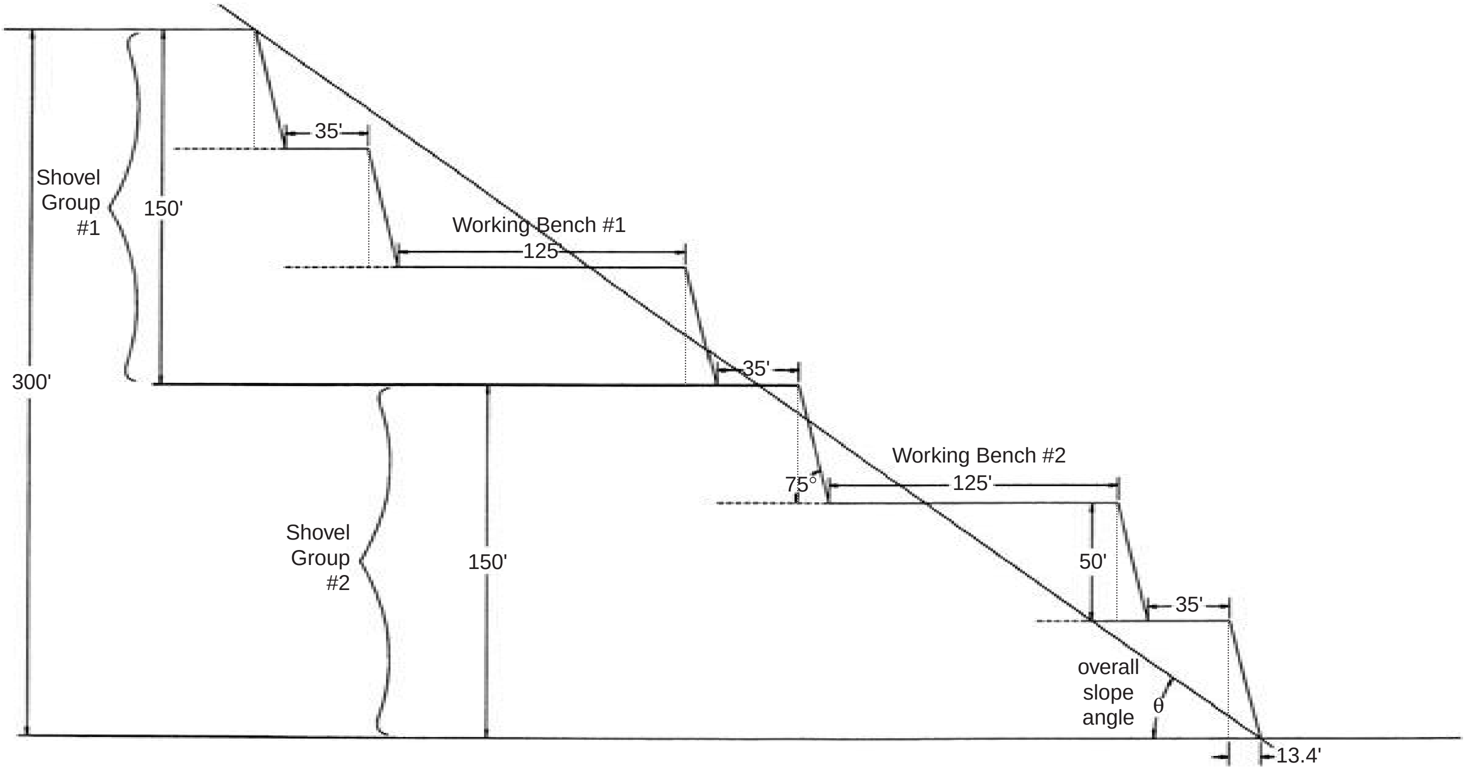


Figure 4.58. Overall slope angle for a slope containing two working benches.

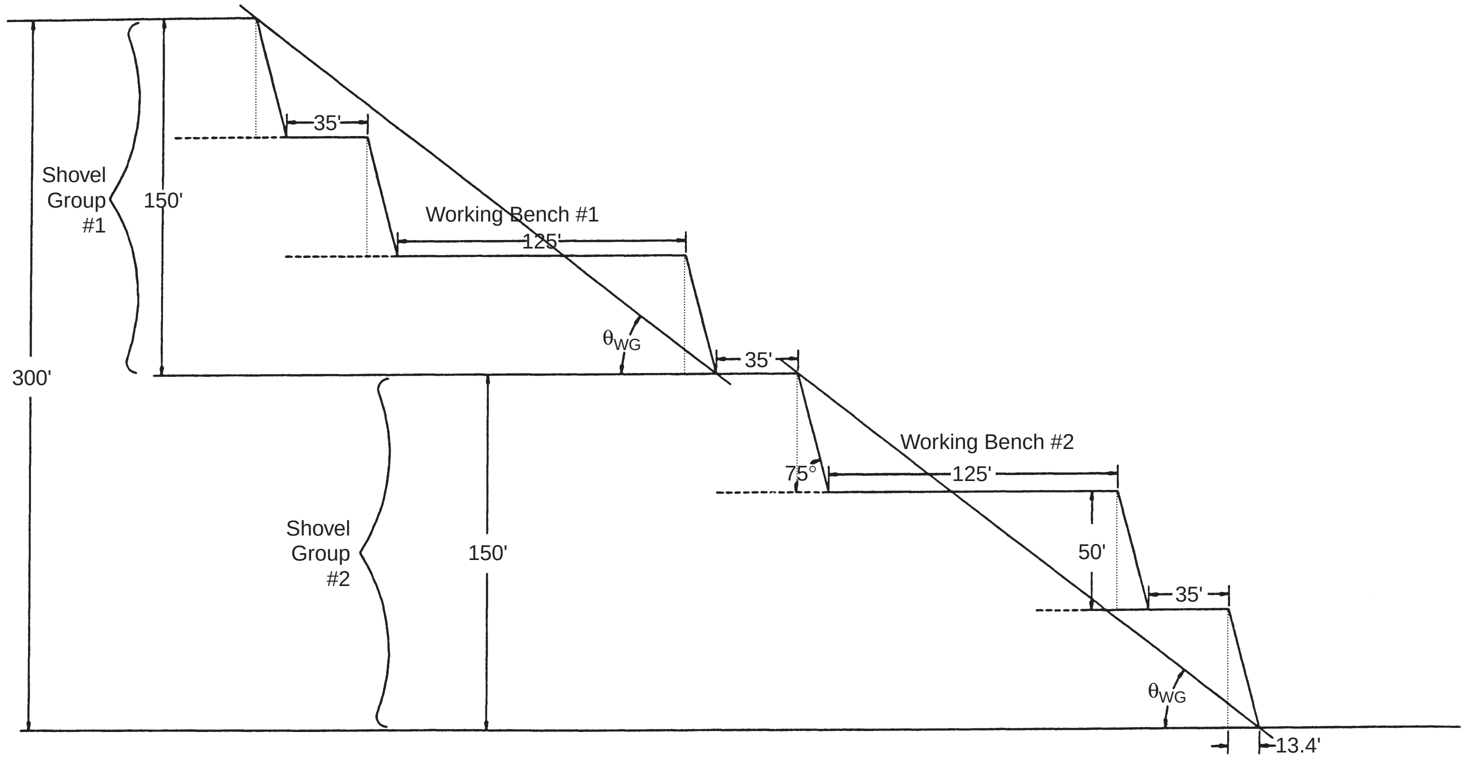


Figure 4.59. Slopes for each working group.

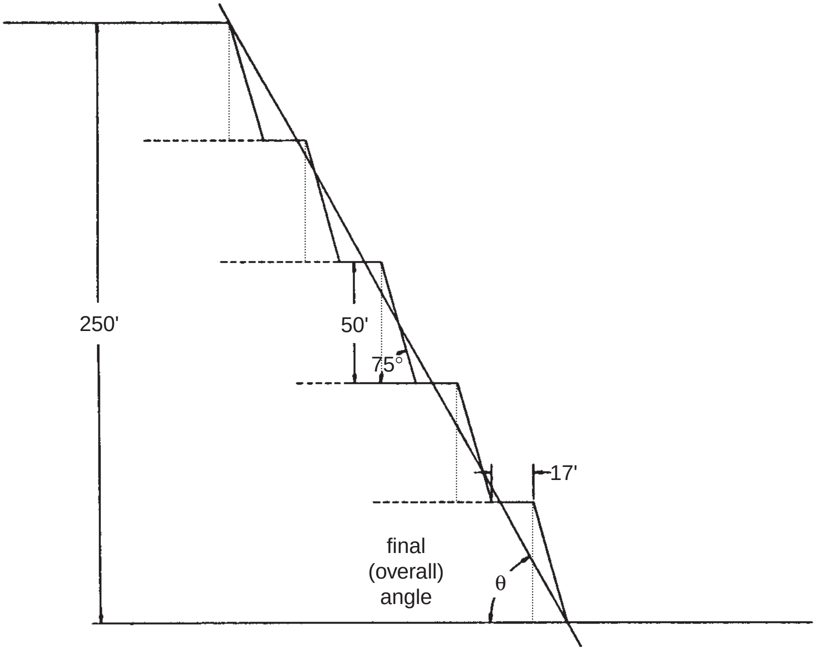


Figure 4.60. Final overall pit slope.

At the end of mining it is desired to leave the final slope as steep as possible. Some of the safety benches will be reduced in width while others may be eliminated entirely. For final walls, a bench width of approximately ⅓ of the bench height is commonly used. For this example with a bench height of 50 ft, the bench width becomes 17 ft. The final pit slope angle, assuming no ramp is needed on this wall (Fig. 4.60), becomes

$$\Theta(\text{final}) = \tan^{-1}\frac{250}{4\times 17 + \frac{250}{\tan 75°}} = 61.6°$$

If the final bench faces could have been cut at 90° instead of 75°, then the final overall pit slope angle would be

$$\Theta(\text{final}) = \tan^{-1}\frac{250}{4\times 17} = 74.8°$$

It is much more likely that the final face angles are 60° and the safety benches 20 ft wide. This gives

$$\Theta(\text{final}) = \tan^{-1}\frac{250}{4\times 20 + \frac{250}{\tan 60°}} = 48°$$

Although much regarding final slope angles has to do with rock structure, care in blasting can make a major impact.

Table 4.2. Classification of open pit slope problems (Hoek, 1970b).

	Category	Conditions	Method of solution
A.	Unimportant slopes	Mining a shallow high grade orebody in favorable geological and climatic conditions. Slope angles unimportant economically and flat slopes can be used.	No consideration of slope stability required.
B.	Average slopes	Mining a variable grade orebody in reasonable geological and climatic conditions. Slope angles important but not critical in determining economics of mining.	Approximate analysis of slope stability normally adequate.
C.	Critical slopes	Mining a low grade orebody in unfavorable geological and climatic conditions. Slope angles critical in terms of both economics of mining and safety of operation.	Detailed geological and groundwater studies followed by comprehensive stability analysis usually required.

4.6 FINAL PIT SLOPE ANGLES

4.6.1 *Introduction*

During the early feasibility studies for a proposed open pit mine, an estimate of the safe slope angles is required for the calculation of ore to waste ratios and for the preliminary pit layout. At this stage generally the only structural information available upon which to base such an estimate is that obtained from diamond drill cores collected for mineral evaluation purposes. Sometimes data from surface outcrops are also available. How well these final slope angles must be known and the techniques used to estimate them depends upon the conditions (Table 4.2) applicable. During the evaluation stage for categories B and C, the best engineering estimate of the steepest safe slope at the pit limits in each pit segment is used. Since the information is so limited, they are hedged with a contingency factor. If the property is large and has a reasonably long lifetime, initially the exact slope angles are of relatively minor importance. The effect of steeper slopes at the pit limits is to increase the amount of ore that can be mined and therefore increase the life of the mine. The effect of profits far in the future has practically no impact on the net present value of the property.

During the pre-production period and the first few years of production, the operating slopes should however be as steep as possible while still providing ample bench room for optimum operating efficiency. The minimization of stripping at this stage has a significant effect on the overall economics of the operation. The working slopes can then be flattened until they reach the outer surface intercepts. Steepening operations then commence to achieve the final pit slopes (Halls, 1970). Cases do occur where the viability of an orebody is highly dependent on the safe slope angle that can be maintained. Special measures, including the collection of drillhole data simply for making slope determinations are then taken.

There are a number of excellent references which deal in great detail with the design of pit slopes. In particular *Rock Slope Engineering* by Hoek & Bray (1977), and the series of publications developed within the *Pit Slope Manual* series produced by CANMET should be mentioned. This brief section focusses on a few of the underlying concepts, and presents some curves extracted largely from the work of Hoek (1970a, 1970b) which may be used for making very preliminary estimates.

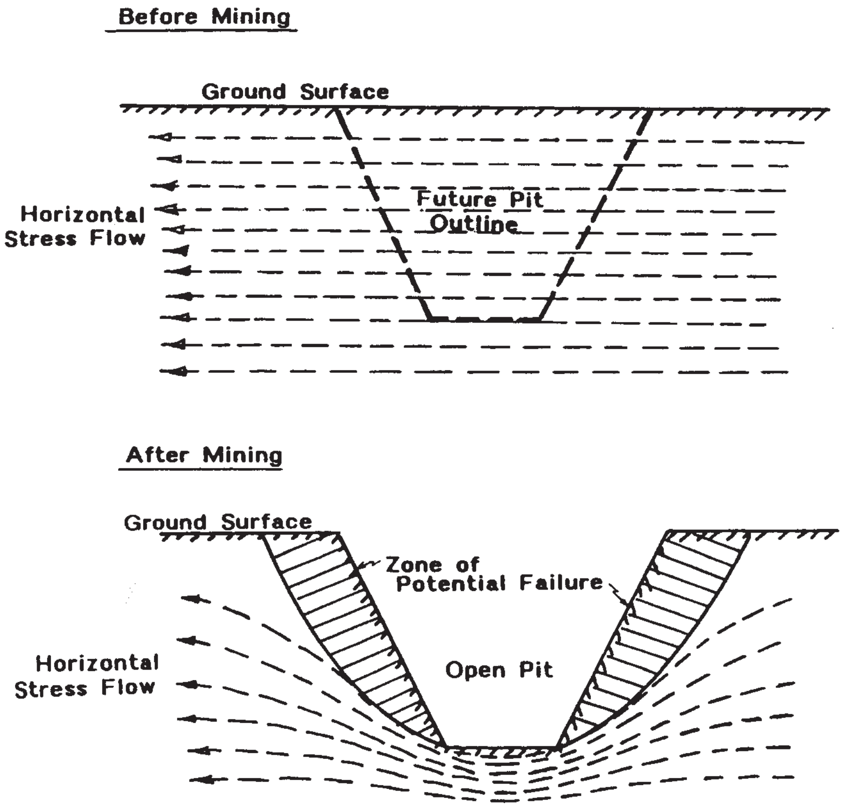
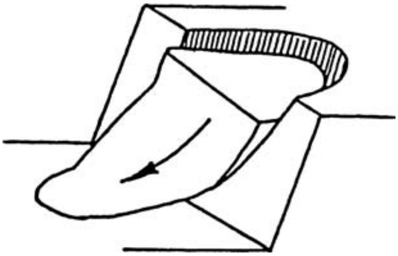


Figure 4.61. Horizontal stress redistribution due to the creation of a pit.

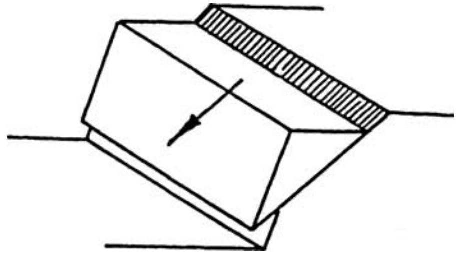
4.6.2 *Geomechanical background*

Figure 4.61 shows diagrammatically the horizontal flow of stress through a particular vertical section both with and without the presence of the final pit. With the excavation of the pit, the pre-existing horizontal stresses are forced to flow beneath the pit bottom (and around the pit ends).

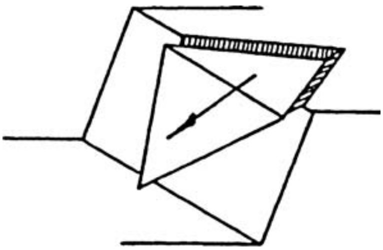
The vertical stresses are also reduced through the removal of the rock overlying the final slopes. This means that the rock lying between the pit outline and these flow lines is largely distressed. As a result of stress removal, cracks/joints can open with a subsequent reduction in the cohesive and friction forces restraining the rock in place. Furthermore, ground water can more easily flow through these zones, reducing the effective normal force on potential failure planes. As the pit is deepened, the extent of this destressed zone increases, and the consequences of a failure becomes more severe. The chances of encountering adverse structures (faults, dykes, weakness zones, etc.) within these zones increase as well. Finally, with increasing pit depth, the relative sizes of the individual structural blocks making up the slopes become small compared to the overall volume involved. Thus the failure mechanism may change from one of structural control to one controlled by the characteristics of a granular mass. Figure 4.62 shows the four major types of failure which occur in an open



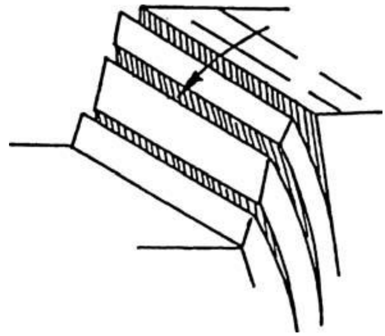
a. Circular failure in overburden soil, waste rock or highly fractured rock with no identifiable structural pattern.



b. Plane failure in rock with highly ordered structure such as slate.



C. Wedge failure on two intersecting discontinuities.



d. Toppling failure in hard rock which can form columnar structure separated by steeply dipping discontinuities.

Figure 4.62. The most common slope failure types (Hoek & Bray, 1977).

pit. In this section the discussion will concentrate on planar failure along major structures and circular failure.

4.6.3 *Planar failure*

Planar failure along various types of discontinuities can occur on the bench scale, interramp scale and pit wall scale (major fault, for example). Bench face instabilities due to the daylighting of major joint planes means that the overall slope must be flattened to provide the space required for adequate safety berms. The final slope is made up of flattened bench faces, coupled with the safety berm steps. The design slope angle may be calculated once an average stable bench face angle is determined. Since one is concerned with final pit wall stability, the analysis in this section applies to a major structure occurring in the pit wall, although the same type of analysis applies on the smaller scale as well. Figure 4.63 shows the dimensions and forces in a rock slope with a potential failure plane. The Mohr-Coulomb failure criterion has been used.

The following definitions apply:

i is the average slope angle from horizontal (degrees),
β is the angle of the discontinuity from the horizontal (degrees),

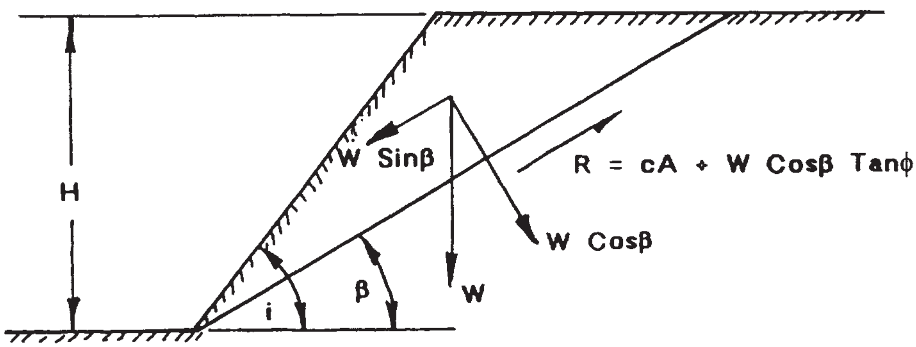


Figure 4.63. Dimensions and forces in a rock slope with a potential failure plane (Hoek, 1970a).

W is the block weight,
R is the resisting force,
c is the cohesion,
ϕ is the friction angle,
$W \cos \beta$ is the normal force,
$W \sin \beta$ is the driving force,
A is the area of the failure plane.

The factor of safety (F) is defined by

$$F = \frac{\text{Total force available to resist sliding}}{\text{Force tending to induce sliding}} \tag{4.1}$$

For the case shown in Figure 4.63 (drained slope) Equation (4.1) becomes

$$F = \frac{cA + W \cos \beta \tan \phi}{W \sin \beta} \tag{4.2}$$

If there is water present, then the factor of safety is expressed as

$$F = \frac{cA + (W \cos \beta - U) \tan \phi_a}{W \sin \beta + V} \tag{4.3}$$

where U is the uplift force along the base of the block due to water pressure, and V is the horizontal force along the face of the block due to water in the tension crack, ϕ_a is the friction angle (as affected by the water). Typical values for the cohesive strength and friction angles of soils and rock are given in Tables 4.3 and 4.4. As the height H of the slope increases the relative contribution of the cohesion to the total resistance decreases. For very high slopes, the stable slope angle approaches the friction angle ϕ. Hoek (1970a) has presented the relationship between slope height and slope angle functions for plane failure in a drained slope given in Figure 4.64.

Assume for example that the average planned slope angle i is 70°, the orientation of the potential failure plane β is 50° and the friction angle ϕ is 30°. Thus

$$X = 2\sqrt{(i - \beta)(\beta - \phi)} = 2\sqrt{20 \times 20} = 40°$$

From Figure 4.64 the slope height function Y is read as

$$Y = 14$$

Table 4.3. Cohesive strengths for 'intact' soil and rock (Robertson, 1971).

Material description	c (lb/ft^2)	c (kg/m^2)
Very soft soil	35	170
Soft soil	70	340
Firm soil	180	880
Stiff soil	450	2200
Very stiff soil	1600	7800
Very soft rock	3500	17,000
Soft rock	11,500	56,000
Hard rock	35,000	170,000
Very hard rock	115,000	560,000
Very very hard rock	230,000	1,000,000

Table 4.4. Friction angles (degrees) for typical rock materials (Hoek, 1970a).

Rock	Intact rock ϕ	Joint ϕ	Residual ϕ
Andesite	45	31–35	28–30
Basalt	48–50	47	
Chalk	35–41		
Diorite	53–55		
Granite	50–64		31–33
Graywacke	45–50		
Limestone	30–60		33–37
Monzonite	48–65		28–32
Porphyry		40	30–34
Quartzite	64	44	26–34
Sandstone	45–50	27–38	25–34
Schist	26–70		
Shale	45–64	37	27–32
Siltstone	50	43	
Slate	45–60		24–34
Other materials		**Approximate ϕ**	
Clay gouge (remoulded)		10–20	
Calcitic shear zone material		20–27	
Shale fault material		14–22	
Hard rock breccia		22–30	
Compacted hard rock aggregate		40	
Hard rock fill		38	

Knowing that

$$c = 1600\,\text{lb/ft}^2$$
$$\gamma = 160\,\text{lb/ft}^3$$

the limiting ($F = 1$) slope height H with such a structure passing through the toe is found using

$$Y = 14 = \frac{\gamma H}{c} = \frac{160}{1600}H$$

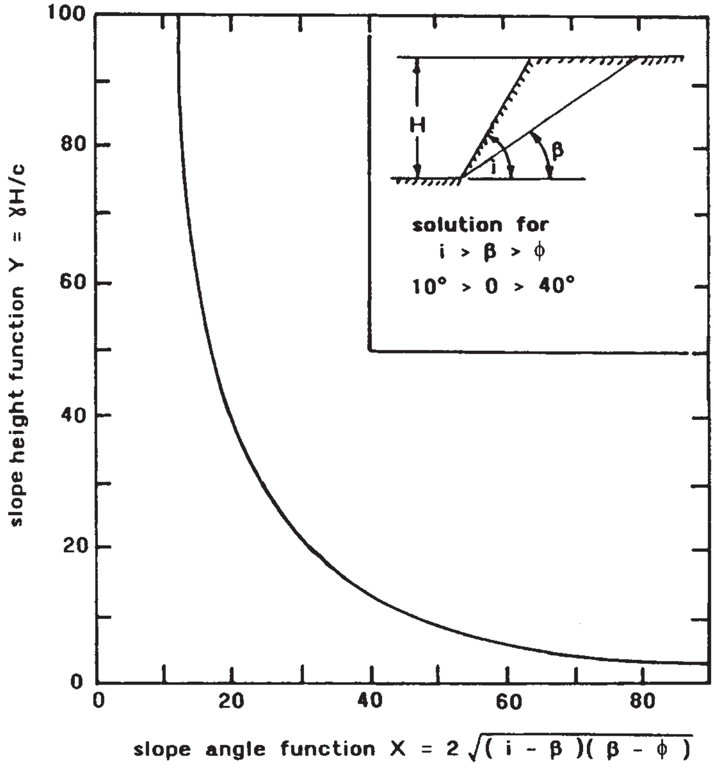


Figure 4.64. Relationship between slope height and slope angle functions for plane failure in a drained slope (Hoek, 1970a).

Thus

$$H = 140\,\text{ft}$$

If the planned pit depth is 500 ft, one could determine the limiting ($F = 1$) pit slope angle. The slope height function is

$$Y = \frac{\gamma H}{c} = \frac{160 \times 500}{1600} = 50$$

From Figure 4.64 one finds that

$$X = 17.5$$

Solving for i yields

$$i = 57.7^\circ$$

The general family of curves corresponding to various safety factors is given in Figure 4.65.

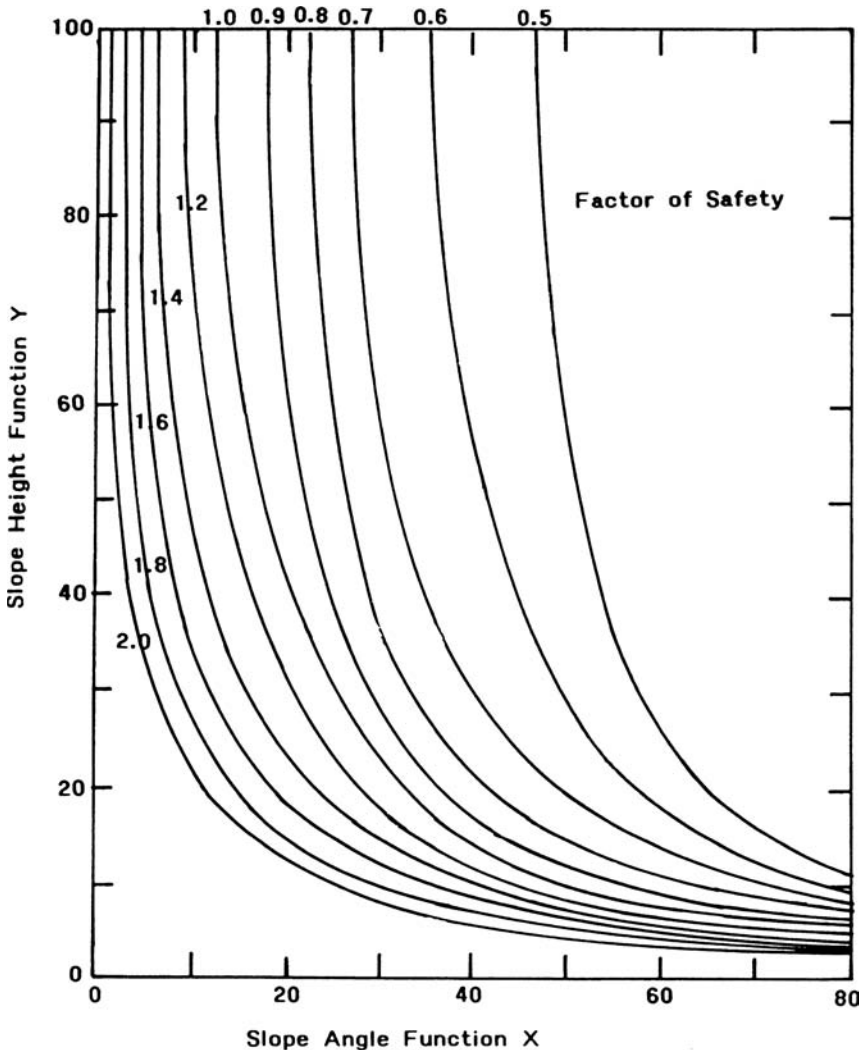


Figure 4.65. Slope design chart for plane failure including various safety factors (Hoek, 1970a).

The question naturally arises as to what an appropriate safety factor might be? This depends on the confidence one has in the 'goodness' of the input data and also on the function of the structure. Jennings & Black (1963) have provided the following advice:

For permanent structures, such as earth dams, F should not be less than 1.5 for the most critical potential failure surface, but for temporary constructions, where engineers are in continual attendance, a lower factor may be accepted. In civil engineering work, construction factors of safety are seldom allowed to be less than 1.30. An open pit is a 'construction' of a very particular type and it is possible that a factor of 1.20–1.30 may be acceptable in this case.

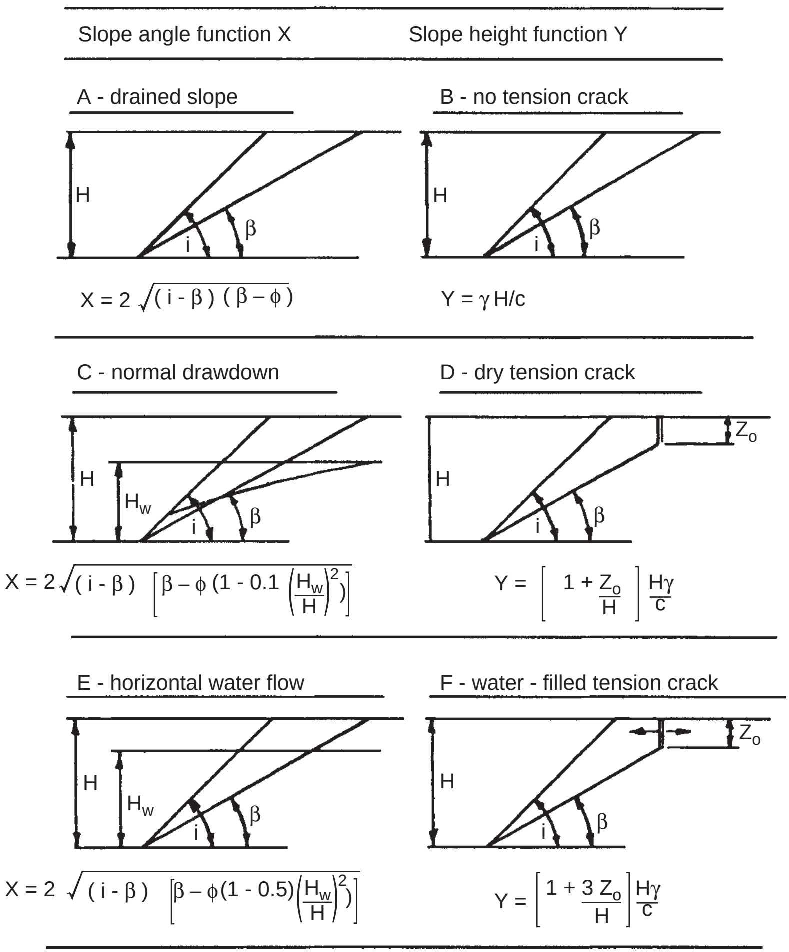


Figure 4.66. Slope angle and slope height functions for different water and tension crack conditions (Hoek, 1970a).

The confidence placed in any value calculated as the factor of safety of a slope depends upon the accuracy with which the various factors involved can be estimated. The critical items are the selection of the most adverse surface for potential failure, the measurement of the shear strength of the materials on this surface and the estimation of the water pressures in the soil pores and in any fissures along the surface.

If one were to select a safety factor of 1.2 for the previous example, one finds that for $Y = 50$, $X = 13.5$. The slope angle becomes

$$i = 54.6^\circ$$

The example applies for the very special case of a drained slope without a tension crack. Often a tension crack will be present and there can be a variety of different slope water conditions. Hoek (1970a) has developed a simple way of handling these. Figure 4.66 provides three different expressions for X corresponding to different slope water conditions

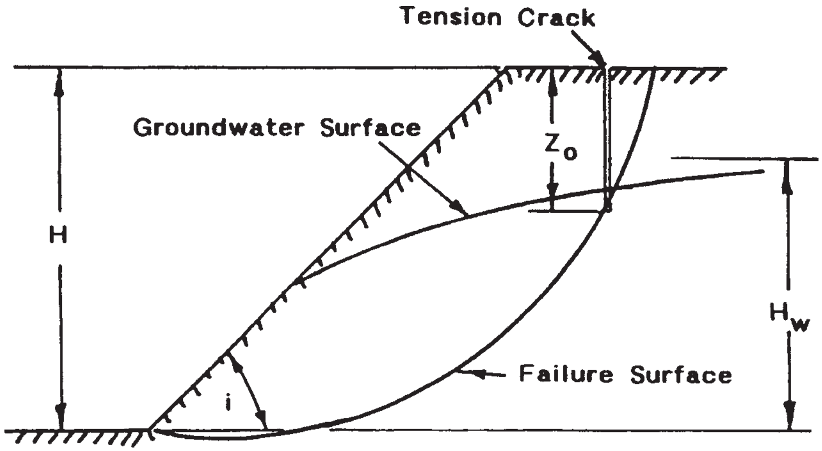


Figure 4.67. Diagrammatic representation of circular failure in a slope (Hoek, 1970a).

and three different expression for *Y* relating to the tension crack. Thus nine different *X*–*Y* combinations are possible. The one used in the earlier examples was combination *A*–*B*. From Figure 4.66 one finds the *X*–*Y* combination most appropriate to the problem at hand.

The known values are substituted and Figure 4.65 is used to determine the desired missing value. The interested reader is encouraged to evaluate the effect of different slope water conditions on the slope angle.

4.6.4 *Circular failure*

Hoek (1970a) has applied the same approach to the analysis of circular failures (Fig. 4.67).

Such deep seated failures occur when a slope is excavated in soil or soft rock in which the mechanical properties are not dominated by clearly defined structural features. This type of failure is important when considering the stability of:

– very high slopes in rock in which the structural features are assumed to be randomly oriented,
– benches or haul road cuts in soil,
– slimes dams,
– waste dumps.

Figure 4.68 gives the relationship between the slope height function and slope angle function for circular failure in drained slopes without a tension crack ($F = 1$). The corresponding chart, including different safety factor values, is given in Figure 4.69. To accommodate different tension crack and slope water conditions, Figure 4.70 has been developed. This set of curves is used in exactly the same way as described earlier.

4.6.5 *Stability of curved wall sections*

The approaches discussed to this point have applied to pit wall sections which can be approximated by two-dimensional slices. Open pits often take the form of inverted cones or have portions containing both convex and concave wall portions (Figure 4.71).

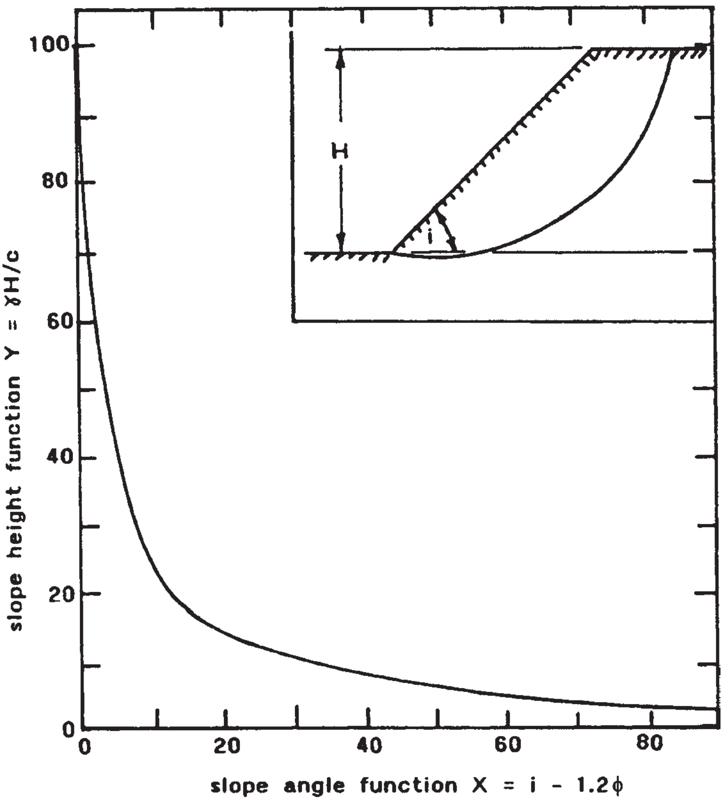


Figure 4.68. Plot of slope height versus slope angle functions for circular failure analysis (Hoek, 1970a).

Very little quantitative information on the effect of pit wall curvature on stability is available from the literature. Convex portions of a pit wall (noses which stick out into the pit) frequently suffer from unstable slopes. The relaxation of lateral stresses give rise to a reduction in the normal stress across potential failure planes and vertical joint systems can open. For concave portions of the pit, the arch shape of the slope tends to induce compressive lateral stresses which increase the normal stress across potential failure planes. The slopes are more stable due to the increased frictional resistance.

Hoek (1970a) suggests that curvature of the slope in plan can result in critical slope differences of approximately 5° from that suggested by the planar analyses. A concave slope, where the horizontal radius of curvature is of the same order of magnitude as the slope height, may have a stable slope angle 5° steeper than for a straight wall (infinite radius of curvature). On the other hand, a convex slope may require flattening by about 5° in order to improve its stability.

However, improved drainage in the convex slopes over that available with the pinched concave shape may provide a stability advantage. Thus, there may be some cancelling of advantages/disadvantages. Hence, each pit curvature situation must be carefully examined.

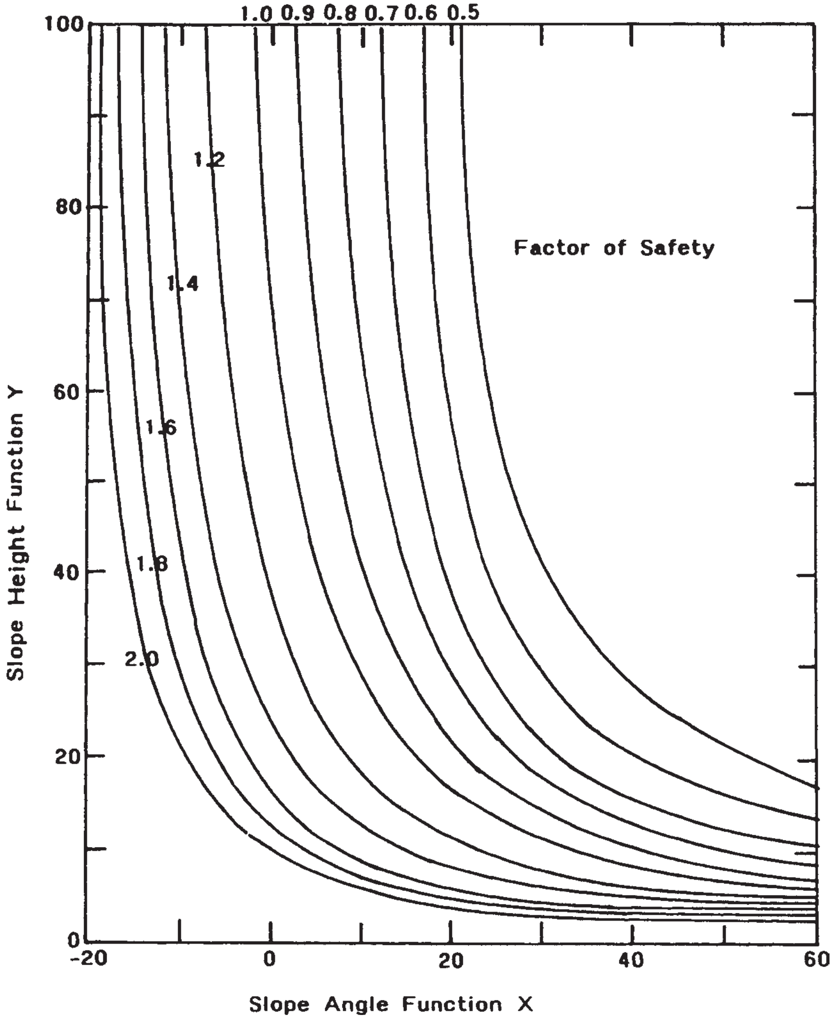


Figure 4.69. Slope design chart for circular failure including various safety factors (Hoek, 1970a).

4.6.6 *Slope stability data presentation*

Figure 4.72 developed by Hoek & Bray (1977) is a good example of how structural geology information and preliminary evaluation of slope stability of a proposed open pit mine can be presented. A contour plan of the proposed open pit mine is developed and contoured stereoplots of available structural data are superimposed. In this particular case two distinct structural regions denoted by A and B have been identified and marked on the plan. Based simply on geometry (of the pit slopes and structures), the potential failure types are identified. Each of these would then be examined using appropriate material properties and ground water conditions. Required design changes, additional data collection, etc. will emerge.

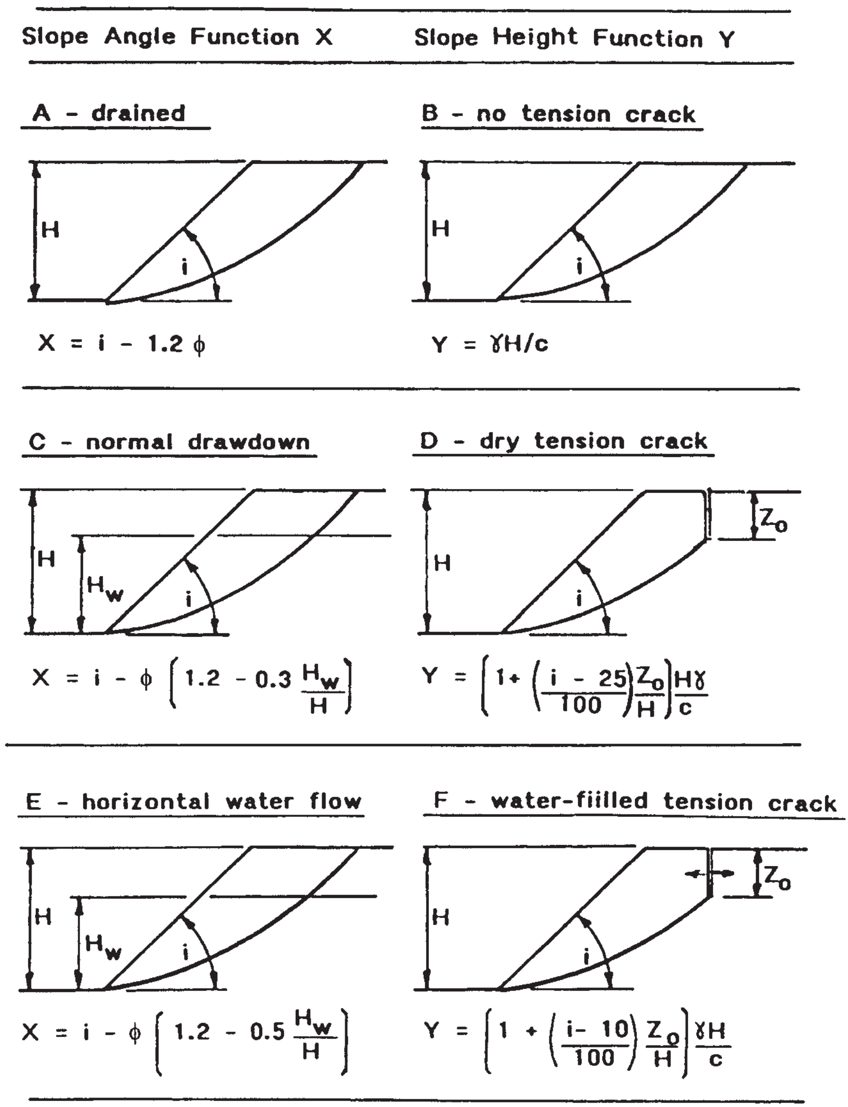


Figure 4.70. Slope angle and slope height functions for different water and tension crack conditions (Hoek, 1970a).

4.6.7 *Slope analysis example*

Reed (1983) has reported the results of applying the Hoek & Bray (1977) approach to the Afton copper-gold mine located in the southern interior of British Columbia. For the purpose of analyzing the stability of the walls of the open pit, it was divided into 9 structural domains (Fig. 4.73).

For each structural domain, a stability analysis was made of:

– the relative frequency of the various fault and bedding plane orientations, and
– the orientation of the pit wall in that particular domain.

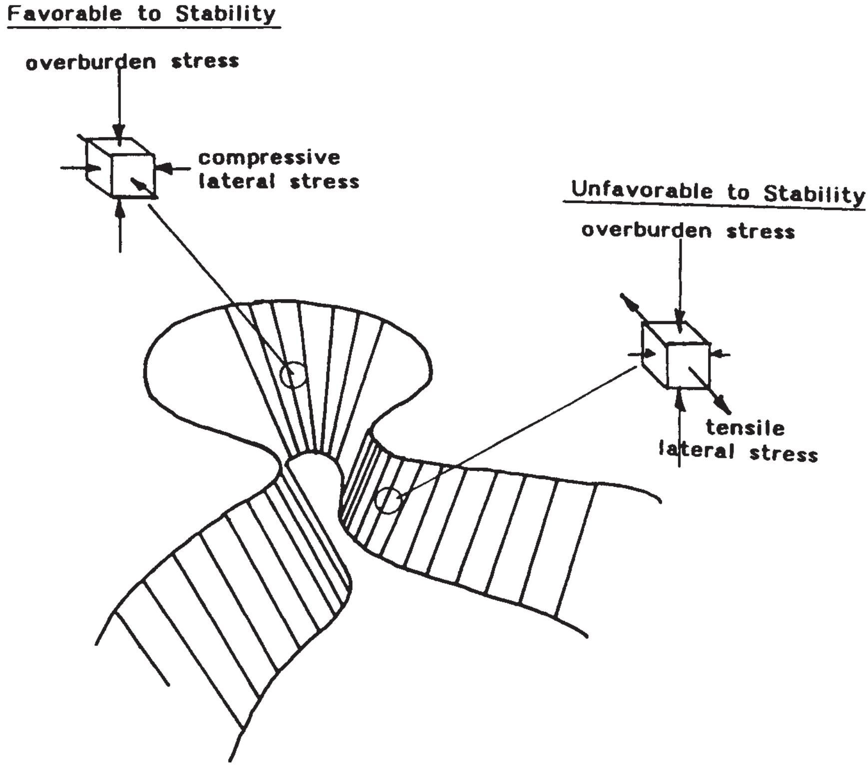


Figure 4.71. Influence of three-dimensional pit shape upon slope stability (Hoek, 1970a).

The safety factors were calculated for plane failure, wedge failure and circular failure in each domain. Table 4.5 shows the results of these stability analyses.

The 'maximum safe slope angle' for the pit wall in each domain corresponds to a calculated safety factor of 1.2. The results in Table 4.5 predict wall failure in all domains if the slopes are wet. The mine, however, lies in a semi-arid area and expected ground water quantities were small. In addition, horizontal drainholes would be used to reduce ground water pressures in domains 3 and 6. Problems would still be expected in domains 3 and 6. Since domain 3 is a relatively narrow domain and the probability of a major slide occurring was small, the design slope of the wall in that area was not flattened. At the time the paper was written (1983), the pit had reached a depth of 480 ft. Two failures had been experienced in domain 3 and several berm failures in domain 6. There was no indication of impending major failures. Final pit depth was planned to be 800 ft.

4.6.8 *Economic aspects of final slope angles*

Figure 4.74 illustrates the volume contained in a conical pit as a function of final slope angle and depth.

For a depth of 500 ft and an overall final pit angle of 45°, 1.4×10^7 tons of rock must be moved. Within the range of possible slopes (20° to 70°) at this depth the volume to be

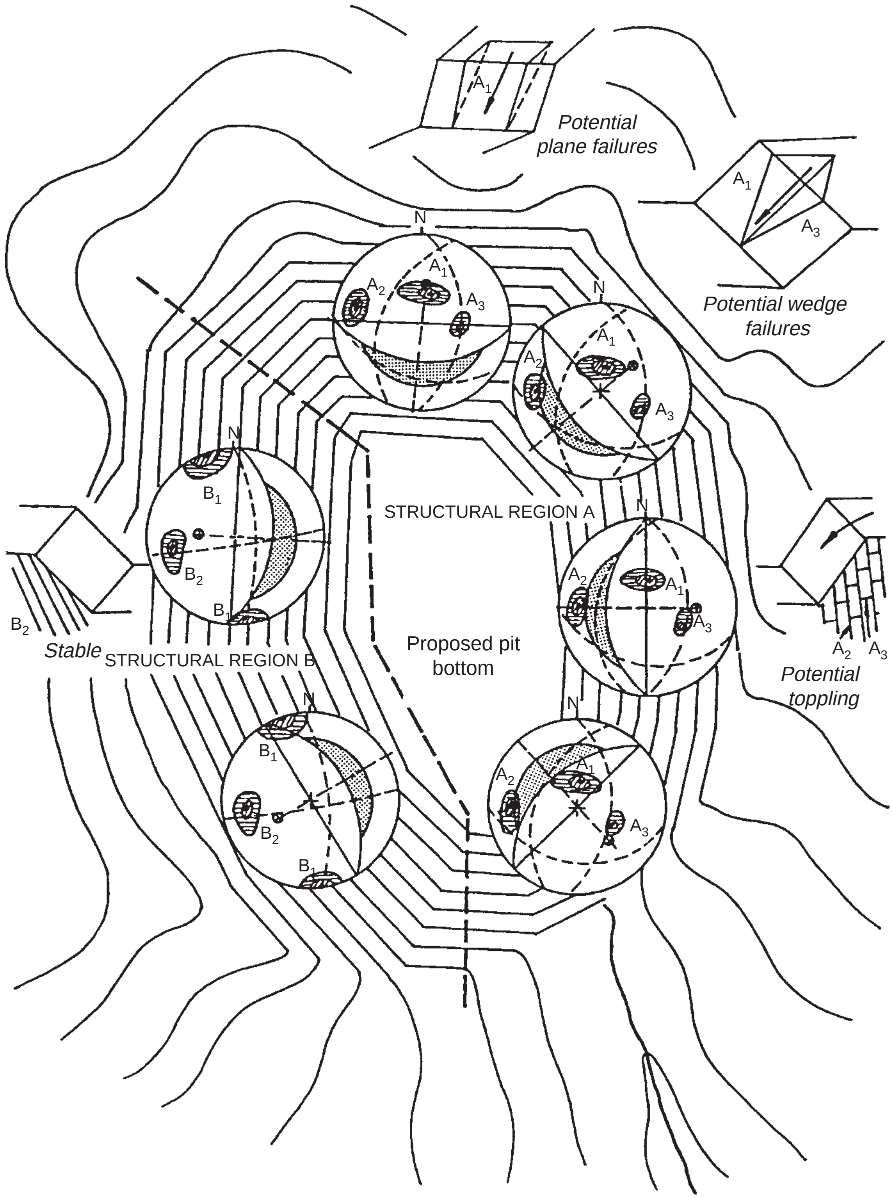


Figure 4.72. Presentation of structural geology information and preliminary evaluation of slope stability of a proposed open pit mine (Hoek & Bray, 1977).

moved approximately doubles for every 10° flattening of the slope. Flattening the slope of the 500 ft deep conical pit from 50° to 40° increases the mass of rock from 1.0×10^7 to 2.0×10^7 tons. This simple example shows that the selection of a particular slope can have a significant impact on the scale of operations and depending upon the shape, size and grade of the ore contained within the pit, on the overall economics.

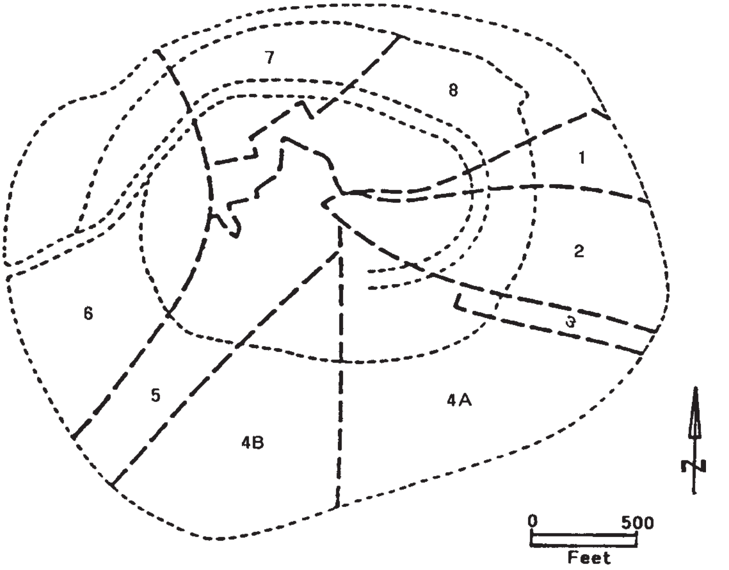


Figure 4.73. Division of the Afton open pit into 9 structural domains (Reed, 1983).

Table 4.5. Calculated and design slope angles for the Afton mine (Reed, 1983).

Domain	Maximum safe slope angle		Design slope angle
	Wet	Dry	
1	24°	54°	45°
2	52°	52°	45°
3	24°	41°	45°
4A	27°	49°	45°
4B	45°	42°	45°
5	22°	42°	45°
6	28°	39°	40°
7	33°	42°	40°
8	32°	43°	40°

4.7 PLAN REPRESENTATION OF BENCH GEOMETRY

Figure 4.75 is a cross-sectional representation of an open pit mine. Figure 4.76 is a 'birds-eye' (plan) view of the same pit. No attempt has been made to distinguish between the toes and crests (which are marked in Fig. 4.77) and hence the figure is difficult to interpret.

Several different techniques are used by the various mines to assist in plan representation and visualization. In Figure 4.78 the bank slopes have been shaded and the benches labelled with their elevations.

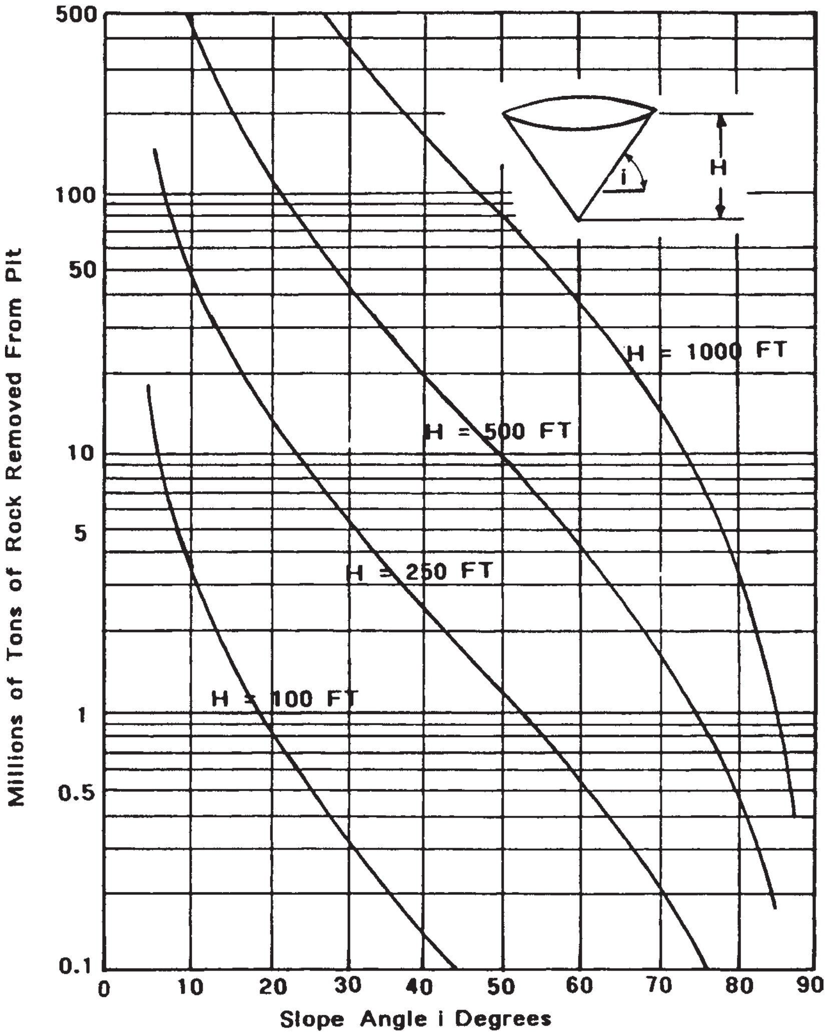


Figure 4.74. Influence of pit depth and slope angle on the amount of rock removed in mining a conical open pit (Hoek & Pentz, 1970).

Figure 4.79 is an example of this type of representation for an actual mine. An alternative is to draw the crests with solid lines and the toes with dashed lines. The result is shown in diagrammatic form in Figure 4.80 and for an actual property in Figure 4.81. Note that the banks have also been shaded. This is however seldom done. This system of identifying toes and crests is recommended by the authors.

Some companies use the opposite system labelling the crests with dashed lines and the toes by solid lines (Fig. 4.82). The Berkeley pit shown in Figure 4.83 is one such example where this system has been applied.

If there are a great number of benches and the scale is large, there can be difficulties in representing both the toes and the crests. Knowing the bench height and the bench face angle

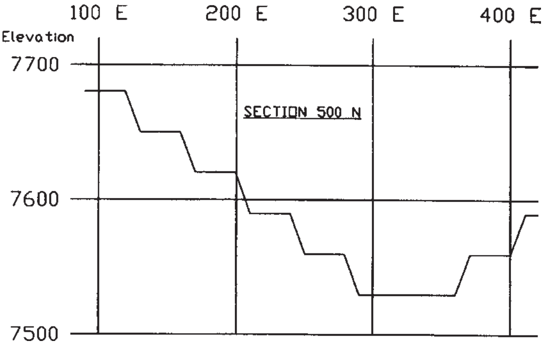


Figure 4.75. Cross-section through an open pit mine.

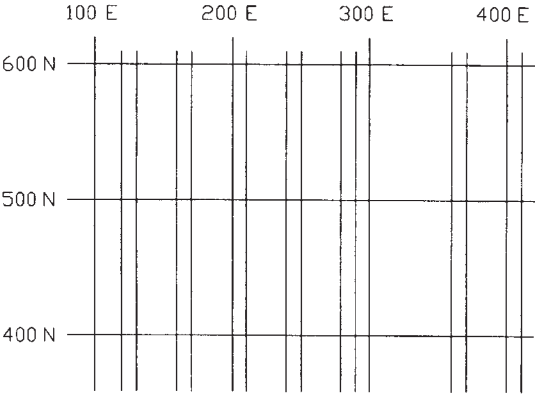


Figure 4.76. Plan view through the portion of the pit shown in cross-section in Figure 4.75 (toes and crests depicted by solid lines).

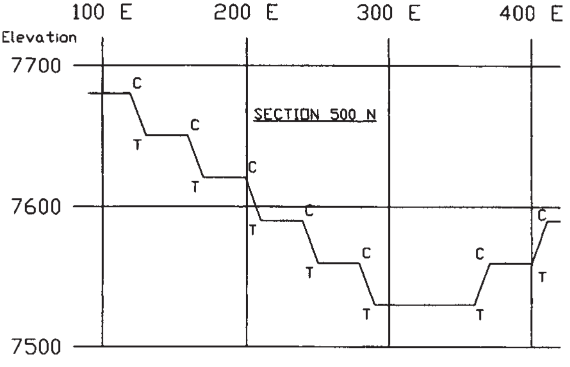


Figure 4.77. Cross-section through a portion of an open pit with toes and crests labelled.

it is a simple matter to construct, if needed, the toes presuming that the crests are given or vice versa. Hence only one set of lines (crests or toes) is actually needed. When only one line is used to represent a bench, the most common technique is to draw the median (mid bench) elevation line at its plan location on the bench face. This is shown in section and plan in Figures 4.84 and 4.85, respectively.

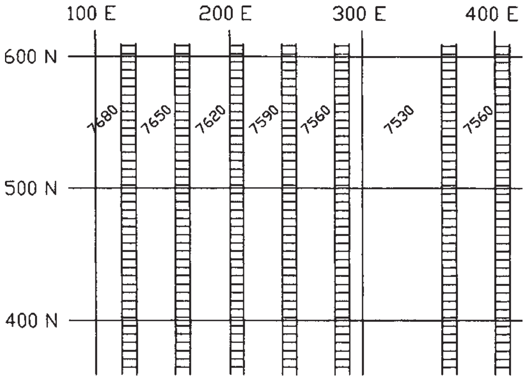


Figure 4.78. Plan view with the bench faces shaded and the flat segment elevations labelled.

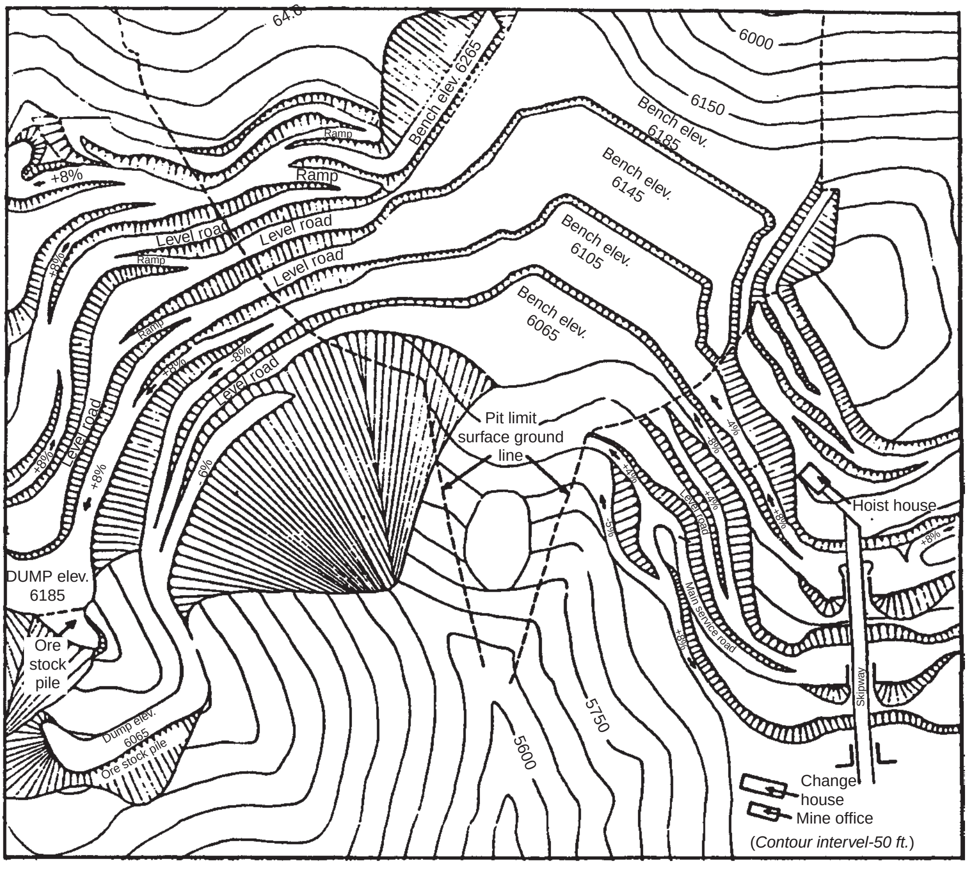


Figure 4.79. Example of slope surface shading described in Figure 4.78 (Ramsey, 1944).

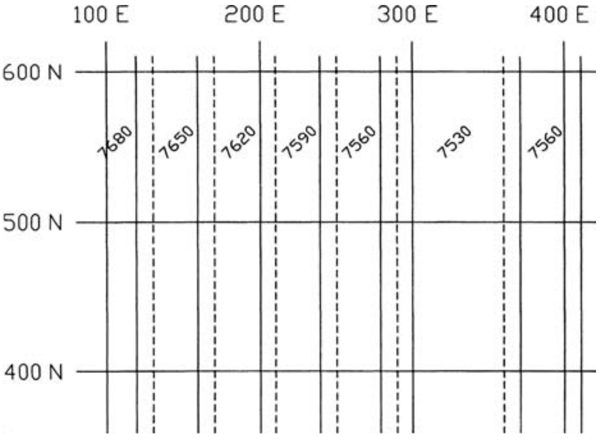


Figure 4.80. Plan view of a portion of the open pit (crests are denoted by solid lines and toes by dashed lines).

An actual example of its use is given in Figure 4.86. An enlarged view of a section of the pit is shown in Figure 4.87. The elevation label is located half way between the median contour lines. This is the actual location for this elevation and corresponds to the bench elevation at that point.

It is a relatively simple matter to go from median lines to actual bench representation (toes and crests) and vice versa. This process is depicted in Figure 4.88. The median contour line in the center will be replaced by the toe-crest equivalent. The road is 100 ft wide and has a grade of 10%. The bench height is 40 ft, the bank width is 30 ft and the width of the safety bench is 50 ft. The process begins by adding the center lines halfway to the next contour lines (Fig. 4.88b). Toe and crest lines are added (Fig. 4.88c) and the edge of road is drawn (Fig. 4.88d). Finally the construction lines are removed (Fig. 4.88e). The reader is encouraged to try this construction going back and forth from toes and crests to median lines.

4.8 ADDITION OF A ROAD

4.8.1 *Introduction*

Roads are one of the more important aspects of open pit planning. Their presence should be included early in the planning process since they can significantly affect the slope angles and the slope angles chosen have a significant effect on the reserves. Most of the currently available computerized pit generating techniques discussed in the following chapter do not easily accommodate the inclusion of roads. The overall slope angles without the roads may be used in the preliminary designs. Their later introduction can mean large amounts of unplanned stripping or the sterilization of some planned reserves. On the other hand a flatter slope angle can be used which includes the road. This may be overly conservative and include more waste than necessary.

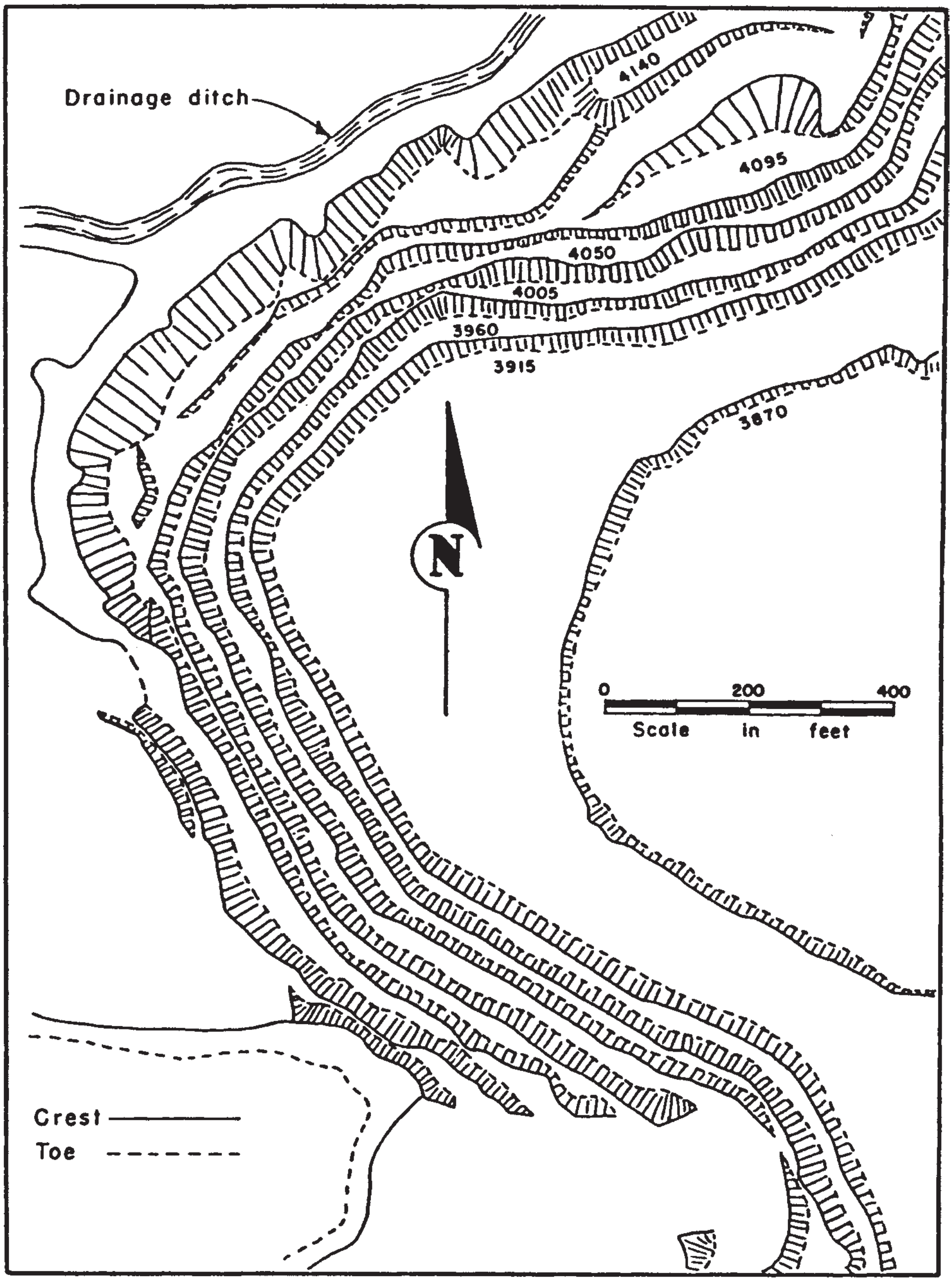


Figure 4.81. Example of the mapping procedure described in Figure 4.80 (Hardwick & Stover, 1960).

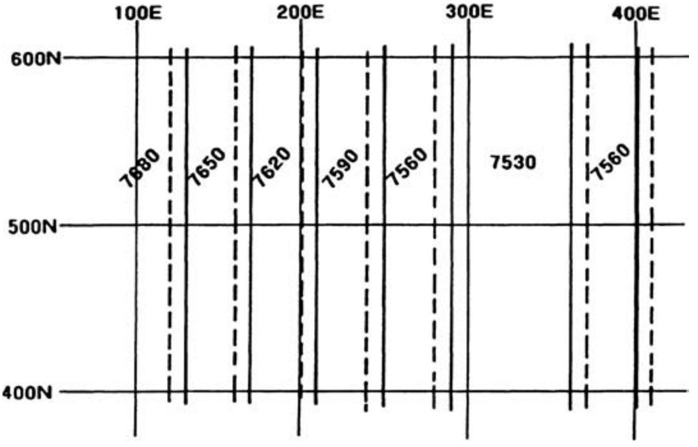


Figure 4.82. Plan view of a portion of the open pit (crests denoted by dashed lines and toes by solid lines).

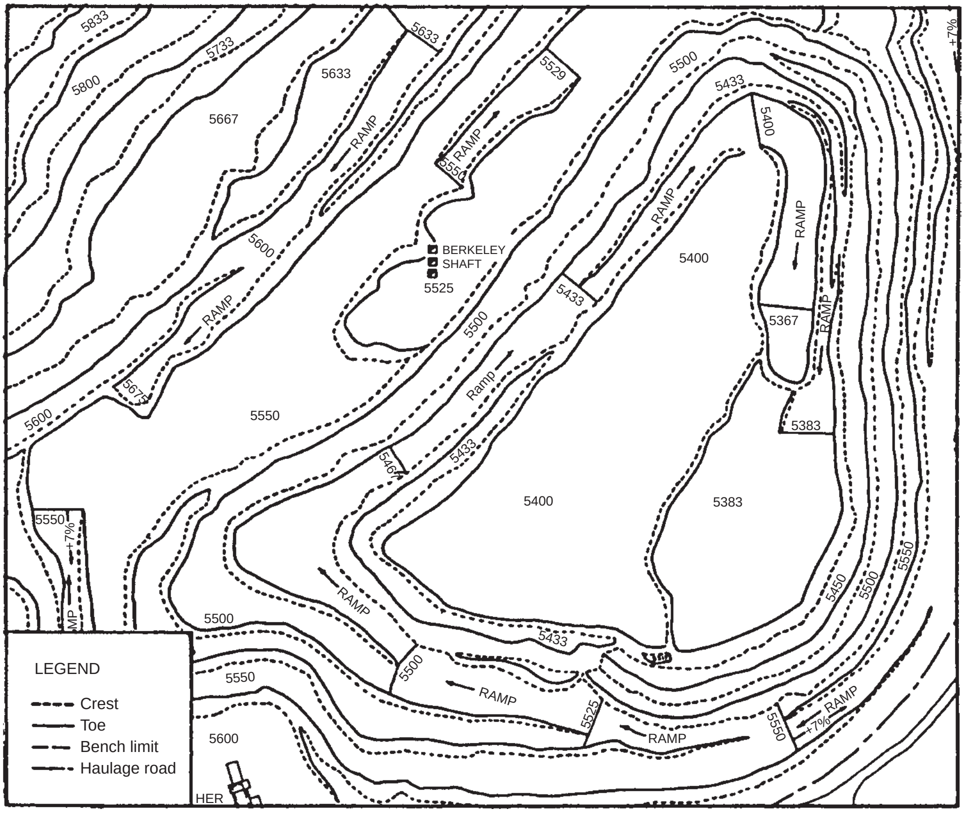


Figure 4.83. Example of the mapping procedure described in Figure 4.82 (McWilliams, 1959).

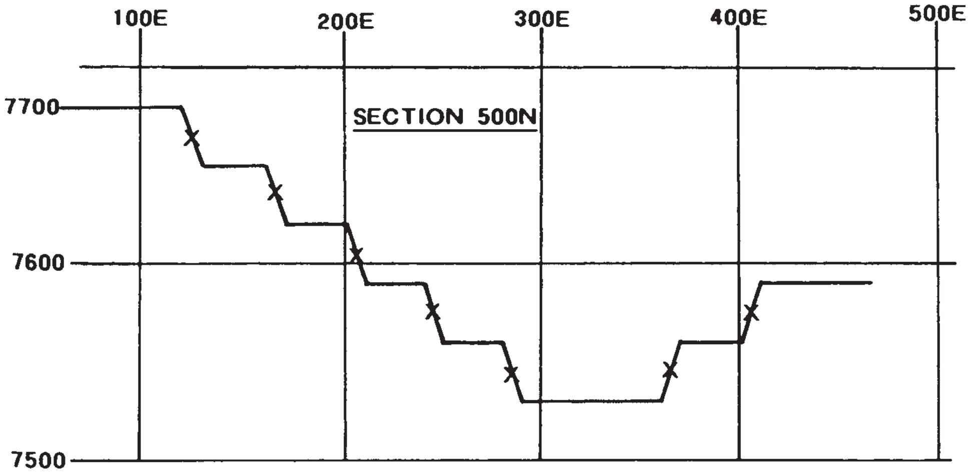


Figure 4.84. Procedure of denoting the median midbench elevation line on the bench face.

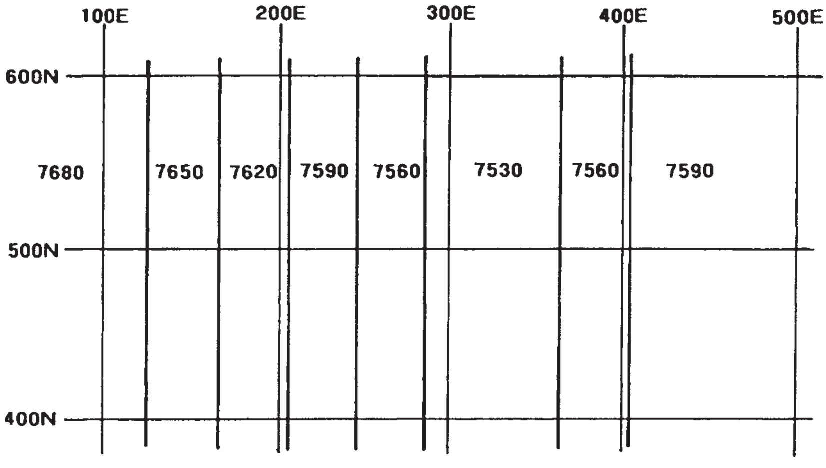


Figure 4.85. Resulting plan view corresponding to the midbench representation of Figure 4.84. The given elevations are bench toe elevations.

Until rather recently, rail haulage was a major factor in open pit operations. Because of the difficulties with sharp turns and steep grades, a great deal of time was spent by mine planners in dealing with track layout and design. Rubber tired haulage equipment has presented great flexibility and ability to overcome many difficulties resulting from inadequate or poor planning in today's pits. However as pits become deeper and the pressure for cost cutting continues, this often neglected area will once again be in focus.

There are a number of important questions which must be answered when siting the roads (Couzens, 1979).

1. The first decision to be taken is where the road exit or exits from the pit wall will be. This is dependent upon the crusher location and the dump points.

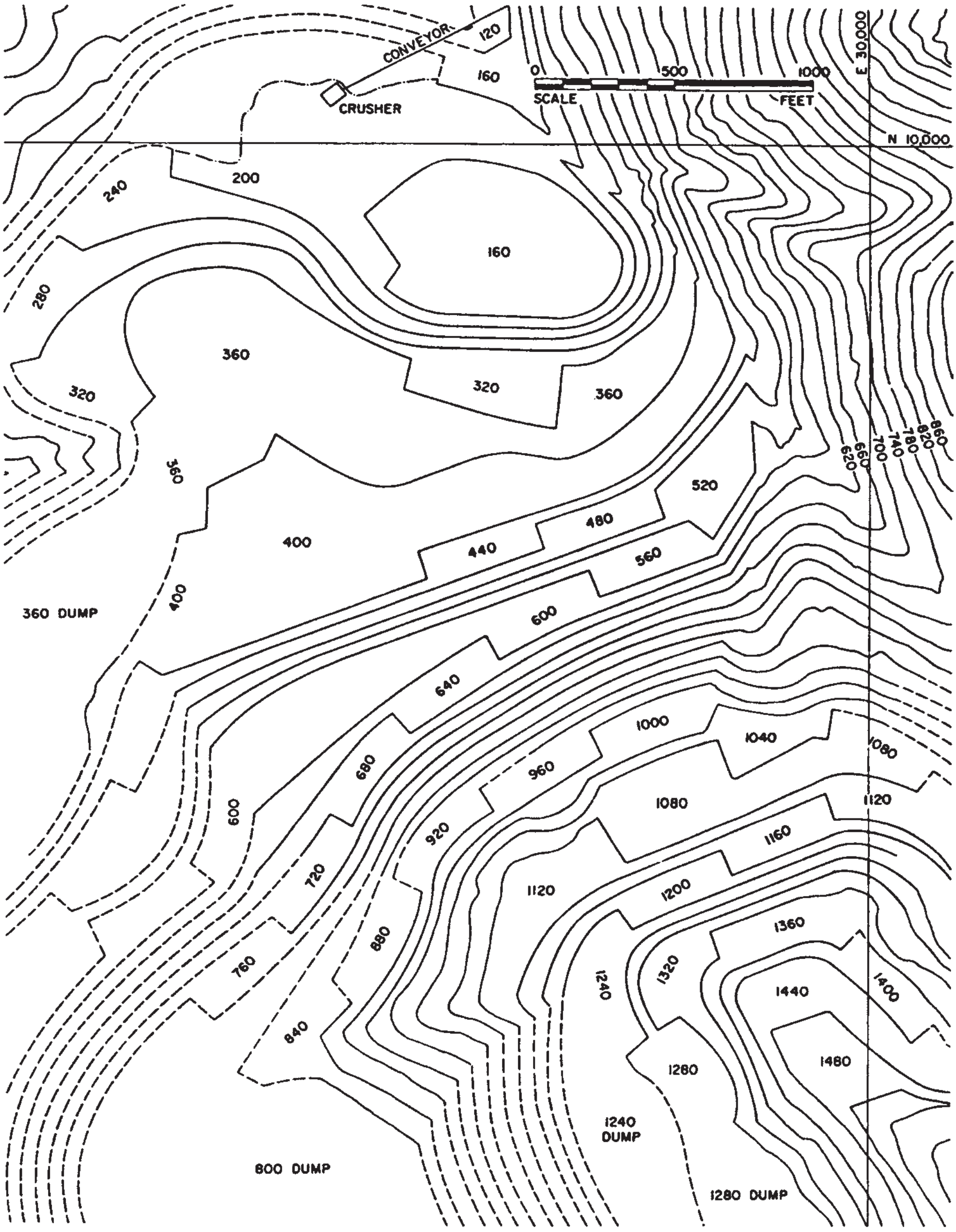


Figure 4.86. Example mining plan composite map based on midbench contours (Couzens, 1979).

2. Should there be more than one means of access? This allows certain flexibility of operation but the cost of added stripping can be high.

3. Should the roads be external or internal to the pit? Should they be temporary or semi-permanent?

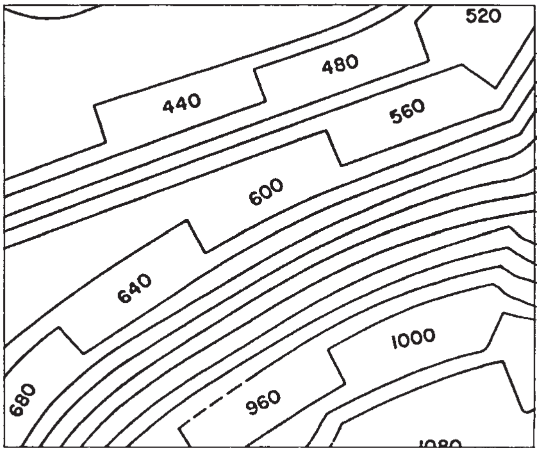


Figure 4.87. An enlarged portion of Figure 4.86 (Couzens, 1979).

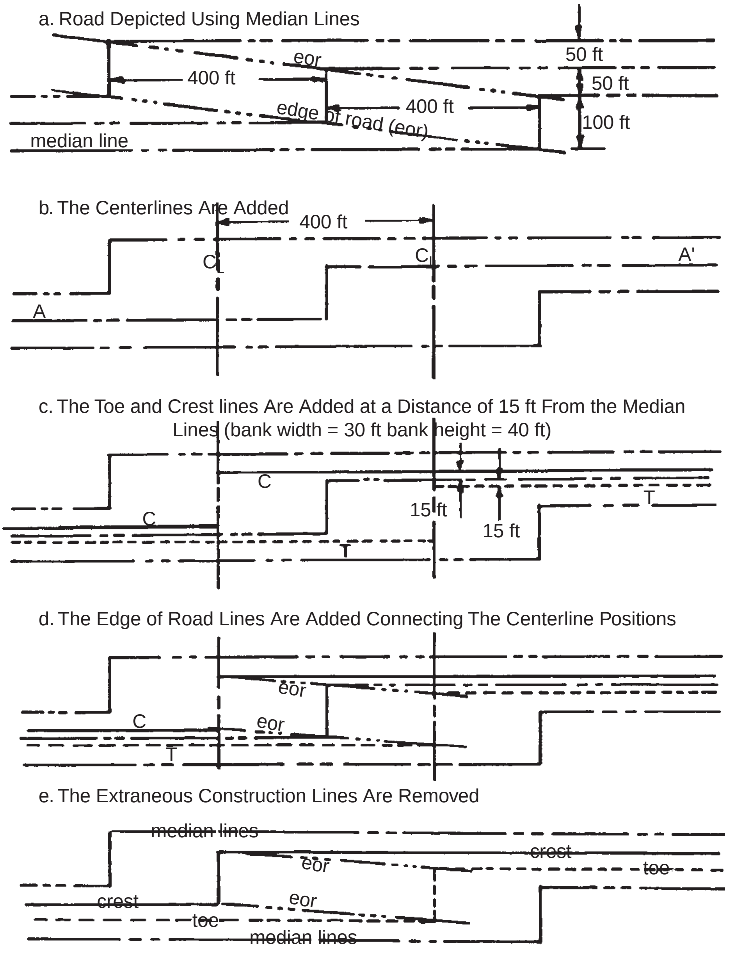


Figure 4.88. Going from midbench contours to a toe and crest representation.

4. Should the road spiral around the pit? Have switchbacks on one side? Or a combination?

5. How many lanes should the road have? The general rule of thumb for 2 way traffic is: road width $\geq 4 \times$ truck width. Adding an extra lane to allow passing may speed up the traffic and therefore productivity but at an increased stripping cost.

6. What should the road grade be? A number of pits operate at 10% both favorable and unfavorable to the haul. A grade of 8% is preferable since it provides more latitude in building the road and fitting bench entries. That is, providing it does not cause too much extra stripping or unduly complicate the layout.

7. What should be the direction of the traffic flow? Right hand or left hand traffic in the pit?

8. Is trolley assist for the trucks a viable consideration? How does this influence the layout?

This section will not try to answer these questions. The focus will be on the procedure through which haulroad segments can be added to pit designs. The procedures can be done by hand or with computer assist. Once the roads have been added then various equipment performance simulators can be applied to the design for evaluating various options.

4.8.2 *Design of a spiral road – inside the wall*

As has been discussed in Section 4.3, the addition of a road to the pit involves moving the wall either into the pit and therefore losing some material (generally ore) or outward and thereby adding some material (generally waste). This design example considers the first case (inside the original pit wall). The second case will be discussed in the following section. This pit consists of the four benches whose crests are shown in Figure 4.89. Both toes and crests are shown in Figure 4.90. The crest-crest dimension is 60 ft, the bench height is 30 ft

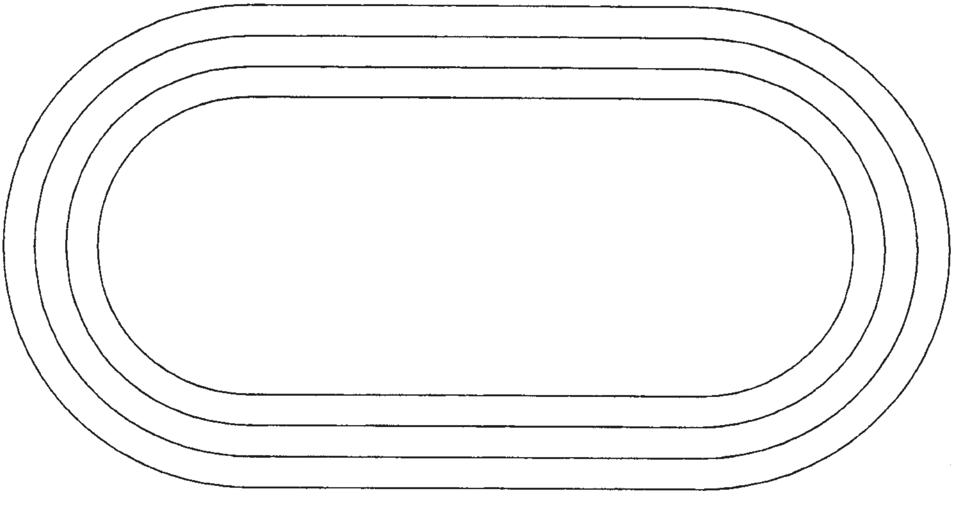


Figure 4.89. The four bench pit with crests shown.

and a road having a width of 90 ft and a grade of 10% is to be added to the north wall. The bench face has an angle of 56°.

Step 1. The design of this type of road begins at the pit bottom. For reasons to be discussed later, the point where the ramp meets the first crest line is selected with some care. In this case, the ramp will continue down to lower mining levels along the north and east walls, thus point A in Figure 4.91 has been selected.

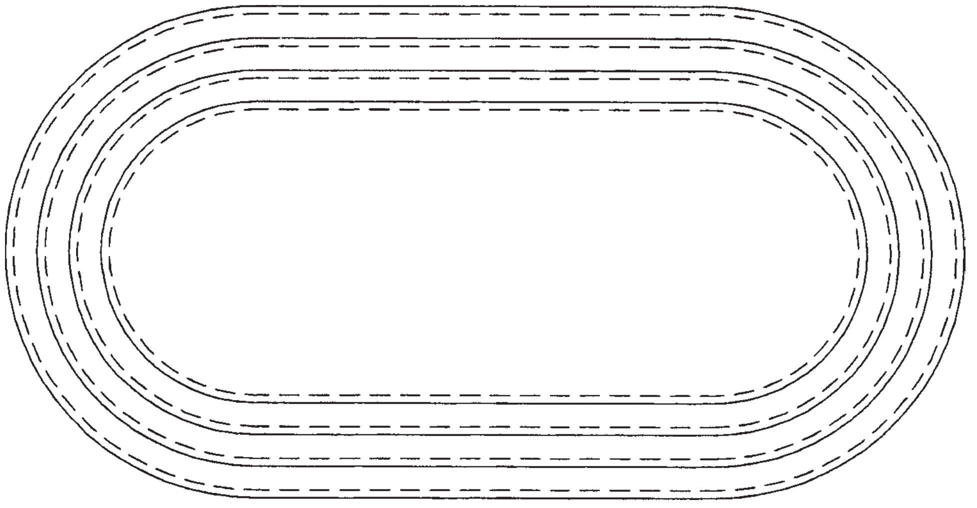


Figure 4.90. The four bench pit with toes added.

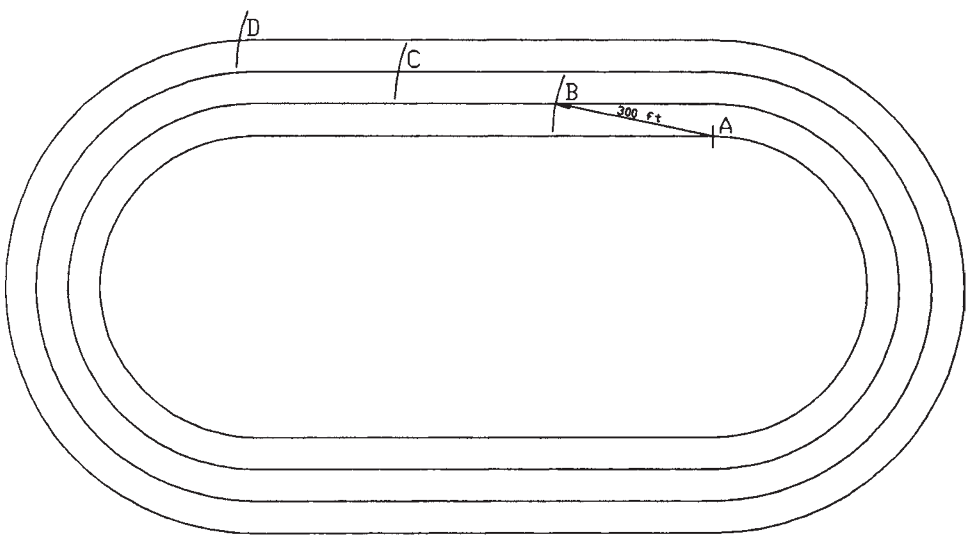


Figure 4.91. Point of ramp initiation and crest intercepts.

Step 2. The locations where the ramp meets the succeeding crests are now determined. Since the bench height H is 30 ft and the road grade G is 10%, the horizontal distance D travelled by a truck going up to the next level is

$$D = \frac{100H}{G(\%)} = \frac{100 \times 30}{10} = 300\,\text{ft}$$

Point B on the crest of the next bench is located by measuring the 300 ft distance with a ruler or by swinging the appropriate arc with a compass. Points C and D are located in a similar way.

Step 3. The crest line segments indicating the road location will be added at right angles to the crest lines rather than at right angles to the line of the road. Hence they have a length (W_a) which is longer than the true road width (W_t). As can be seen in Figure 4.92, the angle (Θ) that the road makes with the crest lines is

$$\Theta = \sin^{-1}\frac{600}{300} = 11.5°$$

Hence the apparent road width W_a (that which is laid out), is related to the true road width by

$$W_a = \frac{W_t}{\cos\Theta} = 1.02\,W_t = 1.02 \times 90 = 92\,\text{ft}$$

For most practical purposes, little error results from using

$$W_a \cong W_t = W$$

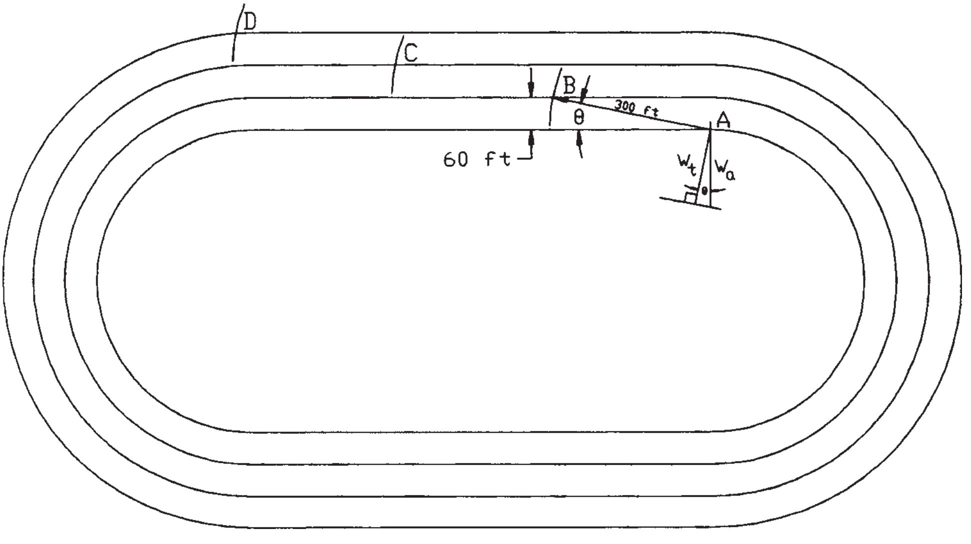


Figure 4.92. Addition of ramp width (Step 3).

Lines of length *W* drawn perpendicular to the crest lines from points A, B, C and D have been added to Figure 4.93a. In addition short lines running parallel to the crest starting at the ends of these lines have been added. Line *a-a′* is one such line.

Step 4. Line *a-a′* is extended towards the west end of the pit. It first runs parallel to the previous crest line but as the pit end approaches it is curved to make a smooth transition with the original crest line. This is shown in Figure 4.93b. The designer has some flexibility

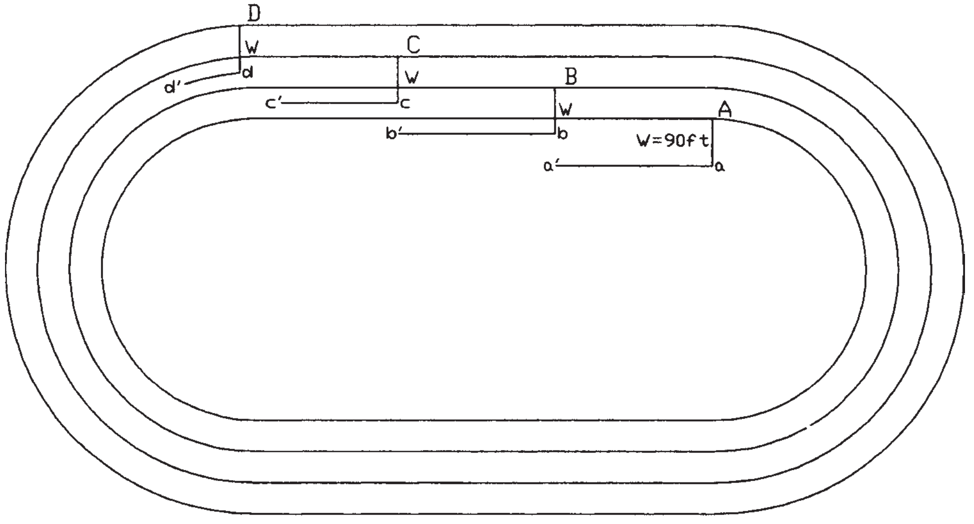


Figure 4.93a. Completing the new crest lines (Step 4).

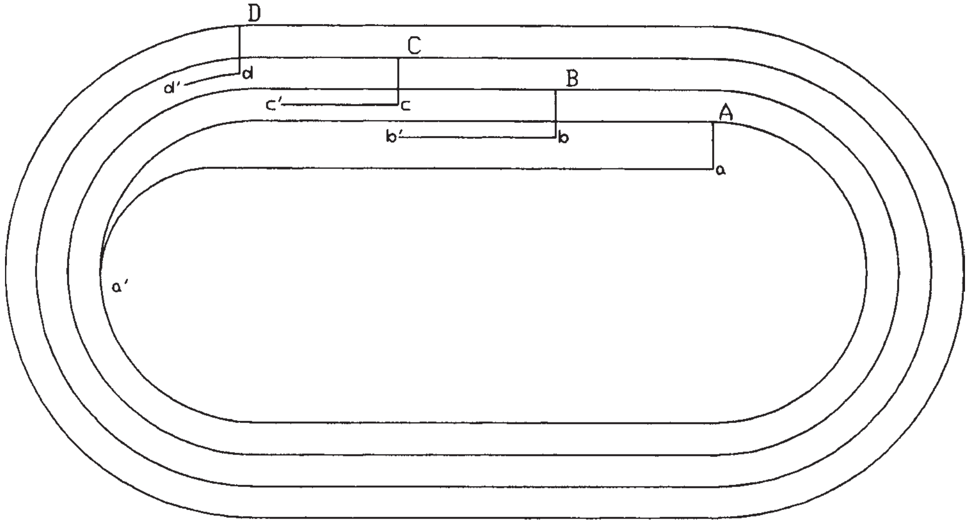


Figure 4.93b. Completing the new crest lines (Step 4).

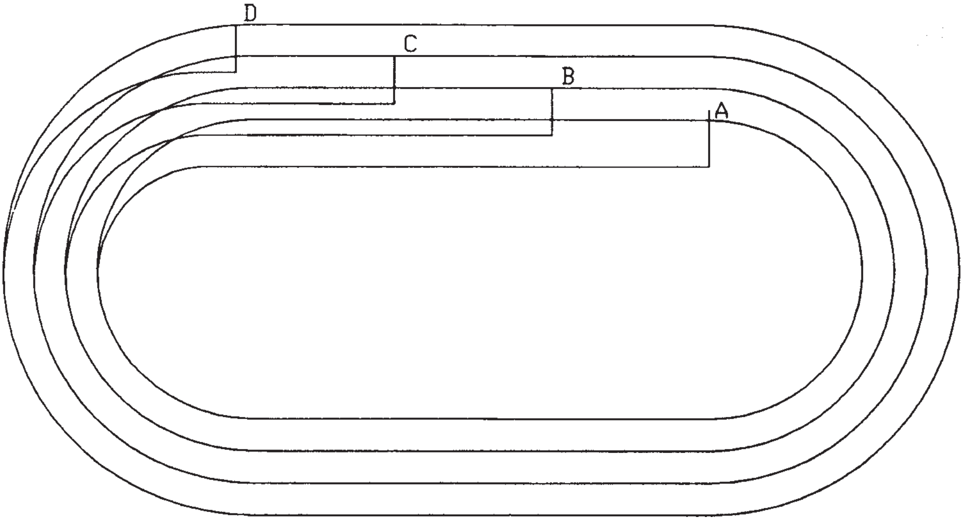


Figure 4.93c. Completing the new crest lines (Step 4).

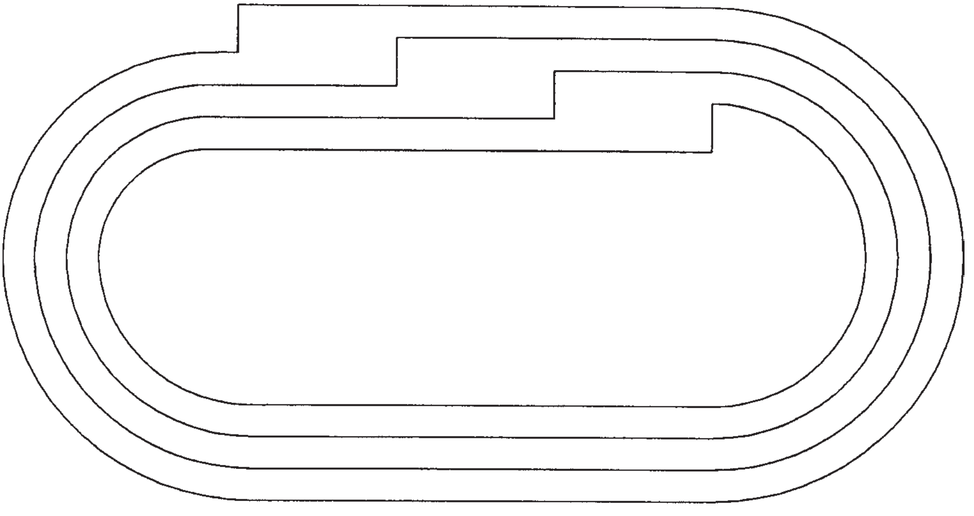


Figure 4.94. The pit as modified by the ramp (Step 5).

on how this transition occurs. Once this decision is made then the remaining crest lines are drawn parallel to this first one. The results are shown in Figure 4.93c.

Step 5. The extraneous lines remaining from the original design are now removed. The resulting crest lines with the included ramp are shown in Figure 4.94.

Step 6. The ramp is extended from the crest of the lowest bench to the pit bottom. This is shown in Figure 4.95. The toe lines have been added to assist in this process. In Figure 4.95,

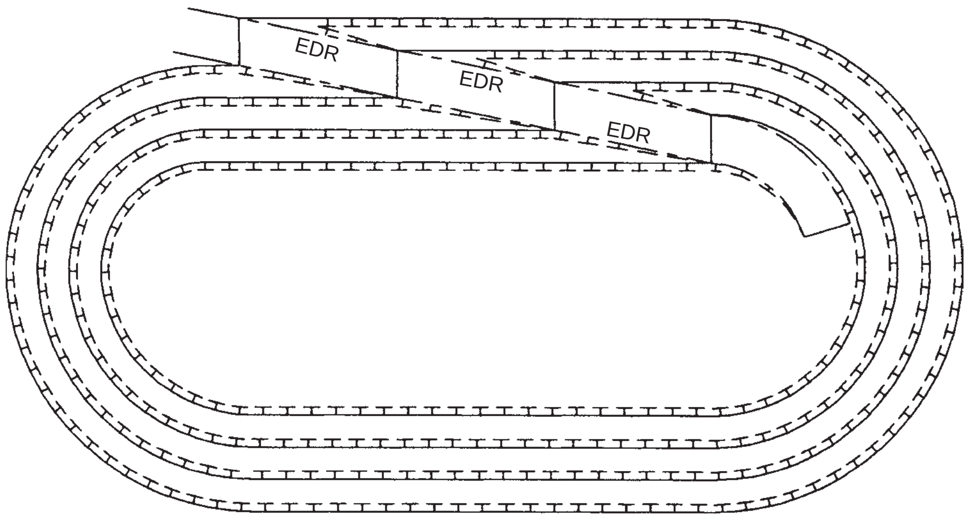


Figure 4.95. Addition of entrance ramp and toe lines (Step 6).

the slopes have been shaded to help in the visualization. The edge of road (EOR) lines shown are also crest lines.

4.8.3 *Design of a spiral ramp – outside the wall*

In the previous section the addition of a spiral ramp lying inside the original pit contours was described. It's addition meant that some material initially scheduled for mining would be left in the pit. For the case described in this section where the ramp is added outside the initial pit shell design, additional material must be removed. The same four bench mine as described earlier will be used:

Bench height = 30 ft
Crest-crest distance = 60 ft
Road width = 90 ft
Road grade = 10%
Bench slope angle = 56°

Step 1. The design process begins with the crest of the uppermost bench. A decision must be made regarding the entrance point for the ramp as well as direction. As shown in Figure 4.96, the entrance should be at point A in the direction shown. Mill and dump locations are prime factors in selecting the ramp entrance point. From this point an arc of length L equal to the plan projection of the ramp length between benches is struck. This locates point B. From point B an arc of length L is struck locating point C, etc.

Step 2. From each of the intersection points A, B, C and D, lines of length W_a (apparent road width) are constructed normal to their respective crest lines. This is shown in

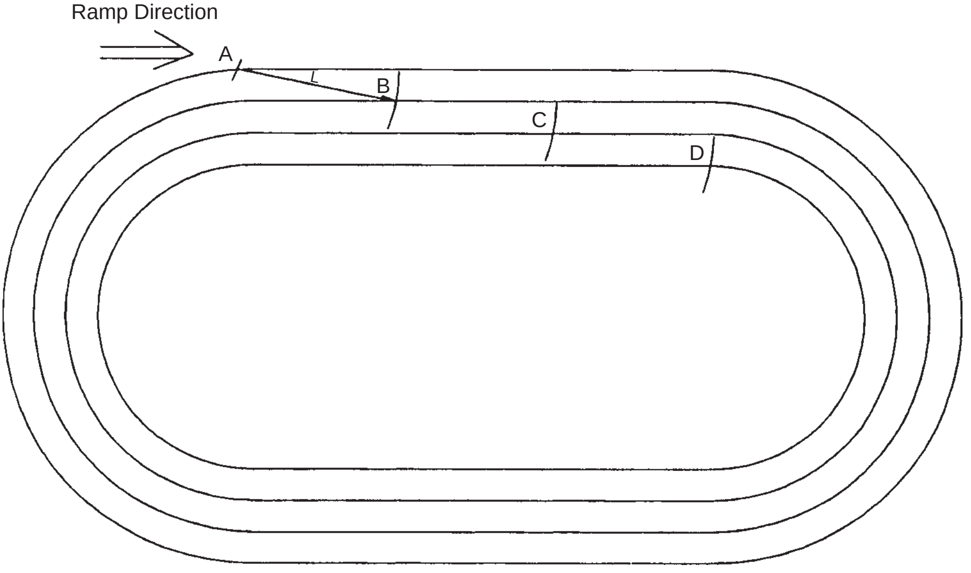


Figure 4.96. Point of ramp initiation and crest intercepts (Step 1).

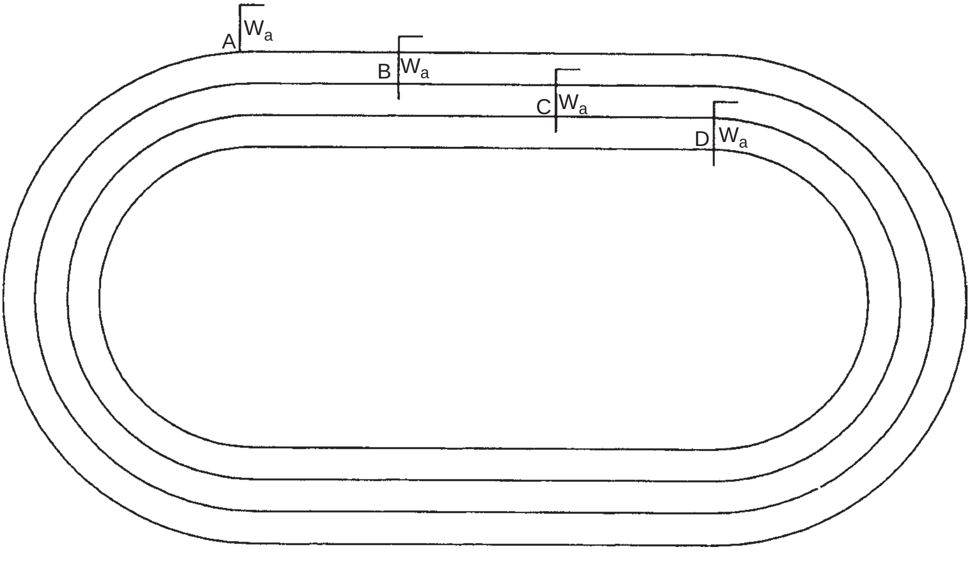


Figure 4.97. Addition of ramp width (Step 2).

Figure 4.97. A short length of line is drawn parallel to the crest line from the end in the ramp direction.

Step 3. Beginning with the lowermost crest, a smooth curve is drawn connecting the new crest with the old. This is shown in Figure 4.98.

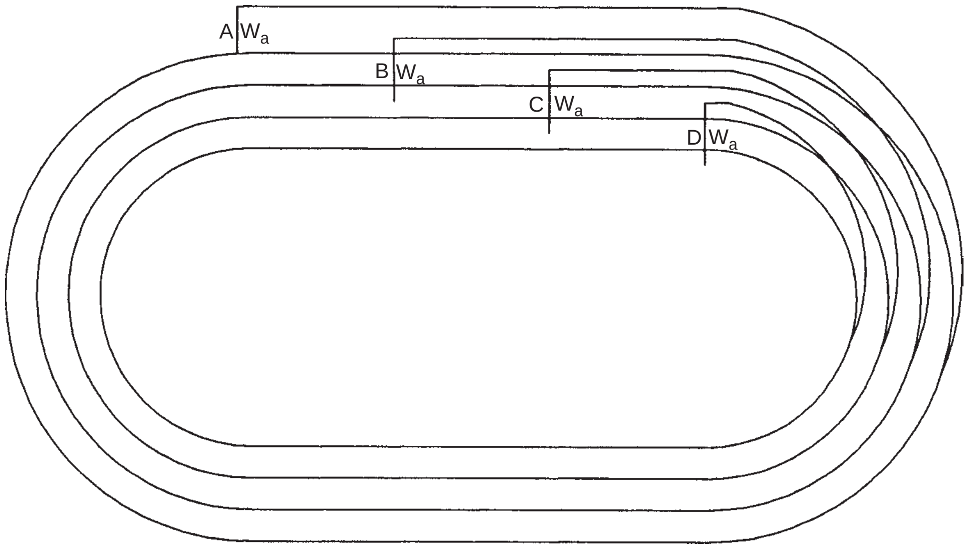


Figure 4.98. Drawing the new crest lines (Steps 3 and 4).

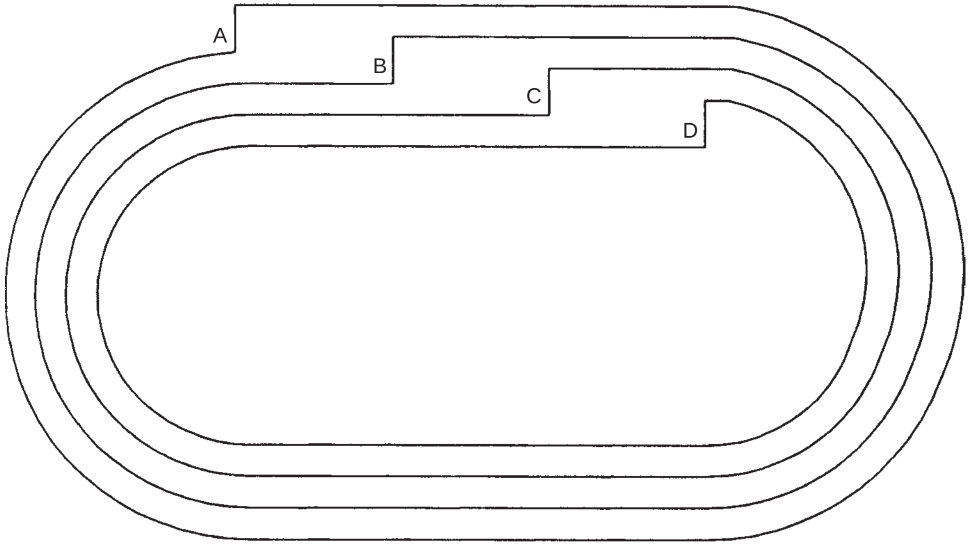


Figure 4.99. The pit as modified by the ramp (Step 5).

Step 4. The remaining new crest line portions are drawn parallel to the first crest working upwards from the lowest bench.

Step 5. The extraneous lines are removed from the design (Fig. 4.99).

Step 6. The toe lines at least for the lowest bench are added and the ramp to the pit bottom added. In Figure 4.100, the slopes have been shaded to assist in viewing the ramp.

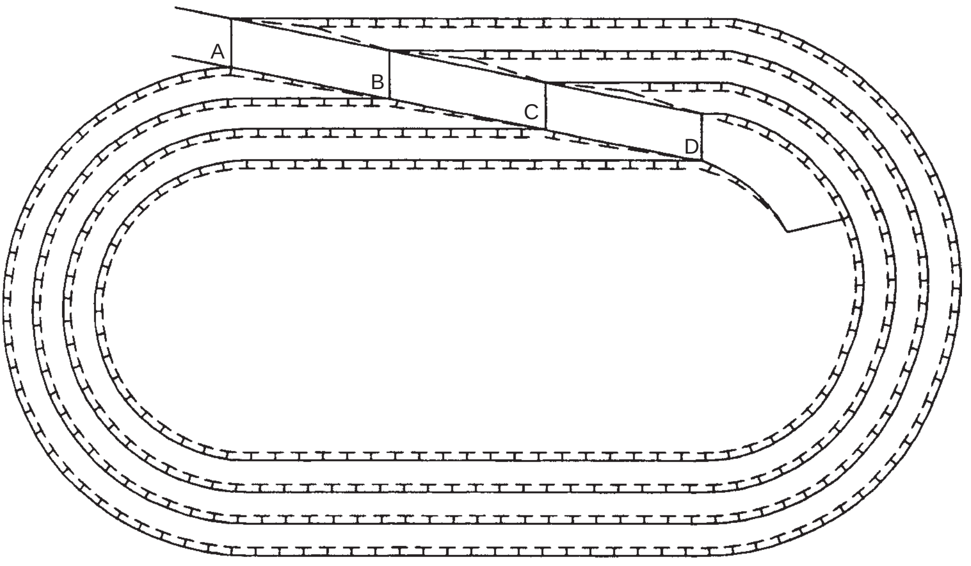


Figure 4.100. Addition of entrance ramp and toe lines (Step 6).

4.8.4 *Design of a switchback*

In laying out roads the question as to whether to:

(a) spiral the road around the pit,
(b) have a number of switchbacks on one side of the pit, or
(c) use some combination.

Generally (Couzens, 1979) it is desirable to avoid the use of switchbacks in a pit. Switchbacks:

– tend to slow down traffic,
– cause greater tire wear,
– cause various maintenance problems,
– probably pose more of a safety hazard than do spiral roads (vision problems, machinery handling, etc.).

Sometimes the conditions are such that switchbacks become interesting:

– when there is a gently sloping ore contact which provides room to work in switchbacks at little stripping cost;
– it may be better to have some switchbacks on the low side of the pit rather than to accept a lot of stripping on the high side.

The planner must take advantage of such things. The general axiom should be to design the pit to fit the shape of the deposit rather than vice versa. If switchbacks are necessary the planner should:

– leave enough length at the switchbacks for a flat area at the turns so that trucks don't have to operate on extremely steep grades at the inside of curves,
– consider the direction of traffic,
– consider problems the drivers may have with visibility,
– consider the effect of weather conditions on the design (ice, heavy rain, etc.).

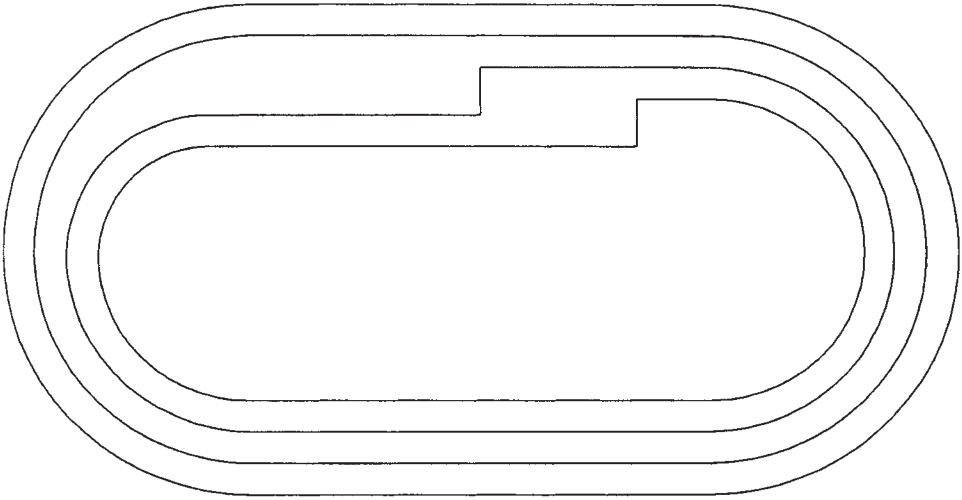


Figure 4.101. The starter pit for switchback addition to the north wall (Step 1).

In this section the steps required to add a switchback to the pit shown in Figure 4.89 will be described. The switchback will occur between the second and third benches on the north pit wall.

Step 1. The design will begin from the pit bottom. In this case the ramp moves into the as-designed pit wall. Figure 4.101 shows the modified pit with the crest lines drawn for benches 4 (lowermost) and 3. This is the same procedure as with the spiral ramp. The bench height has been selected as 30 ft and the road gradient is 10%. Hence the plan distance R is 300 ft.

Step 2. The center C used to construct the switchback is now located as shown in Figure 4.102. There are three distances involved L_1, L_2 and L_3. L_2 is the given crest-crest distance. Distances L_1 and L_3 must now be selected so that

$$L_1 + L_3 = R - L_2$$

In this particular case $L_1 = 0.5R = 150$ ft. Since $L_2 = 60$ ft, then $L_3 = 90$ ft. The center C is located at $L_2/2 = 30$ ft from the 3 construction lines. A vertical line corresponding to road width W is drawn at the end of L_3.

Step 3. In Figure 4.103 the curve with radius $R_2 = L_2/2$ is drawn from C. This becomes the inner road radius. It should be compared with the turning radius for the trucks being used. A second radius $R_3 = 2W$ is also drawn from C. The intersection of this curve with the horizontal line drawn from C becomes a point on the bench 2 crest. It is noted that actual designs may use values of R_3 different from that recommended here. This is a typical value. Portions of the bench 2 crest lines have been added at the appropriate distances.

Step 4. A smooth curve is now added going from line $a - b$ through crest point CP to line $c - d$. The designer can use some judgement regarding the shape of this transition line.

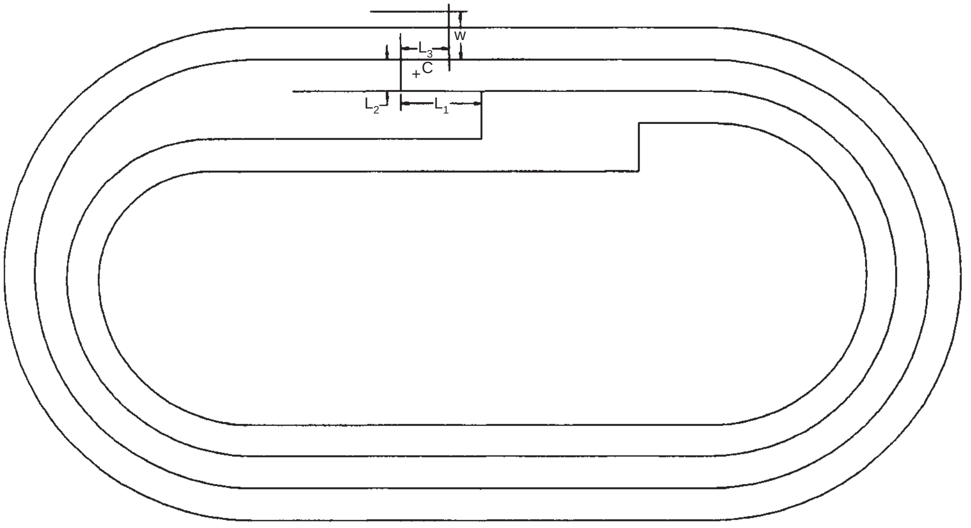


Figure 4.102. Construction lines for drawing the switchback (Step 2).

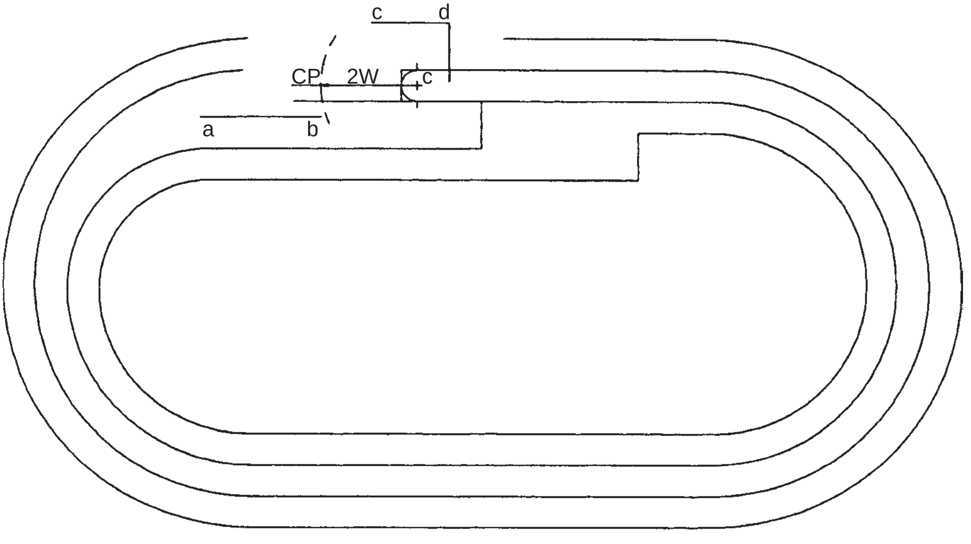


Figure 4.103. Crest lines and the crest point for bench 2 (Step 3).

Figure 4.104 shows the results. The lines surrounding point C simply represent edge of road (EOR).

Step 5. The crest line for bench 1 is then added parallel to that drawn for bench 2 (Fig. 4.105).

Step 6. The final crest line representation of the pit is drawn (Fig. 4.106). As can be seen the switchback occupies a broad region over a relatively short length. Thus it can be logically placed in a flatter portion of the overall pit slope.

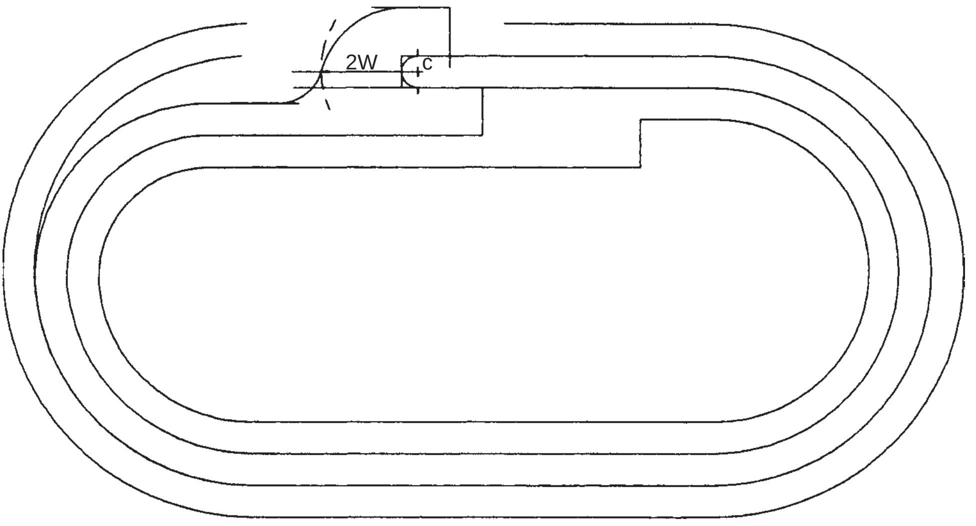


Figure 4.104. The transition curve has been added (Step 4).

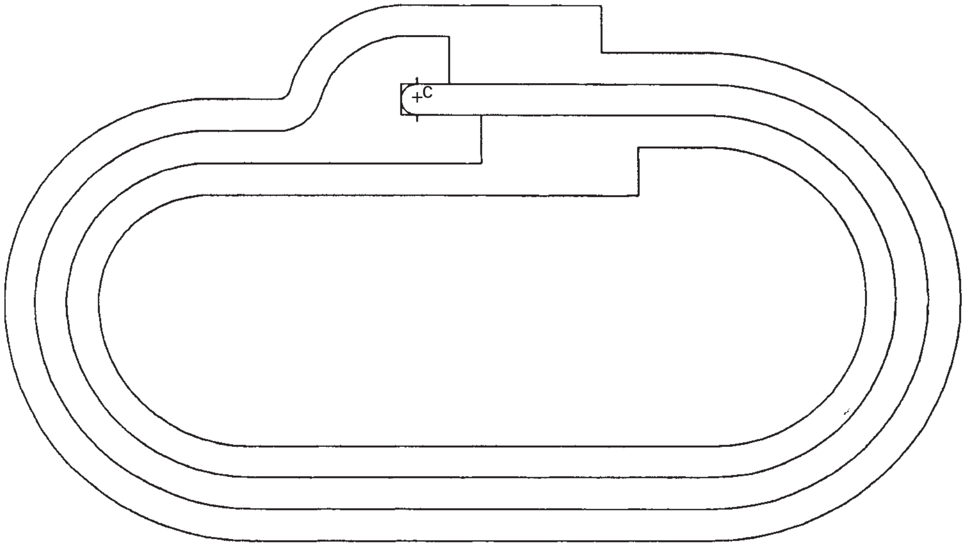


Figure 4.105. The crest line for bench 1 is added (Step 5).

Step 7. The toes are drawn and the lower section of the ramp (between bench 4 crest and the pit floor) added (Fig. 4.107).

Two examples of switchbacks are shown in Figure 4.108.

4.8.5 *The volume represented by a road*

The addition of a haulroad to a pit results in a large volume of extra material which must be removed or a similar volume in the pit which is sterilized (covered by the road). Thus even

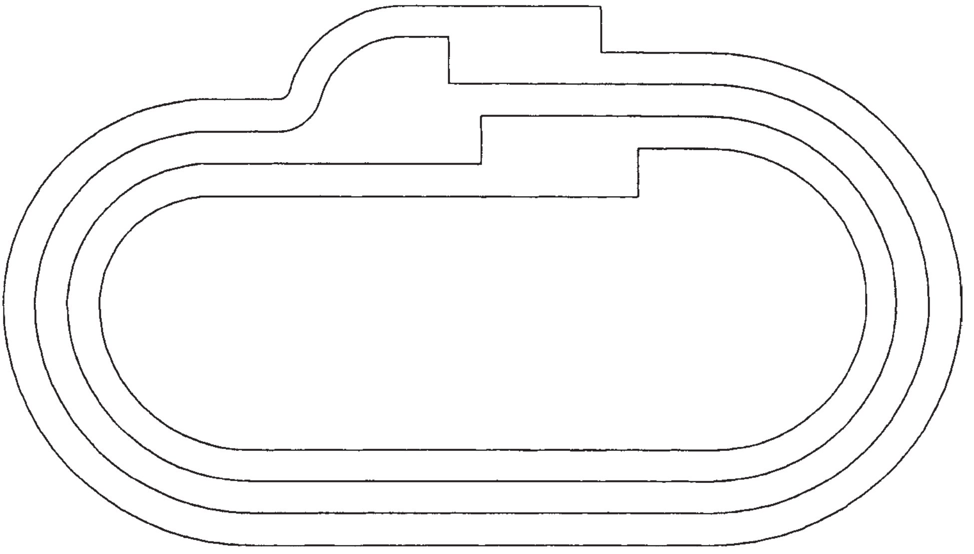


Figure 4.106. The final pit crest lines with the switchback (Step 6).

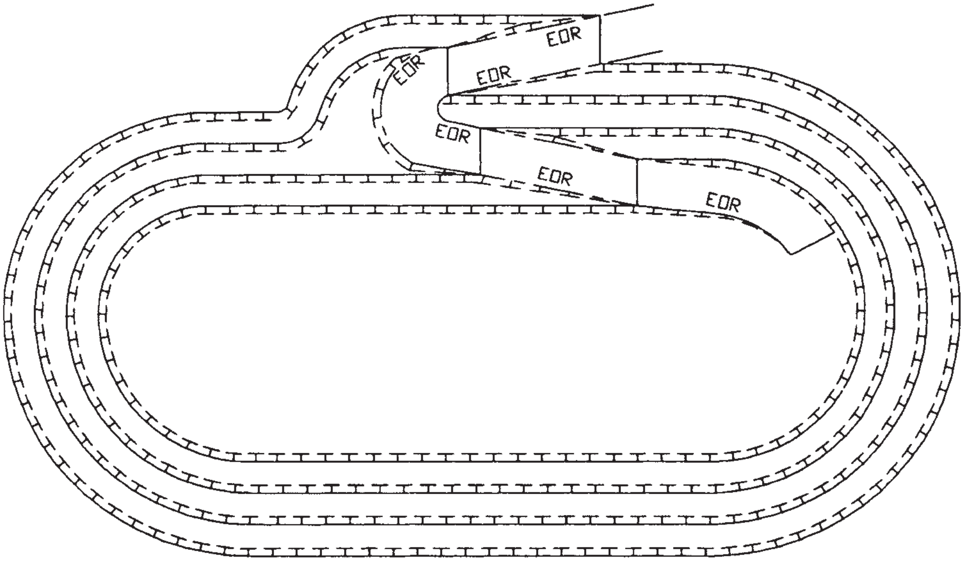


Figure 4.107. The lower entrance ramp and the toes are added (Step 7).

though production flexibility can be improved and the security of having several accesses to the pit can lead to other savings such as steeper interramp slopes, the additional haul roads are associated with significant expense. To demonstrate this, consider the pit shown in Figure 4.109 which contains no haulroad.

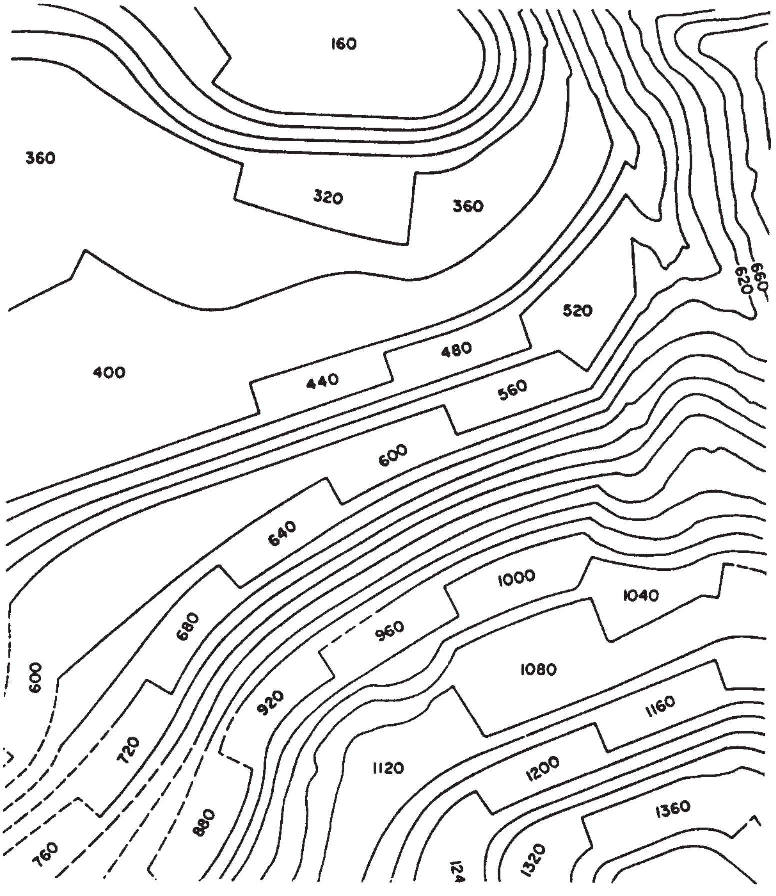


Figure 4.108. An example showing two different switchback (520 region and 1040 region) situations (Couzens, 1979).

The same pit with the road added, is shown in Figure 4.110. The shaded regions show the differences between sections A, B, C, D and E with and without the road.

In plan the length L of the road is

$$L = \frac{(\text{No. of benches} \times \text{Bench height})\ 100}{\text{Road grade (\%)}} = \frac{4 \times 30 \times 100}{10} = 1200\,\text{ft} \qquad (4.4)$$

Because the road is oriented at angle Θ to the pit axis, the length projected along the axis is

$$L_2 = L \cos \Theta = 1176\,\text{ft} \qquad (4.5)$$

The sections are made normal to this axis. They are spaced every 294 ft.

The road areas for each section are shown in Figure 4.111. The shaded boxes are of the same area

$$A = W_A \times \text{Bench height}$$

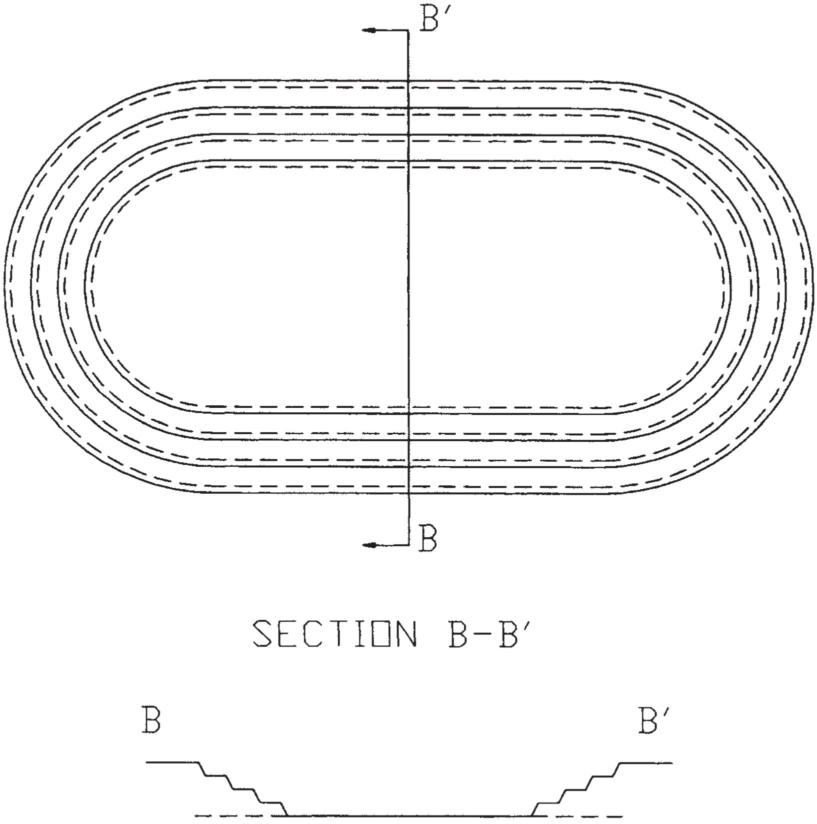


Figure 4.109. Plan and section views of a four bench pit without ramp.

They can be lined up as shown in Figure 4.111a. These in turn can be plotted such as shown in Figure 4.111b.

The volume contained in the ramp is that of a triangular solid of width W_A, length L_2 and height varying linearly from 0 to the pit depth (Fig. 4.112). This can be expressed as

$$V = \frac{1}{2} W_A L_2 \times \text{Pit depth} = \frac{1}{2} W_A L \cos\Theta \times \text{Pit depth} \tag{4.6}$$

which can be simplified to

$$V = \frac{1}{2} W_A \frac{(\text{Pit depth})^2}{\text{Grade }(\%\,)} 100 \cos\Theta \tag{4.7}$$

Since the apparent road width W_A is equal to

$$W_A = \frac{W_t}{\cos\Theta}$$

The simplified road volume formula becomes

$$V = \frac{1}{2} \frac{100 \times (\text{Pit depth})^2}{\text{Grade }(\%)} W_T \tag{4.8}$$

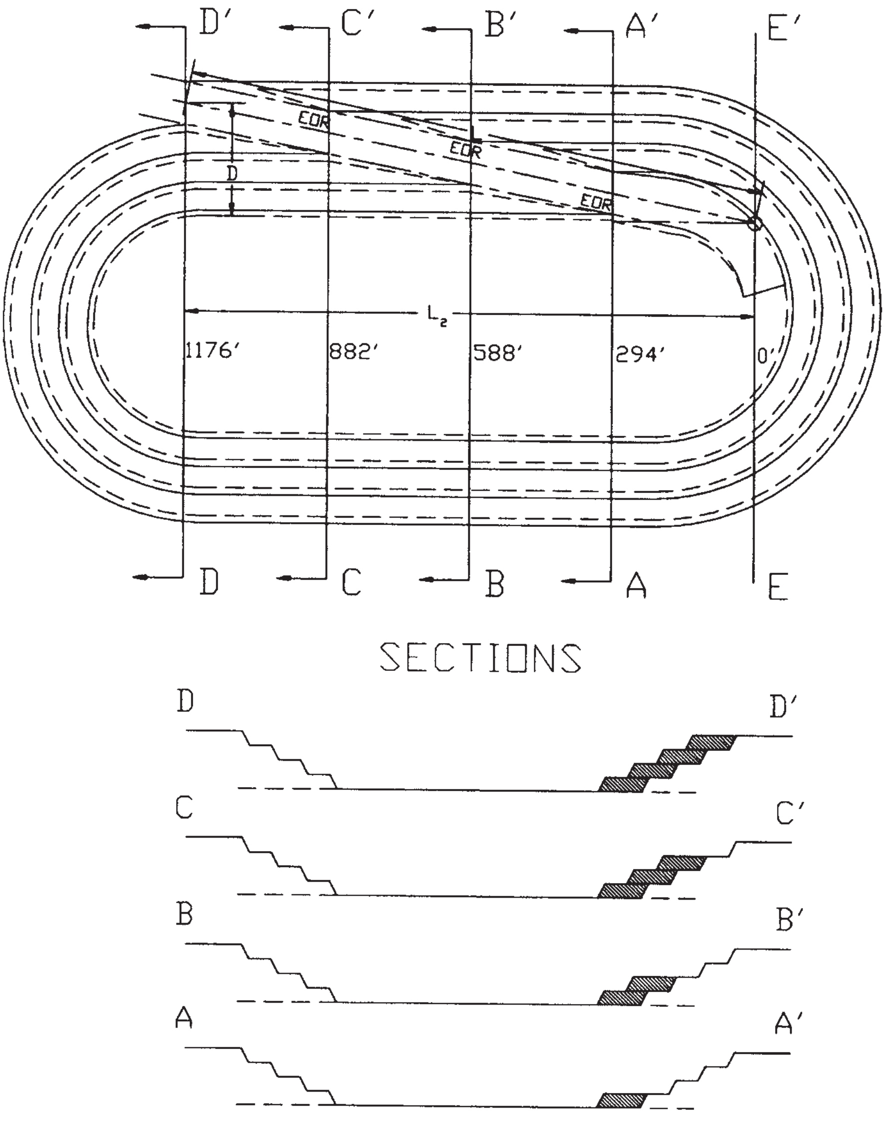


Figure 4.110. Plan and section views of a four bench pit with ramp.

In the present case the volume is

$$V = \frac{1}{2}\frac{100}{10}(120)^2 \times 90 = 6,480,00\,\text{ft}^3 = 240,000\,\text{yd}^3$$

For a tonnage factor of 12.5 ft^3/st, there are 518,400 st involved in the road.

The overall length of the road (L_{ov}) is given by

$$L_{ov} = \sqrt{L^2 + (\text{Pit depth})^2} \tag{4.9}$$

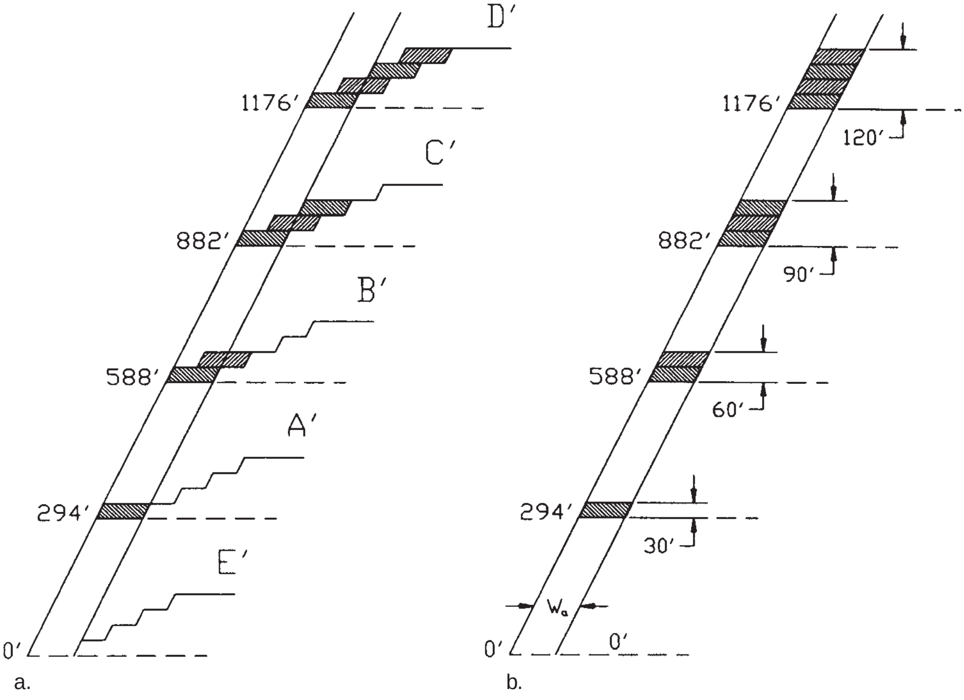


Figure 4.111. Construction to show road volume on each section.

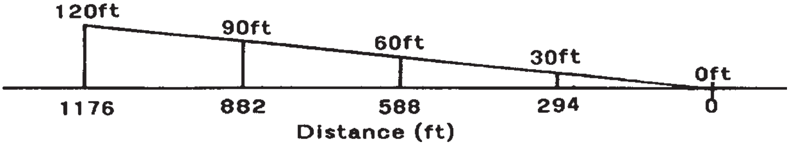


Figure 4.112. The volume involved in the ramp.

In this case it is

$$L_{ov} = \sqrt{(1200)^2 + (120)^2} = 1206\,\text{ft}$$

4.9 ROAD CONSTRUCTION

4.9.1 *Introduction*

Good haulroads are a key to successful surface mining operations. Poorly designed, constructed and maintained roads are major contributors to high haulage costs and pose safety hazards. In this section some of the basic design aspects will be discussed. Figure 4.113 shows a typical cross section through a road.

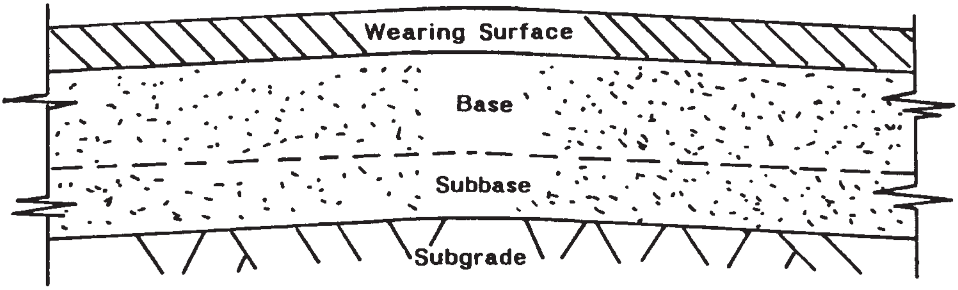


Figure 4.113. Simplified flexible pavement structure (Seelye, 1945).

Generally there are four different layers involved:
– subgrade,
– subbase,
– base,
– wearing surface.

The subgrade is the foundation layer. It is the structure which must eventually support all the loads which come onto the wearing surface. In some cases this layer will simply be the natural earth surface. In other and more usual instances, it will be the compacted rock or soil existing in a cut section or the upper layer of an embankment section.

The wearing surface provides traction, reduces tractive resistance, resists abrasion, ravelling and shear, transmits tire load to the base and seals the base against penetration of surface water. Although this surface may be asphalt or concrete, most typically it is crushed rock.

The base is a layer of very high stability and density. It's principal purpose is to distribute or 'spread' the stresses created by wheel loads acting on the wearing surface, so that they will not result in excessive deformation or the displacement of the subgrade. In addition it insulates the subgrade from frost penetration and protects the working surface from any volume change, expansion and softening of the subgrade.

The subbase which lies between the base and subgrade, may or may not be present. It is used over extremely weak subgrade soils or in areas subject to severe frost action. They may also be used in the interest of economy when suitable subbase materials are cheaper than base materials of a higher quality. Generally the subbase consists of a clean, granular material. The subbase provides drainage, resists frost heave, resists shrinkage and swelling of the subgrade, increases the structural support and distributes the load.

4.9.2 *Road section design*

In designing the road section, one begins with the maximum weight of the haulage equipment which will use the road. To be as specific as possible, assume that the haulage trucks have a maximum gross vehicle weight of 200,000 lbs including their 58 st payload. The load is distributed as follows:
– 33% on the front tires, and
– 67% on the dual rear tires.

The load on each of the front tires is 33,000 lbs. For each of the four rear tires (2 sets of duals) the load is 33,500 lbs. Thus the maximum loading to the wear surface is applied by

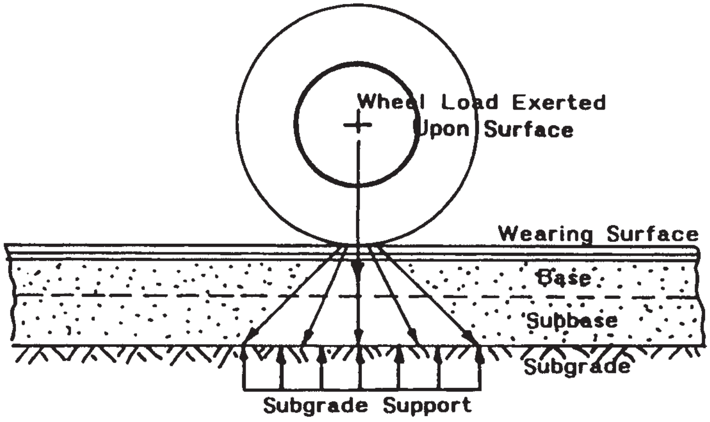


Figure 4.114. Load distribution beneath a tire (Seelye, 1945).

the rear tires. Although the contact pressure between the wheel and the road depends on the tire inflation pressure and the stiffness of the tire side walls, for practical purposes, the contact pressure is assumed to be equal to the tire pressure. Since for this truck, the inflation pressure is about 90 psi, the bearing pressure on the road surface is 90 psi or 12,960 psf. In lieu of knowing or assuming an inflation pressure, Kaufman & Ault (1977), suggest that a value of 16,000 psf (110 psi), will rarely be exceeded. The tire contact area is

$$\text{Contact area (in}^2) = \frac{\text{Tire load (lbs)}}{\text{Tire inflation pressure (psi)}} \tag{4.10}$$

For the rear tires

$$\text{Contact area (in}^2) = \frac{33{,}500}{90} = 372\,\text{inch}^2$$

Although the true contact area is approximately elliptical, often for simplicity the contact area is considered to be circular in shape. The contact pressure is usually assumed to be uniformly distributed. Because

$$\pi r^2 = 372\,\text{inch}^2$$

the radius of the tire contact area is

$$r \cong 11\,\text{inch}$$

and the average applied pressure is 90 psi (12,960 psf). As one moves down, away from the road surface, the force of the tire is spread over an ever increasing area and the bearing pressure is reduced. For simplicity, this load 'spreading' is assumed to occur at 45°. This is shown in Figure 4.114. Thus at a depth of 10 inches beneath the tire, the pressure radius would have increased to 21 inches and the pressure has dropped to 24.7 psi (3560 psf). However, for this truck there are dual rear wheels. Tire width is about 22 inches and the centerline spacing for the tires in each set is about 27 inches. This is shown diagrammatically in Figure 4.115.

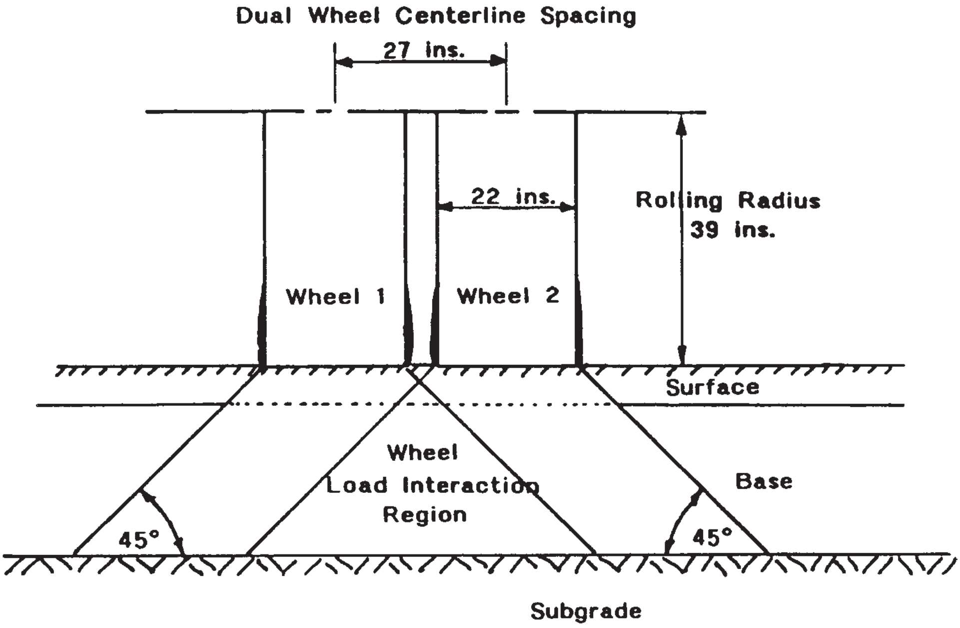


Figure 4.115. Load interaction with dual wheels.

As can be seen, the bearing pressure bulbs from each tire overlap. The greatest effect is observed along the line separating the tires. This interaction changes with tire width, tire separation and depth below the wear surface. To take this into account, Kaufman & Ault (1977) suggest using an equivalent single tire wheel load (L_E) which is 20% higher than the single tire load (L_T). Thus,

$$L_E = 1.20 \times L_T \tag{4.11}$$

In the case of the 58 st capacity truck

$$L_E = 1.20 \times 33,500 \cong 40,000\,\text{lb}$$

The combined subbase, base and wearing surface thickness must be sufficiently large so that the stresses occurring in the subgrade will not cause excessive distortion or displacement of the subgrade soil layer.

As a first guide, one can compare the required wear surface pressure to the bearing capacity of various subgrade materials. These are given in Table 4.6.

As can be seen, any subgrade that is less consolidated than soft rock will require additional material in order to establish a stable base. If, for example, the subgrade is a compact sand-clay soil with a bearing capacity of 6000 psf, then base/subbase materials of suitable strength would have to be placed down to increase the distance between the wear surface and the subgrade. Using the approach described earlier

$$\pi(11 + t)^2 \times 6000 = \pi(11)^2 \times 12,960$$

Table 4.6. Bearing capacities of subgrade materials (Kaufman & Ault, 1977).

Material	1000 psf
Hard, sound rock	120
Medium hard rock	80
Hard pan overlying rock	24
Compact gravel and boulder-gravel formations; very compact sandy gravel	20
Soft rock	16
Loose gravel and sandy gravel; compact sand and gravelly sand; very compact sand – inorganic silt soils	12
Hard dry consolidated clay	10
Loose coarse to medium sand; medium compact fine sand	8
Compact sand-clay soils	6
Loose fine sand; medium compact sand – inorganic silt soils	4
Firm or stiff clay	3
Loose saturated sand clay soils, medium soft clay	2

the minimal required thickness (t) would be

$$t \cong 5\,\text{inch}$$

The technique often applied to determine the working surface, base and subbase thicknesses involves the use of California bearing ratio (CBR) curves. The CBR test is an empirical technique for determining the relative bearing capacity of the aggregate materials involved in road construction. In this test the aggregate material with a maximum size of $^3\!/_4$ inch is placed in a 6 in diameter metal mold. The material is compacted by repeatedly dropping a 10 lb weight through a height of 18 in. After compaction, a cylindrical piston having an end area of 3 inch^2 is pushed into the surface at a rate of 0.05 inch/minutes. The CBR is calculated by dividing the piston pressure at 0.1 or 0.2 inch penetration by reference values of 1000 psi for 0.1 inch and 1500 psi for 0.2 inch. These standard values represent the pressures observed for a high quality, well graded crushed stone reference material. The calculated pressure ratios are multiplied by 100 to give the CBR value expressed as a percent. Figure 4.116 shows design curves based upon the use of CBR values. The subbase thickness has been plotted against CBR/soil type for various wheel loads.

To demonstrate the use of these curves, consider the 58-st capacity truck travelling over a haulroad which the subgrade material is a silty clay of medium plasticity (CBR = 5). One finds the intersection of CBR = 5 and the 40,000 lb equivalent single wheel load. Moving horizontally it is found that the required distance between the wear surface and the subgrade must be a minimum of 28 inches.

Fairly clean sand with a CBR of 15 is available to serve as subbase material. Repeating the process, one finds that this must be kept 14 inches away from the wear surface. The base material is well graded, crushed rock with a CBR rating of 80. The intersection of the 40,000 lb curve and CBR = 80 occurs at 6 inches. This 6 inch gap between the top of the base and the wear surface is intended to accommodate the wear surface thickness. If the actual wear surface is thinner than this, the remaining space is simply added to the base thickness (CBR equal to at least 80). Figure 4.117 shows the final results (Kaufman & Ault, 1977).

In most open pit mines, the wear surface is formed by well graded, crushed rock with a maximum dimension smaller than that used as base. Since traffic loading is directly applied

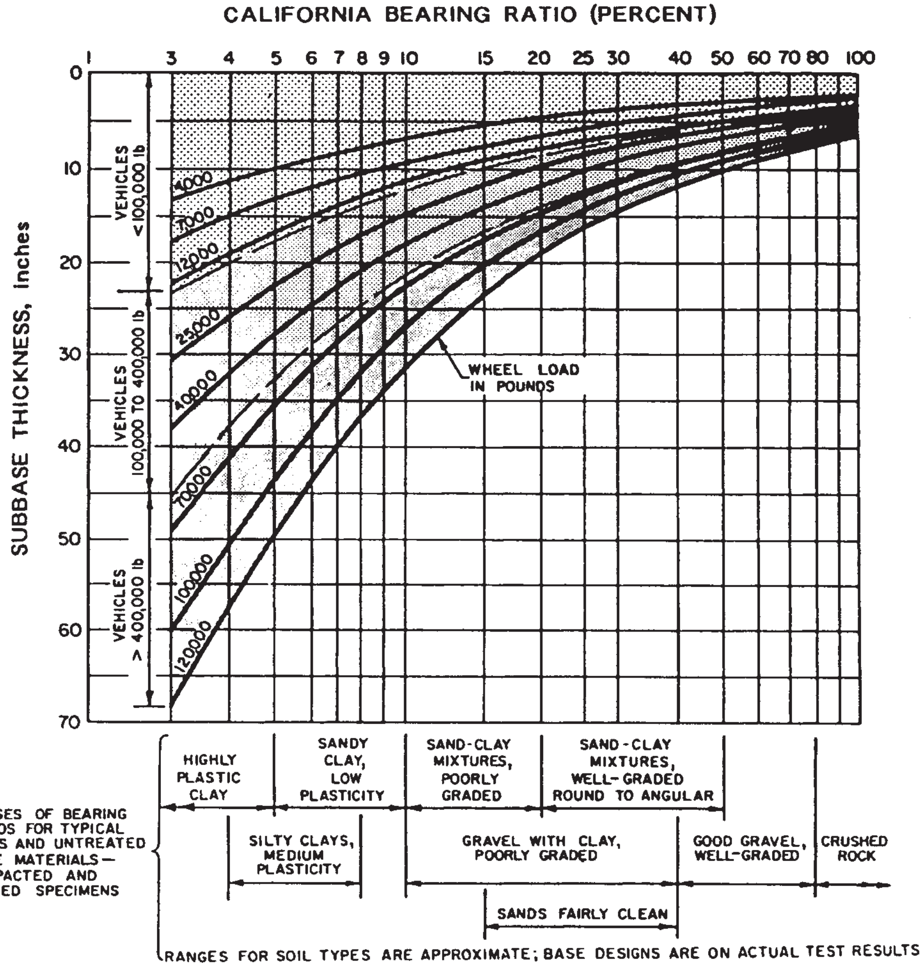


Figure 4.116. CBR curves (Kaufman & Ault, 1977).

to the aggregate layer, the upper most aggregate layer must possess sufficient strength and rutting resistance to minimize both

– bearing capacity failure, and
– rutting failure

within the layer.

The aggregate layer must also possess good wear resistance to minimize attrition under traffic. Table 4.7 indicates an acceptable aggregate size distribution (gradation) for this wearing surface.

Particle gradation is the distribution of the various particle size fractions in the aggregate. A well graded aggregate has a good representation of all particle size fractions from the maximum size through the smaller sizes. This is needed so that particles lock together forming a dense, compact surface. The opposite of a well graded aggregate is one which is poorly graded. Here the particles are all about the same size. Such a distribution might

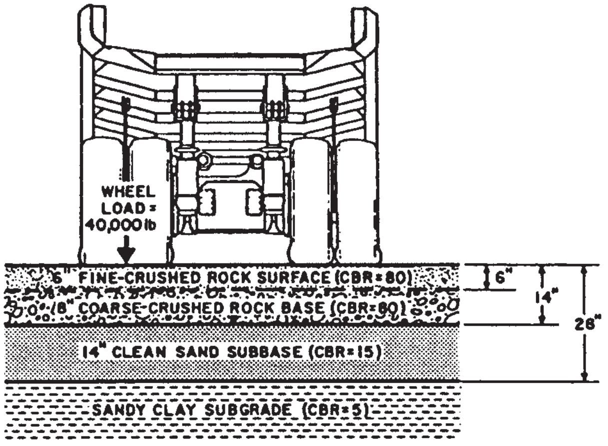


Figure 4.117. Example of mine road construction (Kaufman & Ault, 1977).

Table 4.7. Desired characteristics for a crushed stone running surface (Kaufman & Ault, 1977).

Screen size	Material passing (%)
1½ inches	100
1 inch	98
¾ inch	92
⅜ inch	82
No. 4 mesh	65
No. 10 mesh	53
No. 40 mesh	33
No. 200 mesh	16
Liquid limit	25.2
Plasticity limit	15.8
Plasticity index	9.4
Optimum moisture content during placing	12.2

be used as part of a runaway ramp with the objective being that of creating a high rolling resistance.

The use of CBR curves requires laboratory tests or the assumption of CBR values of subgrade, and available base or subbase materials. The most economical combination is used. The CBR curves show directly the total thickness needed over any subgrade soil. The total subbase and the base thickness is created by putting down a series of relatively thin layers of the correct moisture content. Compaction is done between layers.

4.9.3 *Straight segment design*

Figure 4.118 shows a typical cross-section through a mine haul road carrying two way traffic. As can be seen there are three major components to be considered:

a) travel lane width,
b) a safety berm,
c) a drainage ditch.

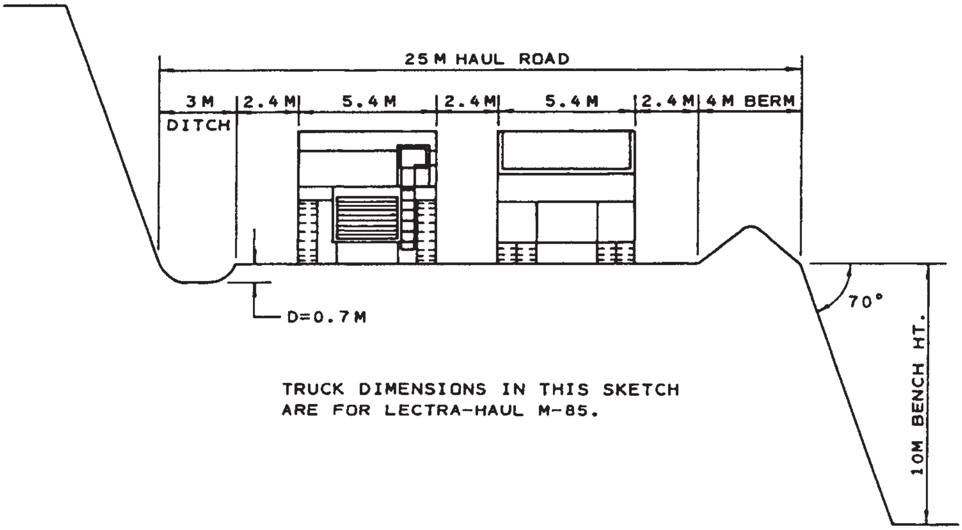


Figure 4.118. Typical design haulroad width for two-way traffic using 85 st capacity trucks (Couzens, 1979).

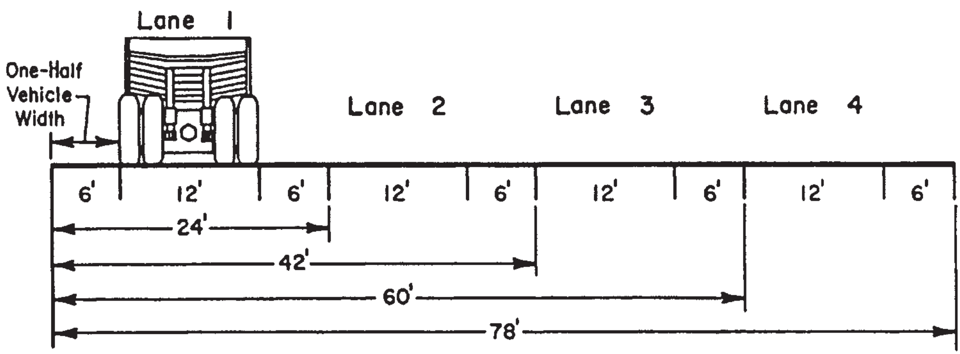


Figure 4.119. Multi-lane road design widths (Kaufman & Ault, 1977).

The width of each is added together to obtain the total roadway width.

The width criteria for the traveled lane of a straight haul segment should be based on the widest vehicle in use.

The 1965 *AASHO Manual for Rural Highway Design* recommends that each lane of travel should provide clearance to the left and right equal to one-half of the vehicle width. This is shown in Figure 4.126 for a 12-ft wide truck.

Values for other truck widths are given in Table 4.8. Typical widths of haulage trucks used in open pit mines are listed in Table 4.9.

For the two-way traffic which is most common in open pit mines, the rule of thumb is that roadway width should be no less than four times the truck width (Couzens, 1979):

$$\text{Roadway width} \geq 4 \times \text{Truck width} \tag{4.12}$$

Table 4.8. Recommended lane widths for tangent sections (Kaufman & Ault, 1977).

Vehicle width (ft)	1 lane	2 lanes	3 lanes
8	16	28.0	40
9	18	31.5	45
10	20	35.0	50
11	22	38.15	55
12	24	42.0	60
13	26	45.5	65
14	28	49.0	70
15	30	52.5	75
16	32	56.0	80
17	34	59.5	85
18	36	63.0	90
19	38	66.5	95
20	40	70.0	100
21	42	73.5	105
22	44	77.0	110
23	46	80.5	115
24	48	84.0	120
25	50	87.5	125
26	52	91.0	130
27	54	94.5	135
28	56	98.0	140

Table 4.9. Widths for various size rear dump trucks (this width includes the safety berm).

Truck size	Approx. width (m)	4 × width (m)	Design width	
			m	ft
35 st	3.7	14.8	15	50
85 st	5.4	21.6	23	75
120 st	5.9	23.6	25	85
170 st	6.4	25.6	30	100

Some mines have two lanes of traffic in one direction to allow passing for loaded uphill traffic. The downhill empty traffic travels in a single lane. A rule of thumb for the width of such a three lane road is 5 times the truck width.

The steps to be followed in selecting a design width are (Kaufman & Ault, 1977):

1. Define the width of all equipment that may have to travel the haulage road.
2. Solicit dimensional data for any anticipated new machines.
3. Determine the overall width of any equipment combinations that may be involved in a passing situation.
4. Delineate the location of road segments requiring a greater than normal width.

There may be wider stretches of road where there is merging of traffic streams such as near a crusher. Curves and switchbacks require special consideration. These will be discussed later.

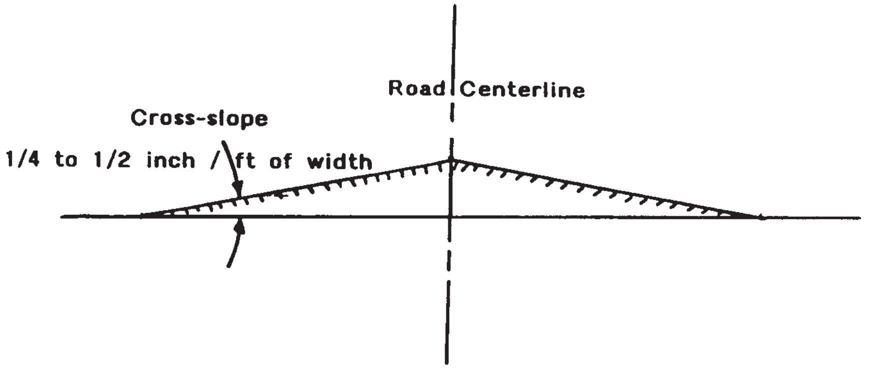


Figure 4.120. Cross slope design.

Table 4.10. Design widths (ft) for curves – single unit vehicles (Kaufman & Ault, 1977).

Radius (*R*) on inner edge of pavement (ft)	One-lane haulageway, vehicle category				Two-lane haulageway, vehicle category				Three-lane haulageway, vehicle category			
	1	2	3	4	1	2	3	4	1	2	3	4
Minimum	29	34	45	70	51	60	79	123	73	86	113	176
25	27	34	44	68	48	60	76	119	68	86	109	170
50	25	31	41	63	44	54	72	110	63	77	103	158
100	24	29	39	59	42	51	69	103	60	73	99	147
150	24	29	39	58	41	50	68	101	59	72	97	145
200	23	29	38	57	41	50	67	101	59	712	96	144
Tangent	23	28	37	56	40	48	65	98	57	69	93	140

The road surface is often slightly crowned such as shown in Figure 4.120, to facilitate water runoff. The cross slope is expressed in inches per foot of width. Most mine roads are constructed of gravel and crushed rock. In this case, except where ice/mud is a problem, the cross slope should be ½ inch per foot (0.04 ft/ft). For relatively smooth road surfaces such as asphaltic concrete which can rapidly shed water or roads which have ice/mud problems, a cross slope of ¼ inch per foot (0.02 ft/ft) is appropriate.

For single lanes, it is necessary to decide whether the left edge should be higher than the right or vice-versa. For three-lane surfaces, there should be a continuous cross slope for the two lanes having traffic in the same direction. It should be noted that the use of a cross-slope increases the steering effort by the driver. Thus there must be a balance between steerability and water drainage.

4.9.4 *Curve design*

For straight sections it was recommended that the left and right vehicle clearances should be half of the vehicle width. In the case of curves this distance must be increased both due to vehicle overhang and increased driving difficulty.

Tables 4.10 and 4.11 provide the design widths as a function of the inner pavement radius for various combinations of vehicle size, vehicle type and roadway types. For

Table 4.11. Design widths (ft) for curves – articulated vehicles (Kaufman & Ault, 1977).

Radius (R) on inner edge of pavement (ft)	One-lane haulageway, vehicle category			Two-lane haulageway, vehicle category			Three-lane haulageway, vehicle category		
	2	3	4	2	3	4	2	3	4
25	38	68	86	66	119	151	95	170	215
50	32	57	71	56	99	124	80	142	177
100	28	48	58	50	83	101	71	119	144
150	27	44	52	47	76	91	68	109	130
200	26	42	49	46	73	85	66	104	122
Tangent	25	41	41	44	71	72	63	102	103

Table 4.12. Minimum single unit haulage truck turning radius (Kaufman & Ault, 1977).

Vehicle weight classification	Gross vehicle weight (GVW) (lb)	Minimum turning radius (ft)
1	$< 100,000$	19
2	100–200,000	24
3	200–400,000	31
4	$> 400,000$	39

reference approximate turning radii are indicated by gross vehicle weight categories in Table 4.12.

For example, if a single unit haulage truck of weight classification 3 is to traverse a 100 ft minimum radius curve, the two lane width should be 69 ft. For a straight road segment the corresponding width is 65 ft. Hence the effect of the curve is to add 4 ft to the width.

Vehicles negotiating curves are forced outward by centrifugal force. For a flat surface this is counteracted by the product of the vehicle weight and the side friction between the roadway and the tires (Fig. 4.121).

For certain combinations of velocity and radius the centrifugal force will equal or exceed the resisting force. In such cases, the vehicle skids sideways. To assist the vehicles around the curves, the roadways are often banked. This banking of curves is called superelevation. The amount of superelevation (cross slope) can be selected to cancel out the centrifugal force. The basic equation is

$$e + f = \frac{V^2}{15R} \tag{4.13}$$

where e is the superelevation rate (ft/ft); f is the side friction factor; V is the vehicle speed (mph); R is the curve radius (ft). If $f = 0$, then the vehicle would round the curve without steering effort on the part of the operator. If however the operator would maintain a speed different from that used in the design, then he would have to steer upslope (in the case of too low a speed) of downslope (too high a speed) to maintain the desired path. Under ice and snow conditions, too slow a speed on such super elevated curves could lead to sliding down the slope.

Table 4.13 gives recommended superelevation rates as a function of curve radius and vehicle speed. The table can also be used to suggest a safe speed for a given radius and superelevation rate.

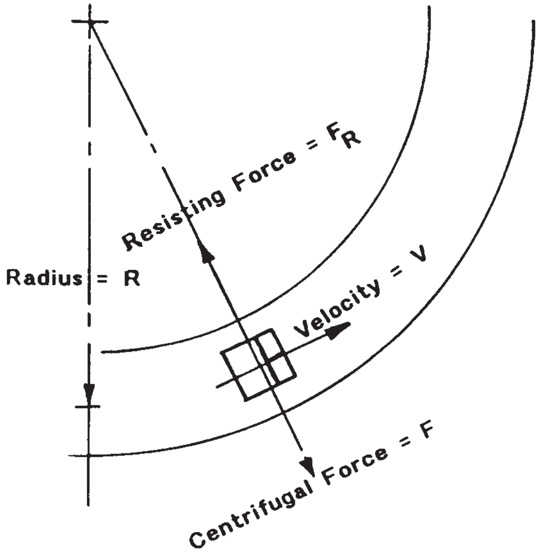


Figure 4.121. Centrifugal force effects on curves.

Table 4.13. Recommended superelevation rates (feet per foot of width) (Kaufman & Ault, 1977).

Radius of curve (ft)	Speed of vehicle (mph)					
	10	15	20	25	30	35 and over
50	0.04	0.04				
100	0.04	0.04	0.04			
150	0.04	0.04	0.04	0.05		
250	0.04	0.04	0.04	0.04	0.06	
300	0.04	0.04	0.04	0.04	0.05	0.06
600	0.04	0.04	0.04	0.04	0.04	0.05
1000	0.04	0.04	0.04	0.04	0.04	0.04

Table 4.14. Recommended rate of cross-slope change (Kaufman & Ault, 1977).

Vehicle speed (mph)	10	15	20	25	30	35 and above
Cross-slope change in 100-ft length of haulageway (ft/ft)	0.08	0.08	0.08	0.07	0.06	0.05

There is a certain distance required to make the transition from the normal cross-slope section to the superelevated portion and back again. This is called the superelevation runout. The purpose is to help ease the operator into and out of the curve. Part of the transition can be placed in the straight (tangent) portion and part in the curve. The design criteria of $\frac{1}{3}$ inch curve and $\frac{2}{3}$ inch the tangent is used here. The recommended rate of cross-slope change as a function speed is given in Table 4.14.

To illustrate the use of this table, assume a vehicle is traveling at 35 mph on tangent with normal cross slope 0.04 ft/ft to the right. It encounters a curve to the left necessitating a superelevation rate of 0.06 ft/ft to the left. The total cross-slope change required is 0.10 ft/ft $(0.04 + 0.06)$. The table recommends a 0.05 ft/ft cross-slope change in 100 ft. Thus the total runout length is computed as 200 ft $[(0.10/0.05) \times 100 = 200]$. One-third of this length should be placed in the curve and two-thirds on the tangent.

4.9.5 *Conventional parallel berm design*

U.S. federal law (MSHA, 1992) contains the following guidance regarding the need for berms/guardrails in open pit mines (Section 57.9300):

(*a*) *Berms or guardrails shall be provided and maintained on the banks of roadways where a drop-off exists of sufficient grade or depth to cause a vehicle to overturn or endanger persons in equipment.*

(*b*) *Berms or guardrails shall be at least mid-axle height of the largest self-propelled mobile equipment which usually travels the roadway.*

(*c*) *Berms may have openings to the extent necessary for roadway drainage.*

(*d*) *Where elevated roadways are infrequently traveled and used only by service or maintenance vehicles, berms or guardrails are not required (when certain very specific conditions are met).*

The principal purpose of these berms is to redirect the vehicle back onto the roadway and away from the edge. Their effectiveness in this regard is controlled by berm face angle, berm facing, the angle of incidence, and primarily by berm height. The stopping of runaway vehicles is accomplished by median berms (Subsection 4.9.6) or special escapeways. One negative effect of berms is the possibility of the vehicles overturning due to climbing the sides.

There are two principal berm designs in common use today. The triangular or trapezoidal shaped berm is generally formed from blasted materials. The sides stand at the angle of repose of the material. The second type is the boulder-faced berm. Here, large boulders, lined up along the haulage road, are backed with earthen material or blasted rock.

For the triangular berms, the design rule of thumb is that the height must be equal to or greater than the static rolling radius (SRR) of the vehicle's tire. For boulder-faced berms, the height of the berm should be approximately equal to the tire height. Figure 4.122 shows the relationship between the static rolling radius and haulage vehicle carrying capacity. Tire height (TH) is about equal to:

$$TH = 1.05 \times 2 \times SRR \tag{4.14}$$

4.9.6 *Median berm design*

Some means should be provided on haulroads to reduce truck speed or handle the truck that loses its brakes. This is particularly true when long, downhill loaded hauls are involved. Currently the most successful technique is through the use of median berms, also known as 'straddle berms' or 'whopper stoppers' (Winkle, 1976a,b). These are constructed of sand or some other fine grained material. The height of these berms is designed to impinge on the under-carriage of the truck. Since the typical distance between the road surface to the undercarriage is of the order of 2 to 3 ft for the range of available haulers, it is not necessary to build a big barrier providing just another crash hazard.

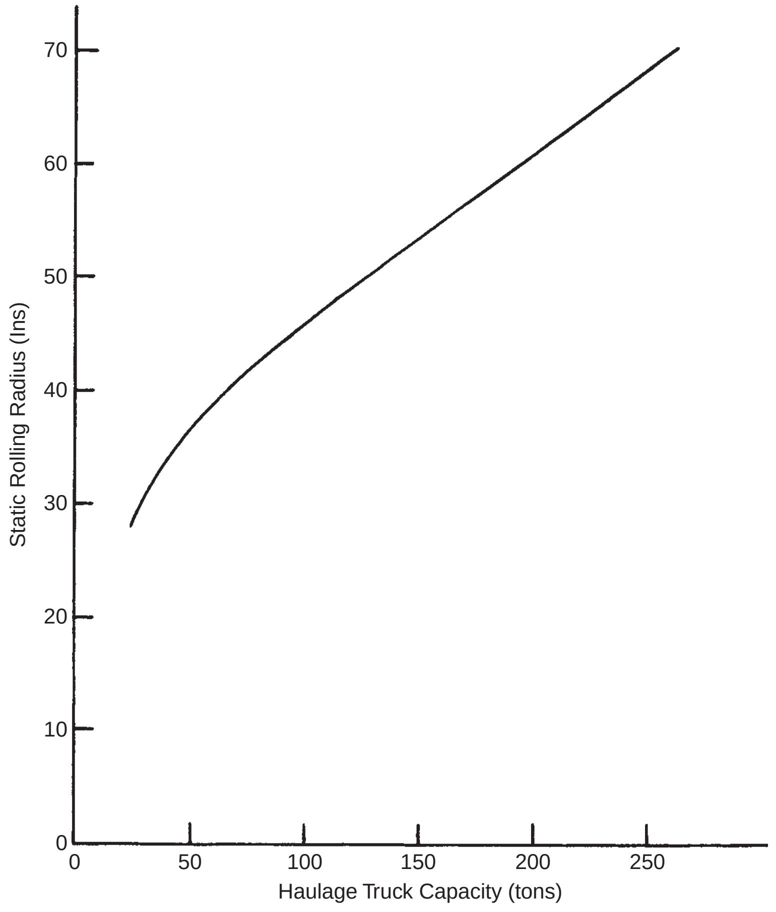


Figure 4.122. Static rolling radius as a function of haulage truck capacity (Goodyear, 1992).

Guidance in median berm design provided by Kaufman & Ault (1977) is given in Figure 4.123. The dimensions corresponding to the letters in the figure are given in Tables 4.15 and 4.16. The vehicle categories are based upon gross vehicle weights.

Training the driver to get onto the berm or into the bank just as soon as they start to lose control of their truck and before they build up speed is as important, or more important, than the berm design itself (Couzens, 1979).

4.9.7 *Haulage road gradients*

A number of rules of thumb regarding haulage road gradients have been provided by Couzens (1979). These are given below:

1. In a pit where there is a considerable vertical component to the haulage requirement, the grade will have to be fairly steep to reduce the length of the road and the extra material necessary to provide the road length. The practical maximum grade is considered to be 10%.

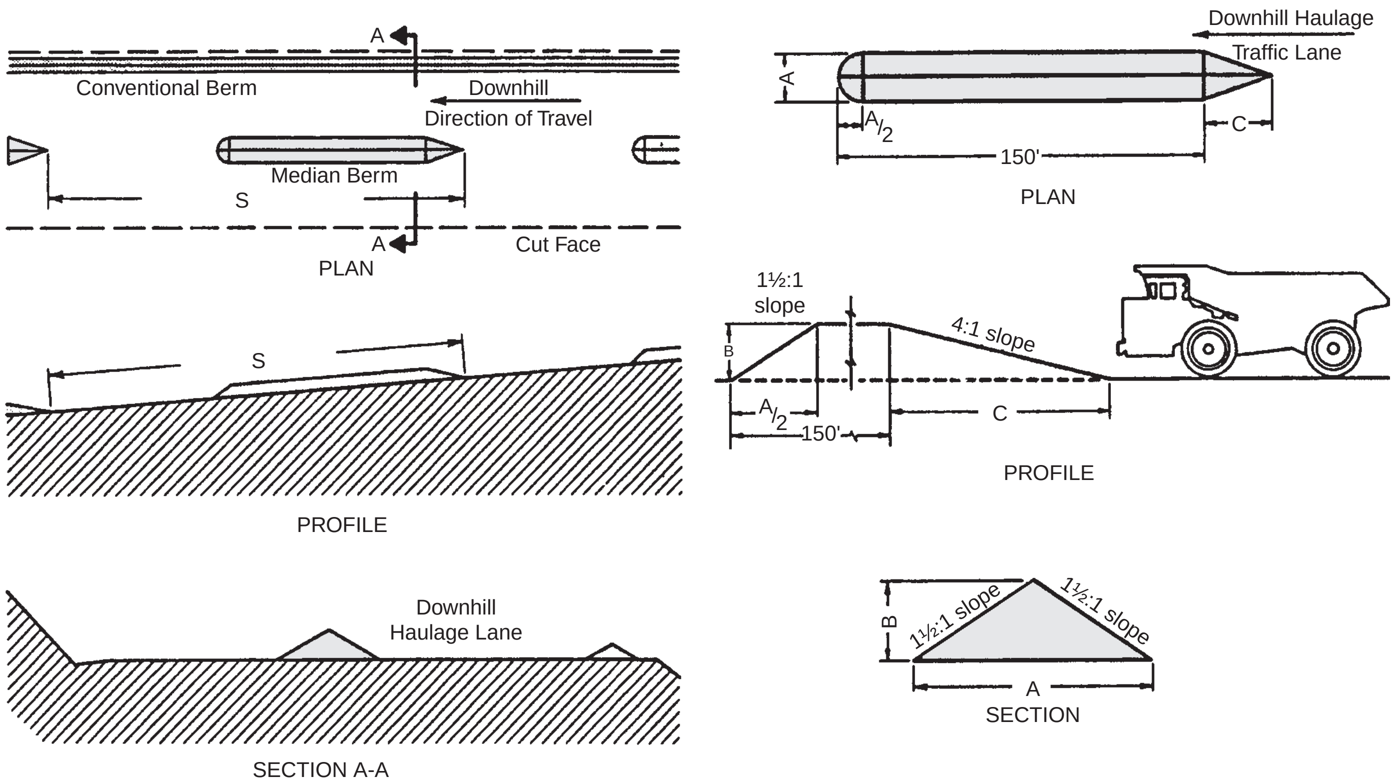


Figure 4.123. Runaway-vehicle collision berms (Kaufman & Ault, 1977)

Table 4.15. Typical median berm dimensions (see Figure 4.123) (Kaufman & Ault, 1977).

	A	B	C
Category 1 13 to 25 st <100,000 lb	11′–12′	3.5′–4′	14′–16′
Category 2 28 to 50 st 100,000–200,000 lb	12′–15′	4′–5′	16′–20′
Category 3 55 to 120 st >200,000–400,000 lb	15′–18′	5′–6′	20′–24′
Category 4 120 to 250 st > 400,000 lb	18′–32′	6′–11′	24′–44′

Table 4.16. Berm spacing (S) expressed in feet assuming that the initial speed at brake failure is 10 mph.

Equivalent downgrade, (%)	Maximum permissible vehicle speed or terminal speed at entrance to safety provision (mph)							
	15	20	25	30	35	40	45	50
1	418	1003	1755	2674	3760	5013	6433	8021
3	140	335	585	892	1254	1671	2145	2674
5	84	201	351	535	752	1003	1287	1604
7	60	144	251	382	537	716	919	1146
9	47	112	195	297	418	557	715	892
11	38	92	160	243	342	456	585	730
13	33	78	135	206	290	386	495	617
15	28	67	117	179	251	335	429	535

A number of pits operate quite well at 10% grades both favorable and unfavorable to the loads.

2. An 8% road grade is probably preferred providing that it does not cause too much extra stripping or unduly complicate the road layout. This grade provides more latitude in: (a) building the road and (b) fitting in bench entries without creating some locally over-steep places, than do steeper grades.

3. There is normally nothing to be gained by flattening the road below 8%, unless there is a long distance to travel without requiring much lift. The extra length on the grade and the complications of fitting the road into the available room or doing extra stripping would probably offset any increase in uphill haul speed.

4. Pit geometry is the prime consideration and roads are designed to fit the particular situation. Thus there often will be a number of different grade segments in haul roads.

4.9.8 *Practical road building and maintenance tips*

The preceding parts of this section have dealt with some of the general road design principles. Winkle (1976a,b) has provided a number of practical tips based upon many years of practical experience. Some of these have been included below.

1. The size of the orebody and the nature of the overlying topography will have considerable impact upon road design. When the orebody is small it will likely be advantageous to strip immediately to the projected pit limits, since some mining inefficiencies occur when mining areas overlap. This dictates an immediate final road layout on the backslope which is planned to avoid expensive modification. For very large orebodies, particularly where an outcrop of ore is exposed, it is highly unlikely that initial stripping will extend to the final planned perimeter. Careful study of the topography is required to ensure proper rapid access. The cost of rehandling material dumped within eventual pit limits must be weighed against increased haul distances, sharp curves, etc.

2. Change in equipment size frequently is a cause of road modification, particularly width. Pit design should incorporate allowance for reasonable future equipment size increases.

3. When mixed haulage fleets with varying speeds are used or where trucks hauling from two or more shovels are using the same haul roads, passing lanes on long grades should be considered.

4. Short radius curves result in reduced productivity, high tire cost, high maintenance cost (particularly electric wheels) and introduce additional safety hazards into the operation. Switchbacks are to be avoided unless a tradeoff of reduced stripping dictates their construction.

5. When curves are necessary in haulroads, superelevation must be designed into the curves. Excessive superelevation is to be avoided since trucks rounding a curve slippery from rain, ice or overwetting can slide inward and possibly overturn. Overly 'supered' curves result in excessive weight and wear on the inside tires.

6. Often curves are constructed to provide an access road into a mining bench from a steeply inclined haul road. To prevent the inside (and lower) side of this superelevated access curve from being at a steeper gradient than the main haul road, it is necessary to reduce (flatten) the center line grade of the curve. The inside grade should not be allowed to exceed the main road grade.

If enough room is available, the inside gradient of the curve should in fact be flatter than the main road grade to compensate for the increased rolling resistance. To accomplish this the design of a transition spiral is necessary.

7. Curves in the flat haul portion just as the trucks are leaving the shovel are quite critical. Due to the centrifugal forces induced by the curve, spill rock is thrown to the outside. Where possible, the return lane should be on the inside of the curve to avoid spill falling in the path of returning trucks and damaging tires. This can be accomplished by the use of crossovers to change traffic from right hand to left hand or vice versa in the necessary area. Adequate warning lights must be used at night to insured the safety at the crossovers. The costs of the warnings are small compared to savings in tire costs.

8. Waste dumps should be designed for placement at a two percent upgrade. This is done for the following reasons:

(a) The increase in dump height and volume occurs with little increase in haul speed or fuel cost. Because of the rapid dump volume increase, the haul distance is reduced.
(b) Better drainage on dumps.
(c) Some additional safety is afforded drivers backing up for dumping.
(d) If eventual dump leaching is planned, the water distribution is less expensive.

9. Within the mining areas, roads are built of the country rock at hand and are surfaced with the best material available within a reasonable haul distance. In the case of using something other than environmental rock to surface roads within the ore zone, double handling costs as well as ore dilution must be considered.

10. Main roads into the pit are usually planned for extended time of use and will justify more expenditure for subbase compaction and surfacing than temporary access roads.

11. If intended for use as a haul road, engineering layout should precede construction of even the shortest road or ramp. Mine survey crews should place desired stakes for initial cuts and fills and grade stakes including finish grade stakes.

12. When shovels are working in coarse, sharp rock faces, loading should be stopped periodically to allow fine material to be brought in and used to cap the surface of the loading area. Similar activity should be performed on waste dumps.

13. Constant attention to haul road surfaces is necessary. Soft spots, holes, 'washboard' areas, etc. should be repaired as soon as possible. Repairs usually consist of digging out the incompetent road material and replacing it with more desirable rock.

14. Grading of roads often results in a buildup of windrows on road edges. These narrow the roads and place sharp rocks in a position to damage tire sidewalls. Windrow buildup should be removed by loader or careful grader application.

15. Balding or grading of roads and dumps should be done when possible at a time when traffic can be moved to other areas. Many tires have been damaged by trucks driving through windrows created by graders assigned to improve roads and thereby reduce tire costs.

16. Maintenance of haul roads is equally important to good haulage costs as are design and construction. As more tires are damaged in shovel pits and dump areas than on actual haul roads, cleanup around an operating shovel is often assigned to the haulroad rather than the loading function. Road maintenance, to be successful, must have responsible supervision assigned to this task alone.

4.10 STRIPPING RATIOS

Consider the orebody shown in Figure 4.124 which has the shape of a right circular cylinder.

It outcrops at the surface and extends to a depth h. The volume of the contained ore is expressed by

$$V_o = \pi r^2 h \tag{4.15}$$

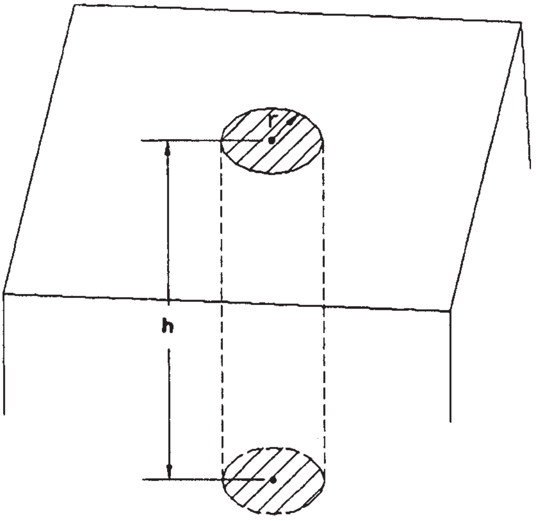


Figure 4.124. Cylindrical orebody.

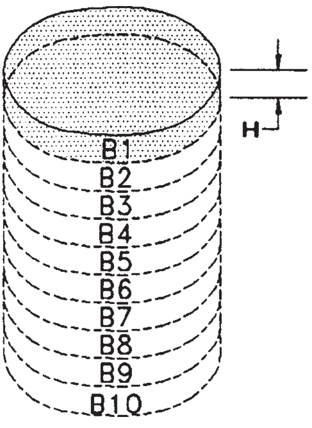


Figure 4.125. Cylindrical orebody mined as a sequence of constant diameter and thickness benches.

where r is the ore radius and h is the ore thickness. In concept, at least, one could remove the ore as a single plug and just leave the remaining hole. In practice, however, the orebody is first divided up into a series of benches of thickness H (Fig. 4.125). The volume of each ore bench B_i is

$$V_b = \pi r^2 H \tag{4.16}$$

In this case it will be assumed that each bench exactly satisfies the required annual production. Hence the pit would increase in depth by one bench per year. The surrounding waste rock has been assumed to have high strength so that these 90° pit walls can be safely achieved and maintained. In this mining scheme no waste is removed.

In reality, vertical rock slopes are seldom achieved except over very limited vertical heights. It is much more common to design using an overall slope angle Θ. As can be seen in Figure 4.126 the shape of the mined space changes from a right circular cylinder to a truncated right circular cone. The height of the truncated portion of the cone is

$$\Delta h = r \tan \Theta \tag{4.17}$$

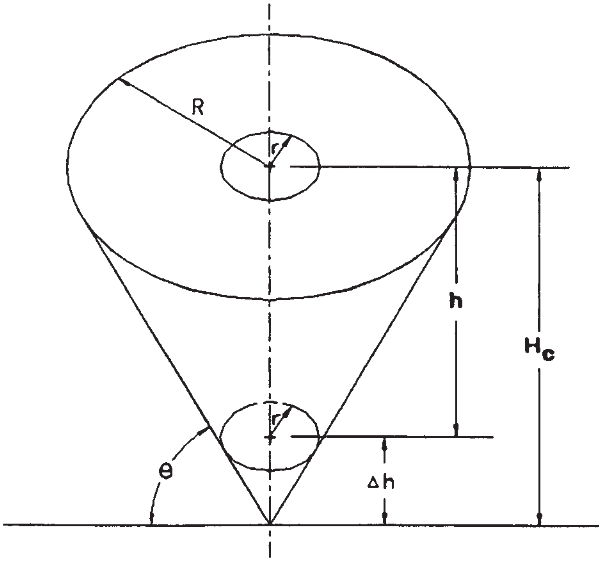


Figure 4.126. The cylindrical orebody mined via a conical pit.

where Θ is the overall slope angle. The height H_c of the cone which includes the orebody is then

$$H_c = h + \Delta h = h + r \tan \Theta \tag{4.18}$$

The base radius R of the circumscribed cone is

$$R = \frac{H_c}{\tan \Theta} = \frac{h}{\tan \Theta} + r \tag{4.19}$$

Using the volume formula for a right circular cone

$$V_{rcc} = \frac{1}{3} A_{bc} H_c \tag{4.20}$$

where A_{bc} is the base area of the cone, H_c is the height of the cone, and V_{rcc} volume of the cone, one can find the following volumes:

Truncated tip

$$V_{tip} = \frac{1}{3}\pi r^2 \Delta h \tag{4.21}$$

Fully circumscribed cone

$$V = \frac{1}{3}\pi R^2 H_c \tag{4.22}$$

Mined volume (ore + waste)

$$V_m = V - V_{tip} = \frac{1}{3}\pi R^2 H_c - \frac{1}{3}\pi r^2 \Delta h \tag{4.23}$$

Volume of waste

$$V_w = V_m - \pi r^2 h \tag{4.24}$$

One of the ways of describing the geometrical efficiency of a mining operation is through the use of the term 'stripping ratio'. It refers to the amount of waste that must be removed to release a given ore quantity. The ratio is most commonly expressed as

$$\text{SR} = \frac{\text{Waste (tons)}}{\text{Ore (tons)}} \tag{4.25}$$

however a wide variety of other units are used as well. In strip coal mining operations for example the following are sometimes seen

$$\text{SR} = \frac{\text{Overburden thickness (ft)}}{\text{Coal thickness (ft)}}$$

$$\text{SR} = \frac{\text{Overburden (yd}^3\text{)}}{\text{Coal (tons)}}$$

The ratio of waste to ore is expressed in units useful for the design purpose at hand. For this example, the ratio will be defined as

$$\text{SR} = \frac{\text{Waste (volume)}}{\text{Ore (volume)}} \tag{4.26}$$

Note that if the waste and ore have the same density, then Equation (4.25) and Equation (4.26) are identical.

If the volumes (or tons) used in the SR calculation correspond to those (cumulatively) removed from the start of mining up to the moment of the present calculation then the overall stripping ratio is being calculated. For this example the overall stripping ratio at the time mining ceases is

$$\text{SR (overall)} = \frac{V_\text{w}}{V_\text{o}} = \frac{V_m - \pi r^2 h}{\pi r^2 h} \tag{4.27}$$

On the other hand a stripping ratio can also be calculated over a much shorter time span. Assume that during year 5, X_o tons of ore and X_w tons of waste were mined. The stripping ratio for year 5 is then

$$\text{SR (year 5)} = \frac{X_\text{w}}{X_\text{o}}$$

This can be referred to as the instantaneous stripping ratio where the 'instant' in this case is 1 year.

If at the end of year 4, $X_{\text{o}4}$ tons of ore and $X_{\text{w}4}$ tons of waste had been mined then the overall stripping ratio up to the end of year 5 is

$$\text{SR (overall to end of year 5)} = \frac{X_{\text{w}4} + X_\text{w}}{X_{\text{o}4} + X_\text{o}}$$

Obviously the 'instant' could be defined as a longer or shorter time period. If during a given day the mine moves 5000 tons of waste and 2000 tons of ore, the instantaneous stripping ratio (for that day) is

$$\text{SR (instantaneous)} = \frac{5000}{2000} = 2.5$$

The determination of final pit limits as will be described in detail in Chapter 5, involves the calculation of a pit limit stripping ratio to be applied to a narrow strip at the pit periphery.

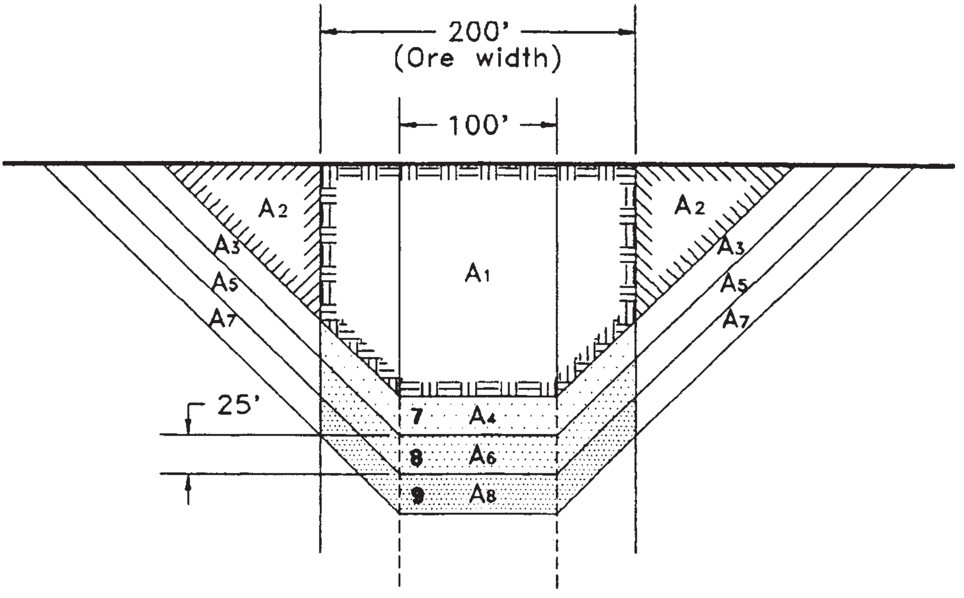


Figure 4.127. Section for stripping ratio calculations.

To illustrate this concept consider the simple cross-section shown in Figure 4.127. It will be assumed that:

– the pit is deepened in bench height increments of 25 ft;
– the minimum pit width is 100 ft;
– overall slope angle is 45°
– the density of the ore and waste is the same;
– the ore is of constant grade.

The original pit on this section (Fig. 4.127), consists of 6 benches and has a depth of 150 ft. The area of ore A_o is

$$A_o = A_1 = 200 \times 100 + 50 \times 150 = 27,500\,\text{ft}^2$$

The area of waste A_w is

$$A_w = 2A_2 = 100 \times 100 = 10,000\,\text{ft}^2$$

The overall stripping ratio SR (overall) is

$$\text{SR (overall)} = \frac{A_w}{A_o} = \frac{10,000}{27,500} = 0.36$$

Deepening of the pit by one bench (bench 7) requires the removal of $2A_3$ of waste. The amount of ore uncovered is A_4

$$A_4 = 100 \times 25 + 100 \times 25 = 5000\,\text{ft}^2$$

$$2A_3 = 125 \times 125 - 100 \times 100 = 5625\,\text{ft}^2$$

The instantaneous stripping ratio is

$$\text{SR (instantaneous)} = \frac{5625}{5000} = 1.125$$

The overall stripping ratio with bench 7 removed is

$$\text{SR (overall)} = \frac{15,625}{32,500} = 0.48$$

With mining of bench 8, another 5000 ft^2 of ore (A_6) is removed. This requires the stripping of

$$2A_5 = (150)^2 - (125)^2 = 6875\,\text{ft}^2$$

of waste. The instantaneous stripping ratio is

$$\text{SR (instantaneous)} = \frac{6875}{5000} = 1.375$$

The overall stripping ratio is

$$\text{SR (overall)} = \frac{22,500}{37,500} = 0.60$$

For bench 9:

$$A_8 = 5000\,\text{ft}^2$$
$$2A_7 = (175)^2 - (150)^2 = 8125\,\text{ft}^2$$

$$\text{SR (instantaneous)} = \frac{8125}{5000} = 1.625$$
$$\text{SR (overall)} = \frac{30,625}{42,500} = 0.72$$

As can be seen in this simple example, with each cut, the same amount of ore 5000 ft^2 must pay for an increasing amount of waste. The overall stripping ratio is less than the instantaneous value. There becomes a point where the value of the ore uncovered is just equal to the associated costs with the slice. This would yield the maximum pit on this section. Assume that in this case the breakeven stripping ratio is 1.625. Then the final pit would stop with the mining of bench 9. Through pit deepening, the walls of the pit are moved away or 'pushed back' from their original positions. The term 'push-back' is used to describe the process by which the pit is deepened by one bench.

4.11 GEOMETRIC SEQUENCING

There are several ways in which the volume of Figure 4.126 can be mined. As before, the first step in the process is to divide the volume into a series of benches (see Fig. 4.128). If a single bench is mined per year then the ore production would remain constant while both the total production and the stripping ratio would decrease. This would lead to a particular cash flow and net present value.

For most mining projects, a large amount of waste mining in the early years of a project is not of interest.

An alternative mining geometry is shown in Figure 4.129 in which a number of levels are mined at the same time. The overall geometry looks much like that of an onion.

An initial 'starter-pit' is first mined. In this example, the pit bottom extends to the edge of the orebody and the slope angle of the starter pit is the same as that of the final pit (Θ).

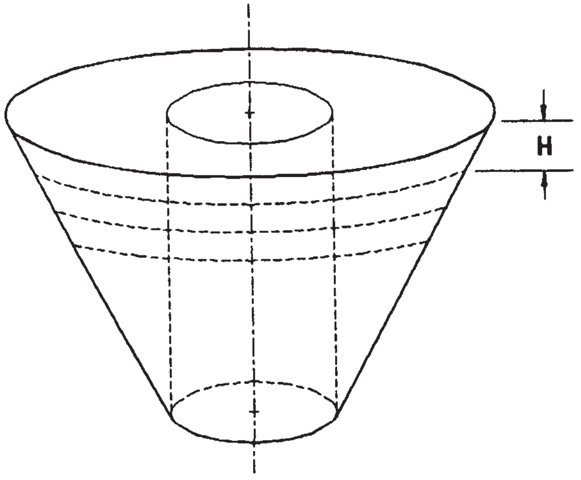


Figure 4.128. Sequential geometry 1 (Fourie, 1992).

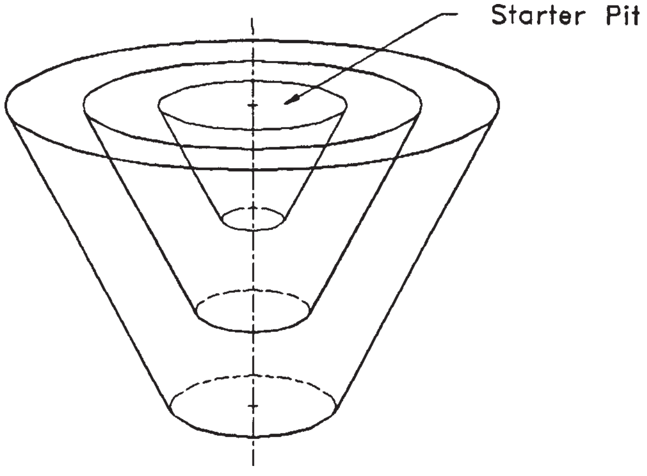


Figure 4.129. Sequential geometry 2 (Fourie, 1992).

In theory one could then slowly ‘eat-away’ at the sides and bottom of this starter pit until the final pit geometry is achieved. There are practical limits however on the minimum size ‘bites’ which can be considered both for planning and execution. The ‘bites’ in surface mining terms are called *push-backs* or phases. For modern large pits the minimum push-back distance (thickness of the bite) is of the order of 200 to 300 ft. For smaller pits it can be of the order of 100 to 200 ft. In this particular example the push-backs result in the pit being extracted in a series of concentric shells. The amount of material (ore and waste) contained in each shell is different. Hence for a constant production rate there might be x years of ore production in shell 1, y years of production in shell 2, etc. Eventually there will be a transition in which mining is conducted in more than one shell at a given time.

Sequencing within a pit shell and between shells becomes important. To this point simple concentric shells have been considered. The next level of complication is to split the overall pit into a number of sectors such as shown in Figure 4.130. Each sector (I → V) can be

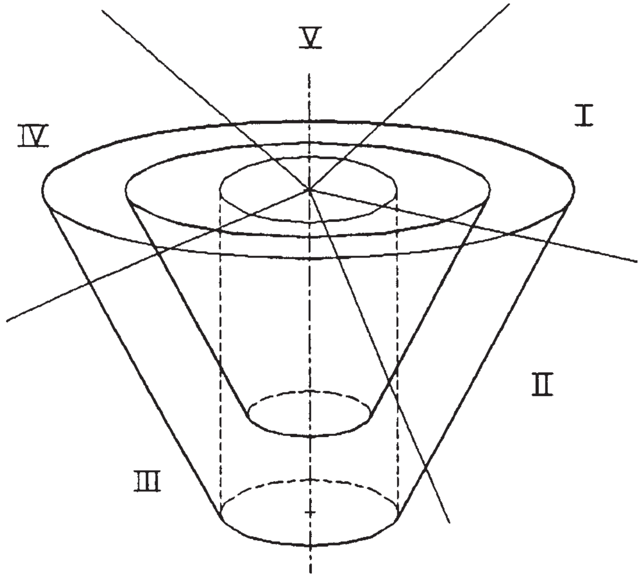


Figure 4.130. Sequential geometry 3 (Fourie, 1992).

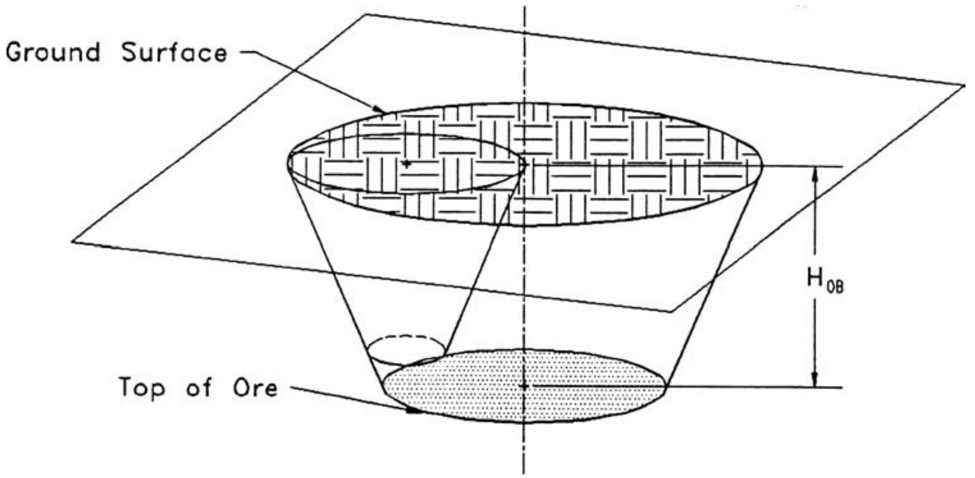


Figure 4.131. Sequential geometry 4 (Fourie, 1992).

considered as a separate production or planning unit. A natural basis for dividing the pit this way is due to slope stability/design considerations.

It has been assumed that the orebody outcrops (is exposed) at the surface. If this is not the case, such as is shown in Figure 4.131, then a preproduction or stripping phase must be first considered.

Due to cash flow considerations a variety of aspects enter:

- desire to reach the ore as quickly as possible,
- requirement to expose enough ore to maintain the desired plant production,
- combination of higher grade ore at greater depth versus lower grade at shallower depth.

The geometry-sequencing decisions then become even more complex.

It has been assumed for the sake of simplicity that the 'ore' is of one quality. Generally the values will vary in *X–Y–Z* space. The same is true for rock quality. Hence new dimensions are added to an already complex overall mine geometry-sequencing problem. Furthermore, the 'simple' addition of a haulage road to provide additional access can have a major effect on mine geometry and economics.

4.12 SUMMARY

In summary, pit geometry at any given time is influenced by many factors. Obviously the overlying material must be removed prior to removing that underlying. A certain operating space is needed by the equipment for efficiently removing the rock. The slope materials largely dictate the slope angles which can be safely used. In addition the sequencing of these geometries is extremely important so that the desired economic result (revenue and costs) is realized. Production rates, ore reserves and mine life are often highly price dependent. Hence mining geometry is a dynamic rather than static concept. To evaluate the many individual possibilities and combinations of possibilities, the computer has become invaluable.

The planning engineer must fully understand the basic geometric components which are combined to yield the overall pit geometry at any time in the life of the mine.

REFERENCES AND BIBLIOGRAPHY

Anonymous. 1978. Road design halts runaway vehicles. *Coal Age* 83(7): 144–147.

Anonymous. 1982. Better haul roads speed operations. *Coal Age* 83(3): 96–99.

Anonymous. 2007. Stabilizing influence: With worker safety and overall productivity at stake, mines are paying more attention than ever to slope stability concerns. E/MJ. 208(5): 60–63. June.

Armbrust, J.C. 1988. Morenci mine-blasting and dewatering of the 4050 drop cut. *SME Annual Meeting Jan. 25–28, 1988*. Soc. of Mining Engineers: preprint No. 88–146.

Atkinson, T. 1992. Design and layout of haul roads. *SME Mining Engineering Handbook*. 2nd Edition (H.L. Hartman, editor) 2: 1334–1342. Littleton, CO: SME.

Atkinson, T., & G. Walton. 1983. Design and layout of haul roads for surface mines. In: *Surface Mining and Quarrying, papers presented at the 2nd Int. Surface Mining and Quarrying Symposium, Oct. 4–6*, 1983. London: Instn. Min. Met.

Barksdale, R.D. (editor) 1991. *The Aggregate Handbook*: 11-1 to 11-23. Washington, D.C.: National Stone Association.

Bauer, A. 1984. Wall control blasting in open pits. In: *Rock Breaking and Mechanical Excavation* (P. Baumgartner, editor) CIM SpecialVolume 30: 3–10. The Canadian Institute of Mining and Metallurgy.

Beale, G., Luther, A., & J. Foster. 1997. Depressuring the pit wall at Sleeper and at the Mag pit. Mining Engineering. 49(11): 40–46.

Booth, P.A., & E.C.F. Hamman. 2007. Saprolites, Structures and Slope Angles – Applying Site-Specific Geotechnical and Mining Knowledge to Achieve the Final Design, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 25–33.

Brawner, C.O. 1969. Three big factors in stable slope design. *Mining Engineering* 21(8): 73–77.

Caccetta, L., Giannini, L.M., & S. Carras. 1986. The optimum design of large open pit mines. In: *Aus. IMM/IR Aust. Newman Combined Group Large Open Pit Mining Conference* (J.R. Davidson, editor): 195–200. Australasian Inst. of Mining and Metallurgy.

Call, R.D. 1986. Cost-benefit design of open pit slopes. In: *1st Open Pit Mining Symposium, Antofagasta, Chile, October 1986*: 1–18.

Call, R.D., Savely, J.P., & D.E. Nicholas. 1977. Preliminary data collection for pit slope design. *Mining Engineering* 29(4): 45–47.

*CANMET. *Pit Slope Manual*. CANMET Publications, 562 Booth Street, Ottawa, Ontario, K1A 0G1, Canada.

Carson, A.B. 1961. *General Excavation Methods*: 42–66. NewYork: McGraw-Hill.

Carter, R.A. 2006. Hybrid system solves GPS problems at Morenci: Terralite XPS uses a dual-signal setup to overcome GPS service interruptions in deep open-pit mines. E/MJ. 207(4): 46–52. May.

Caterpillar 1991. *Caterpillar Performance Handbook*. 22nd Edition.

Cawood, F.T., & T.R. Stacey. 2006. Survey and geotechnical slope monitoring considerations. JSAIMM. 106(7): 495–501.

Chironis, N.P. 1978. How to build better haul roads. *Coal Age* 83(1): 122–128.

Chiwaye, H.T., & T.R. Stacey. 2010. A comparison of limit equilibrium and numerical modeling approaches to risk analysis for open pit mining. JSAIMM. 110(10): 571–580.

Coates, D., McRorie, K., & J. Stubbins. 1963. Analysis of pit slides in some incompetent rocks. *Trans AIME* 226(3): 94–101.

Coates, D.F., Gyenge, M., & J.B. Stubbins. 1965. Slope stability studies at Knob Lake. In: *Proceedings 3rd Canadian Symp. on Rock Mech., Univ. of Toronto, Mines Branch*: 35–46. Queens Printer.

Couzens, T.R. 1979. Aspects of production planning: Operating layout and phase plans. In: *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 217–232. SME.

Dagdelen, K. 2005. Advances in pit slope monitoring and management systems. E/MJ. 206(1): 20–22. January/February.

Decision Making in the Mineral Industry. 1971. CIM Special Volume 12: 339–343.

Denby, B., McClarnon, D., & D. Schofield. 1996. The application of virtual reality in the modeling of surface mining operations. In Surface Mining 1996 (H.W. Glen, editor): 85–92. SAIMM.

Dight, P.M. 2006. Pit wall failures on 'unknown' structures. JSAIMM. 106(7): 451–458.

Dight, P.M. 2011. Slope stability Rock stress: The known unknowns. Mining Magazine. December. pp. 39–40.

Dight, P.M., & C. Windsor. 1986. The current state of the art of pit design and performance. In: *Aus. IMM/IE Aust. Newman Combined Group Large Open Pit Mining Conference, October 1986*: 285–294. Australasian Institute of Mining and Metallurgy.

Dillon, U., & G. Blackwell. 2003. The use of a geographic information system for open-pit mine development – Technical Note. CIM Bulletin 96(1069): 119–121.

East, D.R. 2000. Tailings dam failures –Why do they continue to occur? Mining Engineering. 52(12): 57–61.

Fishler, S.V. 1991. Personal communication. Harnischfeger Corporation.

Fourie, G.A. 1992. Open pit planning and design – basic concepts. In: *SME Mining Engineering Handbook*. 2nd Edition (H.L. Hartman, editor): 1274–1278. Littleton, CO: SME.

Gatzoubares, M.A. 2009. Evaluation of three-dimensional laser scanning and photogrammetry systems for terrestrial surveying and deformation monitoring of rock faces and highwalls in mining operations. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 59–70.

Gill, T. 1999. Planning optimal haul road routes using *Express*. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 375–384. Colorado School of Mines: CSM.

Goodbody, A. 2011. Slope stability: Monitoring movement. Mining Magazine. December. pp. 31–38.

Goodyear. 1992. Personal communication.

Goshtasbi, K., Ataei, M., & R. Kalatehjary. 2008. Slope modification of open pit wall using a genetic algorithm—case study: southern wall of the 6th Golbini Jajarm bauxite mine. JSAIMM. 108(10): 651–656.

Halls, J.L. 1970. The basic economics of open pit mining. In: *Planning Open Pit Mines* (P.W.J. Van Rensburg, editor). Johannesburg: SAIMM.

Hanna, T.M., Azrag, E.A., & L.C. Atkinson. 1994. Use of an analytical solution for preliminary estimates of ground water inflow to a pit. Mining Engineering. 46(2): 149–152.

Hardwick, W.R. 1958. *Open-Pit Mining Methods and Practices at Chino Mines Division, Kennecott Copper Corp., Grant County, N. Mex.* U.S. Bureau of Mines IC 7837.

Hardwick, W.R., & M.M. Stover. 1960. *Open-Pit Copper Mining Methods and Practices, Copper Cities Division, Miami Copper Co., Gila Count, Arizona*. U.S. Bureau of Mines IC 7985.

* For more information see the end of the references.

Hekmat, A., Osanloo, M., & P. Moarefvand. 2011. Investigating the effect of different block dimensions on the economic life of open pit mines. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 287–295.

Hoek, E. 1970a. Estimating the stability of excavated slopes in open cast mines. *Trans IMM* 79(10): A109–A132.

Hoek, E. 1970b. Design charts for excavated slopes. Contributions to the discussion of Section 6 – Planning Open Pit Mines. In: *Planning of Open Pit Mines* (P.W.J. Van Rensburg, editor): 295–302. Johannesburg: SAIMM.

Hoek, E., & J.S. Bray. 1977. *Rock Slope Engineering* (Revised Second Edition): 402 pp. London: Inst. of Mining and Metallurgy.

Hoek, E. & D.L. Pentz. 1970. Review of the role of rock mechanics research in the design of opencast mines. In: *Proceedings of the 9th Commonwealth Mining and Metallurgical Congress, Vol. 1, Mining and Petroleum Technology*: 389–412. London: IMM.

Holman, P. 2007. Management in action – Haul road management: Making the grade. Mining Magazine. December. pp. 22–26.

Hunter, G. 2007. Management in action – Slope stability: Checking slope stability. Mining Magazine. September. pp. 63–67.

Hunter, G. 2009. Laser scanning: There's more than one option. E/MJ. 210(6): 33–34. July/August.

Hustrulid, W.A., McCarter, M.K., & D. Vanzyl. (editors) 2000. Slope Stability in Surface Mining, SME, Littleton, CO, USA.

Jenkins, P., & C. Seymour. 2009. Mining rock mass models: 3D evaluation of the geotechnical environment for optimal project design and planning. AusIMM Journal. Issue 6. Dec.

Jennings, J.E., & R.A.L. Black. 1963. Factors affecting the angle of slope in open-cast mines. Paper presented at the Annual Meeting of AIME, Dallas, Texas, Feb. 24–28, 1963: preprint 63 A011, and *Trans. of the AIME* 226(3): 42–53.

Kaufman, W.W., & J.C. Ault. 1977. *Design of Surface Mine Haulage Roads – A Manual*. USBM IC 8758.

Kim, Y.C., Coates, D.F., & T.J. O'Neil. 1977. A formal approach to economic analysis of pit slope design. In: *15th APCOM, Brisbane, Aust.*: 405–413.

Kliche, C.A. 1999. Rock Slope Stability. SME. Littleton, Colorado. 253 pp.

Kose, H., Aksoy, C.O., Gönen, A., Kun, M., & T. Malli. 2005. Economic evaluation of optimum bench height in quarries. JSAIMM. 105(2): 127–135.

Lane, K.S. 1961. Field slope charts for stability studies. In: *5th Intern. Conf. Soil Mech and Foundation Eng.* 2: 651–655.

Lato, M., Hutchinson, D.J., & M. Diederichs. 2009. Geomechanical feature extraction and analysis of LiDAR data: Iron Ore Company of Canada mine. CIM Magazine. December 2009/January 2010. pp. 82–93.

Long, A.E. 1964. Problems in designing stable open-pit mine slopes. *CIM Bulletin* 57(7): 741–746.

Martin, D.C., & D.R. Piteau. 1977. Select berm width to contain local failures. *E/MJ* 178(6): 161–164.

McCarter, M.K. 1976. Monitoring stability of highwaste dumps. Presented at the 1976 SME-AIME Meeting, Denver, Sept. 1–3, 1976: preprint no. 76-AO-328: 23 pp.

McWilliams, J.R. 1959. *Mining Methods and Costs at the Anaconda Company Berkeley Pit, Butte, Mont*. USBM IC 7888.

Miller, G.G., Stecklin, G.L., & J.J. Labra. 1983. *Improved haul road berm design*. U.S. Bureau of Mines IC 8947.

Mines Branch. 1972. *Tentative Design Guide for Mine Waste Embankments in Canada*. Technical Bulletin TB 145, March 1972.

Moore, P. 2009. Management in action – Slope stability: The harder they fall. Mining Magazine. November. pp. 36–39.

Moss, A.S.E., & O.K.H. Steffen. 1978. Geotechnology and probability in open-pit mine planning. In: *Proceedings of the 11th Commonwealth Mining and Met. Congress*: 543–550.

MSHA (Mine Safety and Health Administration) 1992. Safety and health standards applicable to surface metal and nonmetal mining and milling operations, Part 56. Code of Federal Regulations (CFR 30), Revised as of July 1, 1992.

Nalsmith, N.A., & S.D.N. Wessels. 2005. Management of a major slope failure at Nchanga open pit, Chingola, Zambia. JSAIMM. 105(9): 619–626.

Narendranathan, S. 2009. Probabilistic slope design and its use in iron ore pit optimisations. Iron Ore Conference 2009. Perth, WA. July 27–29. pp. 289–300.

Nichols, H.L., Jr. 1956. *Modern Techniques of Excavation*: 8-8 to 8-23. Princeton, NJ: Van Norstrand.

Noon, D., & N. Harries. 2007. Slope Stability Radar for Managing Rock Fall Risks in Open Cut Mines, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 93–97.

Nutakor, D. 2012. Mitigation of the Cplus failure in the Q7 cut at Rio Tinto's Bingham Canyon Mine. Mining Engineering. 64(6): 107–111. June.

Prawasono, A., Nicholas, D., Young, J., Belluz, N., Schmelter, S.C., & S. Widodo. 2007. The Geometry of the Heavy Sulfide Zone – A Challenge for Mine Planning and Geotech at Grasberg, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 99–105.

Ramsey, R.H. 1944. How Cananea develops newest porphyry copper. *E/MJ* 145(12): 74–87.

Ramsey, R.J. 1945. New Cananea operation now in high gear. *E/MJ*146(9): 72–78.

Reed, A. 1983. Structural geology and geostatistical parameters of the Afton Copper-Gold Mine, Kamloops, B.C. *CIM Bulletin* (856)76(8): 45–55.

Riese, M. 1993. *Specification Sheet for the Bucyrus-Erie Model 155 Shovel*.

Ritchie, A.M. 1963. Evaluation of rockfall and its control. *Highway Research Record* 17: 13–18.

Robertson, A.M. 1971. The interpretation of geological factors for use in slope theory. In: *Proc. Symposium on Planning Open Pit Mines*. Johannesburg. 55–70. Amsterdam: A.A. Balkema.

Ross-Brown, D.M. 1979. Pit limit slope design – analytical design. In: *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 161–184. SME-AIME.

Savely, J.P. 1986. Designing a final wall blast to improve stability. Presented at the SME Annual Meeting, New Orleans, March 2–6, 1986. SME: preprint no. 86–50.

Seegmiller, B.L. 1976. Optimum slopes for future open pit mines: Howto obtain them using a rock mechanics approach. Presented at 1976 Fall SME-AIME Meeting, Denver, CO, Sept. 1–3, 1976: preprint 76-F-326.

Seegmiller, B.L. 1978. How to cut risk of slope failure in designing optimum pit slopes. *E/MJ Operating Handbook of Surface Mining* 2: 92–98.

Seegmiller, B.L. 1979. Pit limit slope design – general comments, data collection remedial stability measures. In: *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 161–184. SME-AIME.

Seelye, E.E. 1945. *Design Data Book for Civil Engineers*. Volume I. NewYork: JohnWiley.

Sharon, R., Rose, N., & M. Rantapaa. 2005. Design and development of the northeast layback of the Betze-Post open pit. Pre-print Paper 05-009. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.

Soderberg, R.L. & R.A. Busch 1977. Design guide for metal and nonmetal tailings disposal. Bureau of Mines Information Circular 8755.

Spang, R.M. 1987. Protection against rockfall – stepchild in the design of rock slopes. In: *Proceedings 6th Int. Conference on Rock Mechanics*: 551–557. Rotterdam: A.A. Balkema.

Stacey, T.R. 2006. Considerations of failure mechanisms associated with rock slope instability and consequences for stability analysis. JSAIMM. 106(7): 485–493.

Steffen, O.K.H., Holt, W., & V.R. Symons. 1970. Optimizing open pit geometry and operational procedure. In: *Planning Open Pit Mines* (P.W.J. Van Rensburg, editor): 9–31. Johannesburg: SAIMM/A.A. Balkema.

Stewart, R.M., & B.A. Kennedy. 1971. The role of slope stability in the economics, design and operation of open pit mines. In: *Stability in Open Pit Mining. SME*.

Strachan, C. 2001. Tailings dam performance from USCOLD incident-survey data. Mining Engineering. 53(3): 49–53.

Tamrock 1978. *Handbook of Surface Drilling and Blasting*: 14, J.F. Olán Oy.

Taylor, D.W. 1948. *Fundamental of Soil Mechanics*. New York: John Wiley.

Taylor, J.B. 1971. Incorporation of access roads into computer-generated open pits.

Taylor, P.F., & P.A. Hurry. 1986. Design, construction and maintenance of haul roads. In: *The Planning and Operation of Open-Pit and Strip Mines* (J.P. Deetlefs, editor): 137–150. Johannesburg: SAIMM.

Thompson, R. 2009. Haul road design considerations. E/MJ. 210(5): 36–43. June.

Thompson, R. 2010. Mine haul road design and management best practices for safe and cost-efficient truck haulage. Paper 10-030. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.

Thompson, R. 2011. Building better haulroads. E/MJ. 212(5): 48–53. June.

Thompson, R. 2011. Building better haul roads: Designing for structural strength. Paper 11-029. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Thompson, R. 2011. Haul road improvement – Healthier haulage. Mining Magazine. November. pp. 32–36.
Thompson, R.J., & A.T. Visser. 1999. Designing and managing unpaved opencast mine haul roads for optimum performance. Paper 99-090. SME Annual Meeting. Mar 1–3. Denver, CO.
Thompson, R.J., & A.T. Visser. 2000. The functional design of surface mine haul roads. JSAIMM. 100(3): 169–180.
Williamson, O.C. 1987. Haul road design for off-highway mining equipment. *World Mining Equipment* 12(3/4): 24–26.
Winkle, R.F. 1976a. Development and maintenance of haulroads in openpit mines. Paper presented at the AIME Annual Meeting Las Vegas, Nevada, Feb. 22–26, 1976. Preprint No. 76-AO-5.
Winkle, R.F. 1976b. Guides to design and control of efficient truck and shovel operations in open-pit mines. M.S. Thesis, Univ. of Arizona.

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Slope Stability in Surface Mining (W.A. Hustrulid, M.K. McCarter and D. Van Zyl, editors) 2000 SME, Littleton, CO, USA.

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REVIEW QUESTIONS AND EXERCISES

1. Summarize the steps in the development of an open pit mine.
2. What "geometries" are involved in pit development?
3. Define or describe the following terms:

– bench height	– working bench
– crest	– cut
– toe	– safety bench/catch bench
– bench face angle	– double benches
– back break	– berms
– bench floor	– angle of repose
– bench width	

4. What are the purposes of safety benches?
5. What is the width of a safety bench?
6. What is the function of a safety berm?
7. What are some typical guidelines for a safety berm?
8. Discuss some of the aspects that enter into bench height selection?
9. Draw a sequence of three benches. Label the crest, toe, bench face angle, bench width and bank width.
10. If the bench face angle is 69°, the bench width is 30 ft and the bench height is 45 ft, determine the overall slope angle.
11. What happens if due to poor excavation practices the actual bench face angle is 66° instead? There are two possibilities to be considered.
12. Discuss the significance of Figure 4.3.
13. What are the purposes of safety benches?
14. Discuss the pro's and con's of double benching. Discuss the practical actions required to create double benches.
15. In actual surface mining operations, what happens to the "catch" benches created during the general operations? How does this affect their function? What actions might need to be taken?
16. Call has suggested the catch bench geometries shown in Figure 4.7 and in Table 4.1. Draw the geometry suggested for a 30m (double bench) height. What would be the corresponding final slope?
17. Discuss the pro's and con's of higher versus lower bench heights.
18. The dimensions for a Bucyrus Erie (BE)9 yd3 shovel are given in Figure 4.9. Some comparable dimensions for larger BE shovels are given below:

Dipper capacity (yd^3)	Dimension						
	A	B	D	E	G	H	I
12	22′–3″	40′–9″	37′–6″	50′–0″	34′–0″	7′–1″	39′–3″
20	30′–0″	55′–0″	50′–6″	65′–5″	44′–9″	9′–3″	50′–1″
27	30′–6″	57′–0″	48′–6″	67′–6″	44′–3″	6′–3″	52′–0″

Based upon the shovel geometries, what would be an appropriate maximum bench height for each?

19. In the largest open pit operations today, the BE 495 or P&H 4100 shovels are being used. Obtain dimensions similar to those given in the table in problem 18 for them.
20. Using Figure 4.10, what would be the reach height for a shovel with a 56 yd^3 dipper capacity?
21. The dipper capacities which are provided for a particular shovel model, normally are based upon material with a density of 3000 lbs/yd^3. What is done if the particular shovel model is used to dig coal? To dig magnetite?
22. Summarize the steps which would be followed in considering the appropriate bench geometry.
23. Compare the digging profile for the shovel shown in Figure 4.9 with a bench face drawn at a 65° angle. What is your conclusion?
24. Summarize the discussion of ore access as presented in section 4.3.
25. A drop cut example has been presented based upon the 9 yd^3 shovel. Rework the example assuming that the 27 yd^3 capacity is used instead. Find the minimum and maximum cut widths. Select an appropriate Caterpillar truck to be used with this shovel (use the information on their website).
26. What would be the volume of the ramp/drop cut created in problem 25?
27. Discuss the different aspects which must be considered when selecting the ore access location.
28. Figure 4.23 shows the situation where the ramp construction is largely in ore. Assume that the diameter of the orebody is 600 ft, the ramp width is 100 ft, the road grade is 10% and the bench height is 40 ft. Determine the approximate amount of ore removed in Figure 4.24.
29. Determine the amount of waste that would be removed if the ramp in Figure 4.24 was entirely constructed in waste.
30. Summarize the factors associated with the ramp location decision.
31. Figures 4.21 through 4.25 show the addition of a ramp to a pit. In this new case the orebody is assumed to be 600 ft in diameter, the road is 100 ft wide and the grade is 10%. Using AutoCad redo the example. What is the final ramp length? As shown, a flat portion 200 ft in length has been left between certain ramp segments.
32. Once the access to the newpit bottom has been established, discuss the three approaches used to widen the cut.
33. Summarize the steps used to determine the minimum required operating room when making parallel cuts.
34. What is the difference between the single and double spotting of trucks? Advantages? Disadvantages?
35. Redo the cut sequencing example described in section 4.4.6 assuming a cut width of 150 ft. If the shovel production rate is 50,000 tpd, how long would it take to exhaust the pushback?
36. A pit is enlarged using a series of pushbacks/laybacks/expansions. What would be the minimum and maximum cut widths using the 27 yd^3 shovel and Caterpillar model 789 trucks, assuming single pass mining? Work the problem assuming both single and double spotting of trucks.
37. Discuss the advantages/disadvantages of double spotting.
38. Redo the example described in section 4.4.5 assuming Caterpillar 993 trucks and the P&H 4100 shovel. The bench height is 50 ft and the bench face angle is 70°.
39. Assume that five 50 ft high benches are being worked as a group (Figure 4.51). Using the data from Problem 38, what would the working slope angles be when the working bench is at level 2?

40. For the slopes identified as "critical" in Table 4.2, what types of actions should be taken?
41. How might the design slope angles change during the life of a mine?
42. How do the expected stress conditions in the walls and floor of the pit change as the pit is deepened. Make sketches to illustrate your ideas.
43. What might happen at the pit bottom for the situation shown in Figure 4.61?
44. What are the four most common types of slope failure? Provide a sketch of each.
45. Assume that a 50 ft high bench is as shown in Figure 4.63. The layering goes through the toe. Assume that the following apply:
    $\phi = 32°$
    c = 100 kPa
    $\rho = 2.45$ g/cm^3
    Bench face angle = 60°
    Bedding angle = 20°
    What is the safety factor?
46. In section 4.6.3 it was determined that for the given conditions the safety factor was 1 (F = 1) for a slope of height 140 ft and a slope angle of 70°. Determine the minimum slope angle if the slope height is 200 ft instead.
47. Redo problem 46 if the required safety factor is 1.2. (See Figure 4.65).
48. Redo problem 46 assuming the presence of a tension crack of length 20 ft. You should consider the two extreme cases: (a) Crack dry; (b) Crack filled with water.
49. What is the safety factor for a slope assuming that the following apply:
    H = 200 ft
    Density = 165 lb/ft^3
    c = 825 lb/ft^2
    $Z_o = 50$ ft
    $H_w = 100$ ft
    i = 40°
    $\beta = 30°$
    $\phi = 30°$
50. A waste dump 500 ft high is planned. It is expected that the face angle will be 50°, the density is assumed to be 1.4 g/cm^3, the cohesion = 0 and the friction angle is 30°. Will it be stable?
51. What techniques might be used to obtain values for ϕ and c appropriate for the materials making up slopes?
52. Discuss the effect of slope wall curvature (in plan) on stability.
53. A discussion of the Afton copper/gold mine has been presented. Check the literature/Internet to see what eventually happened to the slopes.
54. In Figure 4.72, if the entire pit consisted of Structural Region B, what would happen to the North and East walls?
55. Assume that the conical pit shown in Figure 4.74 has a bottom radius of 100 ft. Redraw the figure showing the volume-slope dependence.
56. Discuss the pros' and con's of the different ways of representing bench positions on a plan map.
57. Redo the example in Figure 4.88 using AutoCad.
58. List some of the important questions that must be answered when siting a road.
59. The steps in the design of a spiral road inside the wall have been presented in section 4.8.2. Redo the example assuming a grade of 8%, a bench height of 40 ft, a bench face angle of 65° and a crest to crest dimension of 80 ft. The road width is 100 ft.
60. Redo problem 58 using AutoCad.

61. Redo the example in section 4.8.3 using AutoCad.
62. Redo the example in section 4.8.4 using AutoCad.
63. Why is it generally desirable to avoid switchbacks in a pit? Under what conditions might it become of interest?
64. If the conditions are such that a switchback becomes interesting, what should the planner do?
65. Assume that a conical pit has a depth of 500 m and the overall slope angle is 38°. Consideration is being given to adding a second access. The road would have a width of 40 m. How much material would have to be mined? Follow the approach described in section 4.8.5.
66. Using the data for the Caterpillar 797 haulage truck, answer the following questions:
    a. Load distribution of the front and rear tires.
    b. Contact area for an inflation pressure of 80 psi.
    c. What wheel loading should be used in the road design?
67. In the design and construction of a mine haulage road, what layers are involved? Describe each one starting at the lowest layer.
68. Describe the considerations in determining layer thickness.
69. How do you include the effect of dual-wheel loading?
70. What is meant by the California Bearing Ratio? How is it determined?
71. Suggest a haulage road width for two-way traffic involving Cat 797 trucks.
72. The following road cross-section dimensions apply at a particular mining operation which uses Komatsu 930 trucks:
    - safety berm width = 3.5 m
    - truck width = 7.3 m
    - space between trucks = 5.0 m
    - width of drainage ditch = 2.0 m
    - overall road width = 25 m
    - bench face angle = 75°

    Draw the section. How well does this design correspond to the "rules"?
73. Why are well-designed roads important?
74. How is a runaway ramp constructed?
75. What would happen to the road section shown in Figure 4.117 with the passage of a fully-loaded Cat 793 haulage truck? Be as specific as possible.
76. Assume that the sub-grade is a compact sand-clay soil and that you have the following construction materials: well-graded crushed rock, sand, well-graded gravel. What thicknesses would you recommend when using the Cat 793 trucks? Assume all of the materials have the same cost.
77. What is meant by poorly graded material?Well graded material?
78. In the design of a straight haulage road segment, what major factors should be considered?
79. Would the road design shown in Figure 4.118 apply for the Cat 793 truck? Why or why not?
80. What is the design rule for roadway width assuming two-way traffic?
81. List the steps to be followed in selecting a road design width.
82. What is meant by cross-slope? What are the rules involved? Why is it used?
83. If you were a haulage truck driver, what effect would cross-slope have on you?
84. What is meant by centrifugal force? How does it apply to road design?
85. What special design procedures must be applied to curves? What is meant by super-elevation?
86. What is the effect of rain and snow on super-elevated road segments?

87. Assume a curve radius of 50 ft (inner edge of pavement) and two-lane traffic. The truck is a Cat 793. What should be the minimum width? What should be the super-elevation assuming a curve speed of 15 mph?
88. What are transition zones?
89. Summarize the rules regarding the need for berms/guardrails.
90. What are the two principal berm designs in common use today?
91. On an elevated road involving Cat 793 haulage trucks, suggest a parallel berm design.
92. Summarize the rules presented by Couzens regarding haulage road gradients.
93. Summarize the practical road building and maintenance tips offered by Winkle.
94. Define what is meant by:
    a. Stripping ratio
    b. Overall stripping ratio
    c. Instantaneous stripping ratio
95. Indicate some of the different units used to express stripping ratio for different mined materials.
96. Redo the example shown in Figure 4.127 assuming that:
    – ore width = 400 ft
    – bench height = 40 ft
    – minimum pit width = 200 ft
    – original pit depth = 280 ft (7 benches)

    Calculate the appropriate values for SR(instantaneous) and SR(overall) for the mining of benches 8, 9, and 10.
97. What is meant by the term push-back?
98. Discuss the concepts of geometric sequencing. What is meant by a “phase”?
99. Discuss the different possibilities involved in pit sequencing.

CHAPTER 5

Pit limits

5.1 INTRODUCTION

The time has now come to combine the economics introduced in Chapter 2 with the mineral inventory developed in Chapter 3 under the geometric constraints discussed in Chapter 4 to define the mineable portion of the overall inventory. The process involves the development and superposition of a geometric surface called a pit onto the mineral inventory. The mineable material becomes that lying within the pit boundaries. A vertical section taken through such a pit is shown in Figure 5.1. The size and shape of the pit depends upon economic factors and design/production constraints. With an increase in price the pit would expand in size assuming all other factors remained constant. The inverse is obviously also true. The pit existing at the end of mining is called the 'final' or the 'ultimate' pit. In between the birth and the death of an open-pit mine, there are a series of 'intermediate' pits. This chapter will present a series of procedures based upon:

(1) hand methods,
(2) computer methods, and
(3) computer assisted hand methods

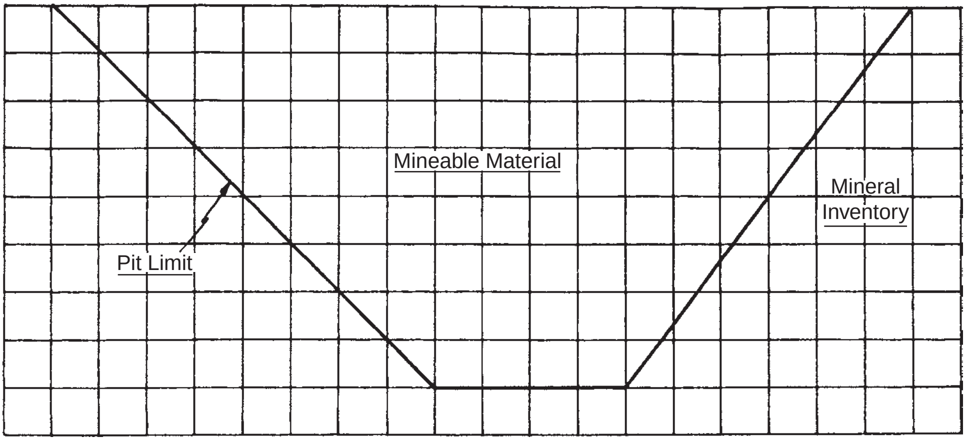


Figure 5.1. Superposition of a pit onto a mineral inventory.

for developing pit limits. Within the pit are found materials of differing value. Economic criteria are applied to assign destinations for these materials based on their value (i.e. mill, waste dump, leach dump, stock pile, etc.). These criteria will be discussed. Once the pit limits have been determined and rules established for classifying the in-pit materials, then the ore reserves (tonnage and grade) can be calculated. In Chapter 6, the steps required to go from the ore reserve to production rate, mine life, etc. will be presented.

5.2 HAND METHODS

5.2.1 *The basic concept*

Figure 5.2 shows an idealized cross-section through an orebody which outcrops at the surface and dips to the left at 45°. There are distinct physical boundaries separating the ore from the over- and under-lying waste. The known ore extends to considerable depth down dip and this will be recovered later by underground techniques. It is desired to know how large the open-pit will be. The final pit in this greatly simplified case will appear as in Figure 5.3. The slope angle of the left wall is 45°. As can be seen a wedge of waste (area A) has been removed to uncover the ore (area B). The location of the final pit wall is determined by examining a series of slices such as shown in Figure 5.4.

For this example the width of the slice has been selected as 1.4 units (u) and the thickness of the section (into the page) as 1 unit. Beginning with strip 1 the volumes of waste (V_w) and ore (V_o) are calculated. The volumes are:

Strip 1:

$$V_{w1} = 7.5u^3$$

$$V_{o1} = 5.0u^3$$

The instantaneous stripping ratio (ISR) is defined as

$$\text{ISR}_1 \text{ (instantaneous)} = \frac{V_{w1}}{V_{o1}} \tag{5.1}$$

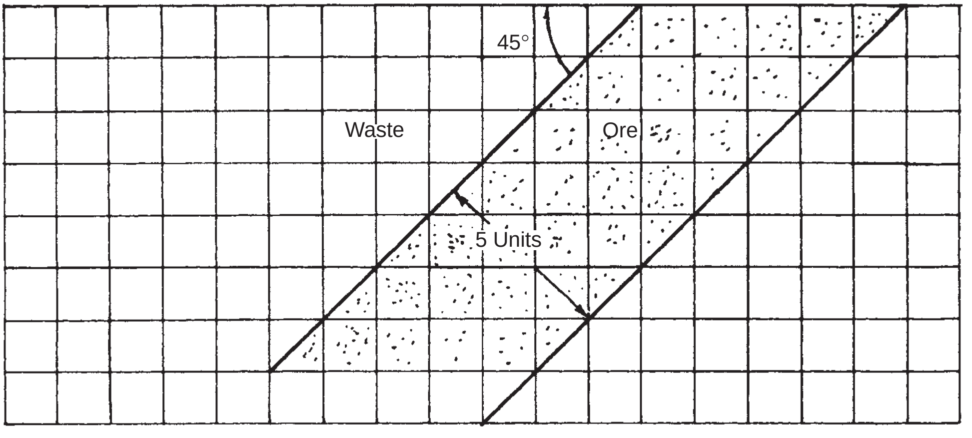


Figure 5.2. Cross-section through an idealized orebody.

Hence

$$ISR_1 = 1.50$$

Assuming that the net value from selling one unit volume of ore (that money remaining after all expenses have been paid) is \$1.90 and the cost for mining and disposing of the waste is \$1/unit volume, the net value for strip 1 is

$$NV_1 = 5.0 \times \$1.90 - 7.5 \times \$1 = \$2.00$$

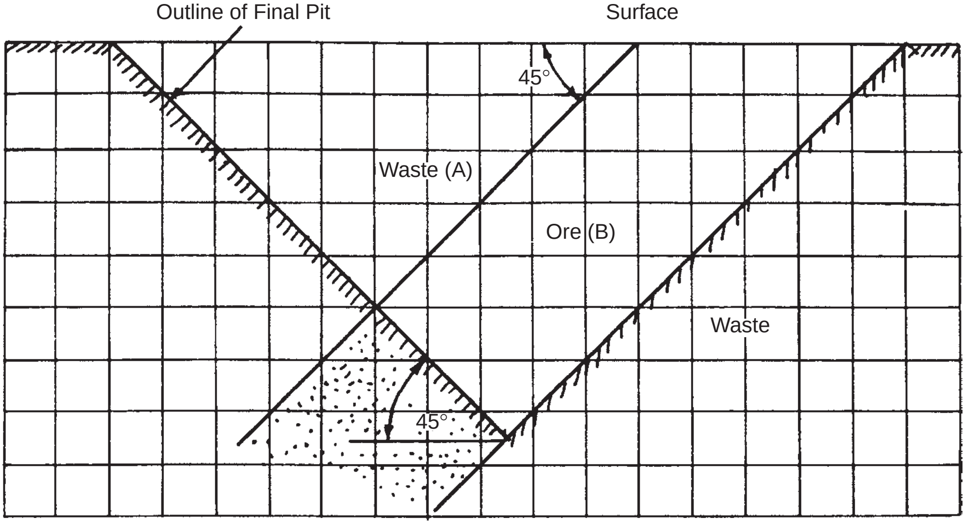


Figure 5.3. Diagrammatic representation of the final pit outline on this section.

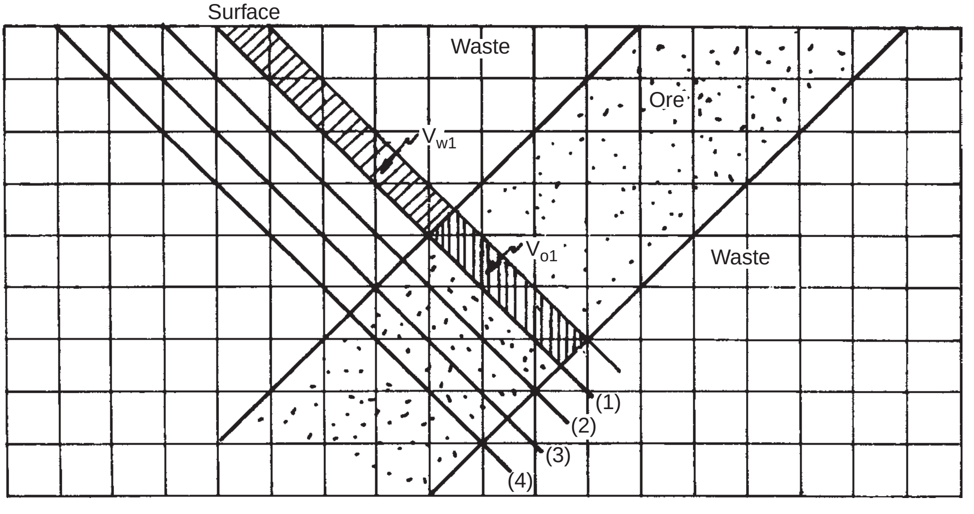


Figure 5.4. Slices used to determine final pit limits.

If the process is now repeated for strips 2, 3 and 4, the results are as given below:

Strip 2:

$$V_{w2} = 8.4u^3$$
$$V_{o2} = 5.0u^3$$
$$ISR_2 = 1.68$$
$$NV_2 = 5.0 \times \$1.90 - 8.4 \times \$1 = \$1.10$$

Strip 3:

$$V_{w3} = 9.45u^3$$
$$V_{o3} = 5.0u^3$$
$$ISR_3 = 1.89$$
$$NV_3 = 5.0 \times \$1.90 - 9.45 \times \$1 = \$0.05 \cong \$0$$

Strip 4:

$$V_{w4} = 10.5u^3$$
$$V_{o4} = 5.0u^3$$
$$ISR_4 = 2.10$$
$$NV_4 = 5.0 \times \$1.90 - 10.5 \times \$1 = -\$1.00$$

As can be seen, the net value changes from (+) to (−) as the pit is expanded. For strip 3, the net value is just about zero. This pit position is termed 'breakeven' since the costs involved in mining the strip just equal the revenues. It is the location of the final pit wall. The breakeven stripping ratio which is strictly applied at the wall is

$$SR_3 = 1.9$$

Since the net value of 1 unit of ore is \$1.90 and the cost for 1 unit of waste is \$1, one can mine 1.9 units of waste to recover 1 unit of ore (Fig. 5.5).

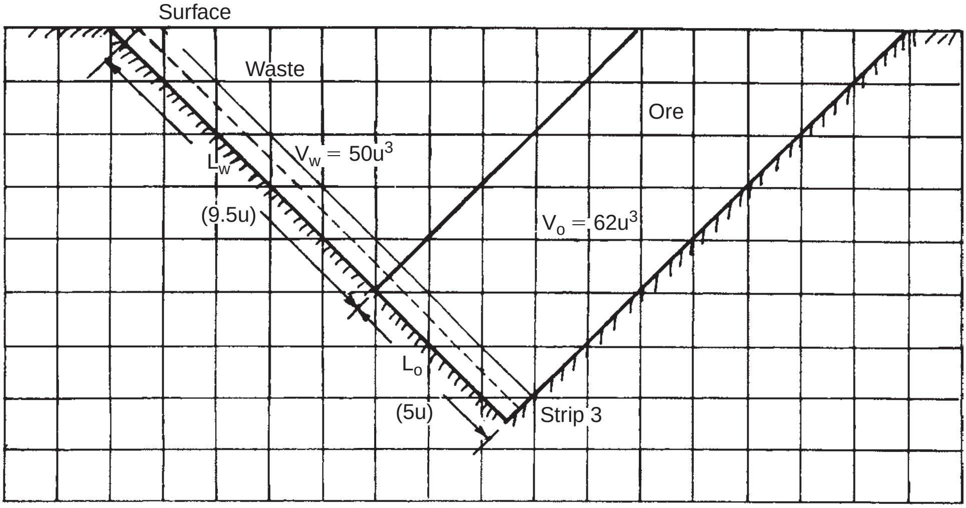


Figure 5.5. Final pit outline showing ore-waste distribution.

The overall stripping ratio (OSR) for this section is calculated as

$$\text{OSR} = \frac{\text{Waste area}}{\text{Ore area}} = \frac{A}{B} \tag{5.2}$$

In this case

$$\text{Waste area} = A = 50\text{u}^3$$
$$\text{Ore area} = B = 62\text{u}^3$$

hence

$$\text{OSR} \cong 0.8$$

This is compared to the instantaneous stripping ratio at the pit boundary

$$\text{ISR (pit limit)} = 1.9$$

The OSR must always be less than the ISR (pit limit).

The net value for the section (assuming unit thickness) is

$$\begin{aligned}\text{NV} &= \text{Ore area} \times \text{Net ore value} - \text{Waste area} \times \text{Waste removal cost}\\ &= B \times \$1.90 - A \times \$1 = 62 \times \$1.90 - 50 \times \$1 = \$68\end{aligned}$$

Whereas the net value is zero at the pit limit, it is positive for the overall section.

In this example the quantities, costs and revenues were all expressed in terms of volumes. Since the strip width and thickness is the same in both ore and waste, the final pit limit in this situation is that position where the *length* of waste (L_w) is just equal to 1.9 times the *length* of ore (L_o) as measured along the midline of the mined strip.

Often the costs/revenues are expressed as a function of weight ($/ton). If the density of the ore and waste is the same then the ratio of lengths can still be used. If they are not, then the different densities must be included in the calculations.

As this chapter proceeds, more realistic geometries both for the pit and the orebody will be introduced. A gradual rather than sharp ore-waste transition will be included. As will be seen, even with these changes the following basic steps involved in determining pit limits remain the same:

1. A slice is selected.
2. The contained value is compared with the costs.
3. If the net value is positive, the pit can be expanded. If negative, the pit contracts.
4. The final pit position is where the net value of the slice is zero.

5.2.2 *The net value calculation*

In the previous section, the nature of the deposit was such that there was no ambiguity regarding what was meant by ore and waste. For many deposits however the distinction is much more subtle. The term ‘cutoff grades’ refers to grades for which the destination of pit materials changes. It should be noted that ‘grades’ were used rather than ‘grade’ since there may be several possible destinations. The simplest case would be that in which there are two destinations: the mill or the waste dump. One cutoff grade is needed. For many operations today there are three possible destinations: the mill, the leach dump and the waste dump. Each of the decisions

– mill or leach?
– leach or waste?

requires a cutoff grade. A definition of cutoff grade which is often used (Davey, 1979):

Cutoff grade = the grade at which the mineral resource can no longer be processed at a profit.

applies to the simple ore-waste decision. This will be used in developing the preliminary pit limits. The only destinations allowed are the waste dump or further processing. With this definition the net value of material as a function of grade must be determined. That grade for which the net value is zero is called the breakeven cutoff grade. This calculation will be illustrated using the example provided by Davey (1979) for copper. The copper ore is milled thereby producing a copper concentrate. This mill concentrate is shipped to a smelter and the resulting blister copper is eventually refined.

In this example the following will be assumed:

Mill recovery rate = 80%

Mill concentrate grade = 20%

Smelting loss = 10 lbs/st of concentrate

Refining loss = 5 lbs/st of blister copper

The steps of the net value computation are outlined below for an ore containing 0.55% copper. All of the costs and revenues will be calculated with respect to one ton of ore.

Step 1. Compute the amount of saleable copper (lbs/st of ore).

(a) Contained copper (CC) is

$$\mathrm{CC} = 2000\,\mathrm{lbs/st} \times \frac{0.55}{100} = 11.0\,\mathrm{lb}$$

(b) Copper recovered by the mill (RM) is

$$\mathrm{RM} = 11.0 \times \frac{80}{100} = 8.8\,\mathrm{lb}$$

(c) Concentration ratio (r). The ratio of concentration is defined as

$$r = \frac{\text{lbs Cu/st of concentrate}}{\text{lbs Cu recovered/st of ore}} \tag{5.3}$$

Since the mill product runs 20% copper there are 400 lb of copper contained in one ton of concentrate. One ton of ore contains 8.8 lb of recoverable copper. Hence

$$r = \frac{400}{8.8} = 45.45$$

This means that 45.45 tons of ore running 0.55% copper are required to produce 1 ton of concentrate running 20%.

(d) Copper recovered by the smelter (RS). The mill concentrate is sent to a smelter. Since the smelting loss is 10 lb/st of concentrate, the smelting loss (SL) per ton of ore is

$$\mathrm{SL} = \frac{10\,\text{lb/st of concentrate}}{45.45\,\text{tons of ore/st of concentrate}} = 0.22\,\mathrm{lb}$$

Thus the recovered copper is

$$\mathrm{RS} = 8.8 - 0.22 = 8.58\,\mathrm{lb}$$

(e) Copper recovered by the refinery (RR). The number of tons of ore required to produce one ton of blister copper is

$$\frac{2,000\,\text{lb/st of blister copper}}{8.58\text{ lb of copper/st of ore}} = 233.1$$

Since refining losses are 5 lb/st of blister copper, the refining loss (RL) per ton of ore is

$$\mathrm{RL} = \frac{5\,\text{lb of copper/st of blister copper}}{233\,\text{tons of ore/st of blister copper}} = 0.02\,\mathrm{lb}$$

Thus the recovered copper is

$$\mathrm{RR} = 8.58 - 0.02 = 8.56\,\mathrm{lb}$$

Step 2. Compute the gross value (*GV*) *for the ore* (*\$/st*). The copper price assumed for this calculation is \$1.00/lb. Furthermore there is a by-product credit for gold, molybdenum, etc. of \$1.77/st of ore. Thus the gross value is

$$\mathrm{GV} = 8.56 \times \$1 + \$1.77 = \$10.33$$

Step 3. Compute the associated total costs (*TC*) (*\$/st*).

(a) Production (operating) costs (PC) excluding stripping are:

Mining	\$1.00
Milling	\$2.80
General and administration (15% of mining and milling)	\$0.57
PC =	\$4.37

(b) Amortization and depreciation (A&D). This amount is charged against each ton of ore to account for the capital investment in mine and mill plant. If the total A&D is \$10,000,000 and overall ore tonnage is 50,000,000 tons, then this value would be \$0.20. In this particular case, 20% of the total production costs will be used.

$$\mathrm{A\&D} = 0.20 \times \$4.37 = \$0.87$$

(c) Treatment, refining and selling (TRS) cost.

– Shipment of mill concentrate to the smelter. Since transport costs \$1.40 per ton of concentrate, the cost per ton of ore is

$$\text{Concentrate transport} = \frac{\$1.40}{45.45} = \$0.03$$

– Smelting cost. Smelting costs \$50.00/st of concentrate. The smelting cost per ton of ore is

$$\text{Smelting} = \frac{\$50.00}{45.45} = \$1.10$$

– Shipment of the blister copper to the refinery. There is a transport cost of $50.00/st of blister copper involved. The cost per ton of ore becomes

$$\text{Blister transport} = \$50.00\frac{8.58}{2000} = \$0.21$$

– Refining cost. Refining costs $130.00/st of blister copper. The refining cost per ton of ore is

$$\text{Refining} = \$130.00\frac{8.58}{2000} = \$0.56$$

– Selling and delivery cost (S&D). The selling and delivery cost is $0.01/lb of copper. Since 8.56 lb are available for sale

$$\text{S\&D} = \$0.09$$

– General plant (GP) cost. These costs amount to $0.07/lb of copper. Hence the GP cost per ton of ore is

$$\text{GP} = \$0.07 \times 8.56 = \$0.60$$

– Total treatment cost is

$$\text{TRS} = \$2.59$$

(d) Total cost per ton of ore is

$$\text{TC} = \$7.83$$

Step 4. Compute net value per ton of ore. The net value is the gross value minus the total costs. Thus for an initial copper content of 0.55%, the net value becomes

$$\text{NV} = \text{GV} - \text{TC} = \$10.33 - \$7.83 = \$2.50$$

Step 5. Compute the net value for another ore grade. Steps 1 through 4 are now repeated for another copper content. In this case 0.35% Cu has been chosen. The by-product credit will be assumed to vary directly with the copper grade. Hence

$$\text{By-product credit} = \$1.77\frac{0.35}{0.55} = \$1.13$$

Assuming the recoveries and unit costs remain the same, the net value is −$0.30.

Step 6. Construct a net value – grade curve. The two points on a net value – grade curve which have been determined by the process outlined above

Point	Net value ($/st)	Grade (% Cu)
1	$2.50	0.55
2	−$0.30	0.35

are plotted in Figure 5.6. Assuming that the net value – % Cu relationship is linear it is possible to find an equation of the form $y = a + bx$ relating net value (y) to grade (x). The result is

$$y = -\$5.20 + \$14.0x$$

where y is the net value ($/st of ore) and x is the percent copper.

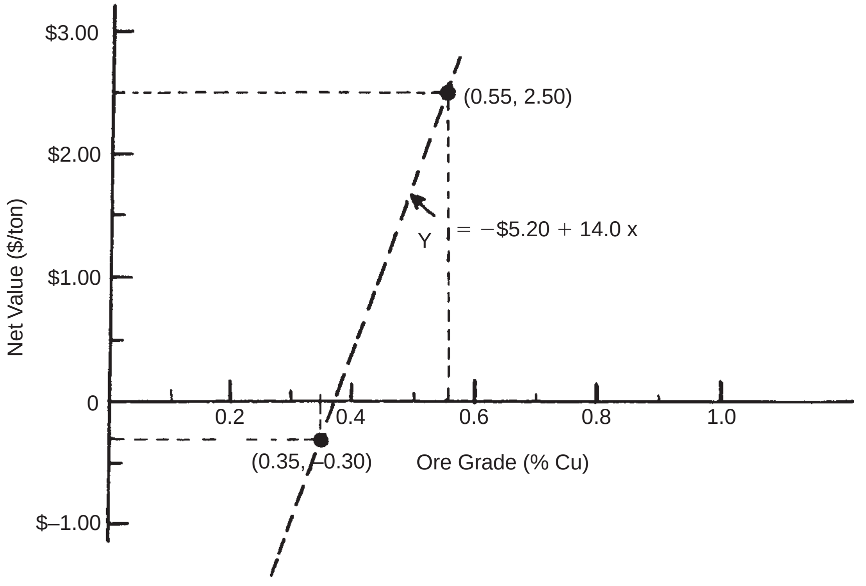


Figure 5.6. Net value – ore grade curve.

Step 7. Determine the breakeven cutoff grade (*for application at the pit limit*). The breakeven cutoff grade is defined as that grade for which the net value is zero. One can determine that point by inspecting Figure 5.6 or by solving the equation found in Step 6 for $y = 0$. One finds that

$$x\,(\text{breakeven}) = 0.37\%\,\text{Cu}$$

Step 8. Developing a stripping ratio – grade curve. The cutoff grade distinguishes that material which can be mined and processed with a net value greater than or equal to zero. Material with a zero net value cannot pay for any stripping. Thus it must be exposed at the surface or be overlying richer blocks which can pay for the required stripping. Assume that the cost for stripping 1 ton of waste is $1.00. Ore with a net value of $1.00 can pay for the stripping of 1 ton of waste. Ore with a net value of $2.00/ton can pay for the stripping of 2 tons of waste, etc. The stripping ratio axis has been added in Figure 5.7 to show this. The net value – grade equation

$$\text{NV} = -\$5.20 + \$14.00 \times (\%\,\text{Cu}) \tag{5.4}$$

can be modified to yield the stripping ratio (SR) – grade relationship

$$\text{SR} = -\$5.20 + \$14.00 \times (\%\,\text{Cu}) \tag{5.5}$$

For a grade of 0.55% Cu, the breakeven stripping ratio is

$$\text{SR}\,(0.55\%) = 2.5$$

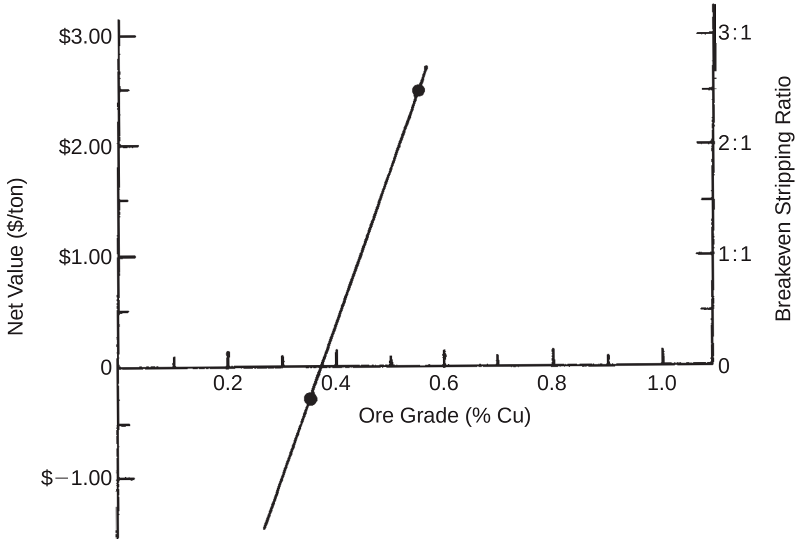


Figure 5.7. Net value and breakeven stripping ratio versus ore grade.

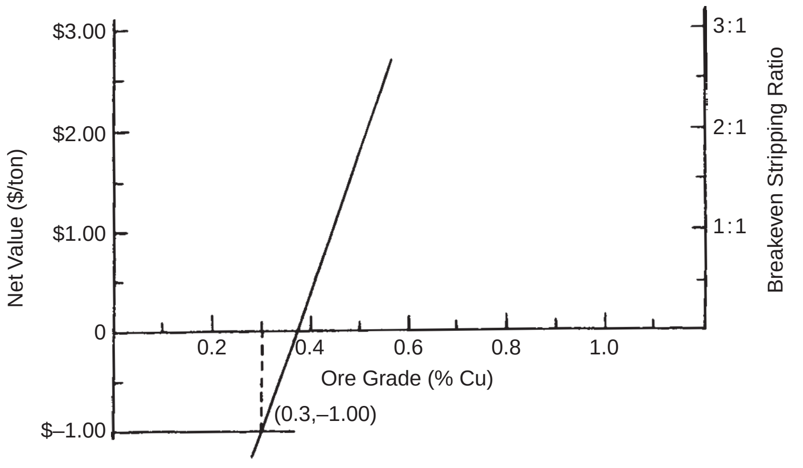


Figure 5.8. Addition of the minimum value portion.

This was as expected since the net value is \$2.50 and the stripping cost is \$1.00

$$\text{SR} = \frac{\$2.50}{\$1.00} = 2.50$$

Step 9. Presenting the final curves. The net value – grade curve should be completed by the addition of the cost of stripping line (SC). This is shown in Figure 5.8. It should be noted that no material can ever have a value less than that of waste. In this case

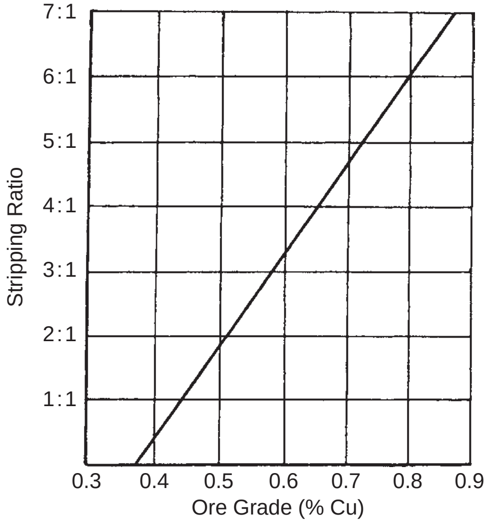


Figure 5.9. Stripping ratio – ore grade curve used for pit limit determinations (Koskiniemi, 1979).

the value of waste is −\$1.00. The horizontal line (NV = −\$1.00) and the NV-grade line (NV = −5.20 + 14.00 × (% Cu)) intersect at a grade of

$$\%\,\mathrm{Cu} = 0.30$$

For grades less than 0.30%, the material is considered as waste with respect to milling. Depending upon the economics, some other treatment process such as dump leaching may be possible. When using hand methods all material having grades less than the breakeven cutoff (0.37% in this case) is considered as waste. The final stripping ratio-grade curve is shown in Figure 5.9.

For the computer techniques discussed later, the portion of the curve lying between 0.30 and 0.37% Cu is also included.

5.2.3 *Location of pit limits – pit bottom in waste*

The application of this curve to locating the final pit wall positions will be illustrated using the vertical section (Fig. 5.10) taken through the block model. The basic process used has been presented by Koskiniemi (1979). Locating the pit limit on each vertical section is a trial and error process. It will be assumed that

– Pit slopes:
  Left hand side = 50°;
  Right hand side = 40°;
– Minimum width of the pit bottom = 100 ft;
– Material densities:
  Ore = 165 lb/ft^3;
  Waste rock = 165 lb/ft^3;
  Overburden = 165 lb/ft^3;

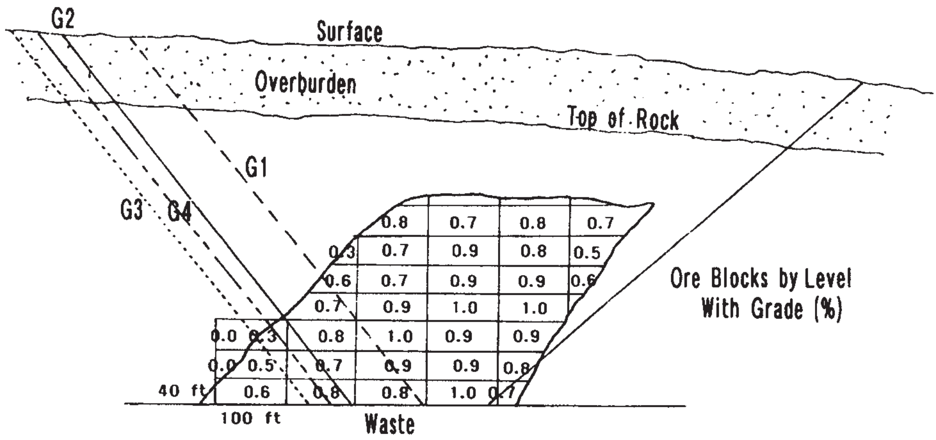


Figure 5.10. Pit limit determination with bottom in waste (Koskiniemi, 1979).

Table 5.1. Bottom in waste: pit limit guess #1 (Line G1).

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
130				
	296			
		30	0.6	18.0
		52	0.7	36.4
		52	1.0	52.0
		52	0.9	46.8
		52	0.8	41.6
130	296	238	$\bar{g} = 0.82$	194.8

SR (actual) $= \frac{130+296}{238} \cong 1.79:1$;
SR (allowable) $\cong 6.2:1$;
Conclusion: move to the left.

– Relative mining characteristics:
  Waste rock = 1;
  Overburden = 1;
– The stripping ratio – ore grade curve of Figure 5.9 applies;
– The pit bottom is at the ore-waste contact.

Overburden as defined here means soil, glacial till, gravel, highly weathered rock, etc. not requiring drilling and blasting prior to removal. Waste rock, on the other hand, does require drilling and blasting. The general procedure will be demonstrated with respect to the left-hand slope.

Step 1. A trial slope (guess #1) is drawn through the section. The lengths and grades are entered into a table such as Table 5.1. The purpose will be to obtain the average ore grade and stripping ratio along this line. The lengths can simply be scaled off the section with enough accuracy. The cutoff grade is 0.37% Cu.

Table 5.2. Bottom in waste: pit limit guess #2 (Line G2).

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
130				
	385			
		52	0.8	41.6
		52	0.7	36.4
		52	0.8	41.6
130	385	156	$\bar{g} = 0.77$	119.6

SR (actual) $= \frac{130+385}{156} \cong 33:1$;
SR (allowable) $\cong 5.6:1$;
Conclusion: move to the left.

Table 5.3. Bottom in waste: pit limit guess #3 (Line G3).

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
130				
	443			
		52	0.5	26.0
		52	0.8	41.6
130	443	104	$\bar{g} = 0.65$	67.6

SR (actual) $= \frac{130+443}{104} \cong 5.51:1$;
SR (allowable) $\cong 3.9:1$;
Conclusion: move to the right.

Table 5.4. Bottom in waste: pit limit guess #4 (Line G4).

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
130				
	435			
		52	0.7	36.4
		52	0.8	41.6
130	435	104	$\bar{g} = 0.75$	78.0

SR (actual) $= \frac{130+435}{104} \cong 5.43:1$;
SR (allowable) $\cong 5.4:1$;
Conclusion: final limit.

Step 2. The average ore grade is determined. The products of ore grade × ore length are determined ($l_{oi}g_{oi}$) and summed ($\sum(l_{oi}g_{oi})$). The sum of the ore lengths is found ($\sum l_{oi}$). The average ore grade $\bar{g}$ is found from

$$\bar{g} = \frac{\sum l_{oi}g_{oi}}{\sum l_{oi}} = 0.82\%\,\mathrm{Cu} \tag{5.6}$$

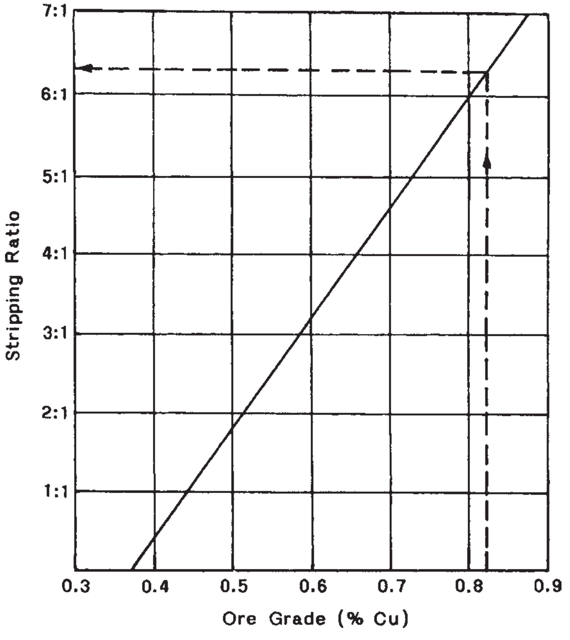


Figure 5.11. Allowable stripping ratio for a grade of 0.82%.

Step 3. The stripping ratio for this line is determined. This must be expressed in the same form as the stripping ratio – ore grade curve. In this case it is required to have tons of waste per ton of ore. Since the densities are all equal and the relative diggability of the overburden and the waste rock is the same, the stripping ratio is simply the ratio of the lengths.

$$\mathrm{SR} = \frac{\text{Length overburden} + \text{Length waste rock}}{\text{Length ore}} = \frac{l_{\mathrm{ob}} + l_{\mathrm{w}}}{l_{\mathrm{o}}}$$

$$\mathrm{SR} = \frac{130 + 296}{238} \cong 1.79 \tag{5.7}$$

Step 4. Determination of the allowable stripping ratio for average ore grade using the SR – ore grade curve (Fig. 5.11). In this case one finds that

$$\mathrm{SR}\ (\text{allowable}) \cong 6.2:1$$

Step 5. Comparison of the actual and allowable SR. Since the actual stripping ratio

$$\mathrm{SR}\ (\text{actual}) = 1.79:1$$

is much less than that allowable (6.2 : 1), the pit slope can be moved to the left.

Step 6. A new guess of the final pit slope location is made and the process repeated. This iteration process is continued until the actual and allowable stripping ratios are close.

Conclusion: Guess #4 is the final position of the left hand slope. Note that the block having grade 0.3 is considered waste since it is below cutoff.

Step 7. Determination of right-hand slope position. The same process is repeated for the right hand wall of the pit. The results are shown in Figure 5.10. The final pit bottom has a width of about 215 ft.

Table 5.5. Bottom in ore: pit limit guess #1 (Line G1).

(a) Left hand side

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
130				
	330			
		48	0.7	33.6
		52	0.8	41.6
		13	0.7	9.1
		39	0.9	35.1
		22	0.8	17.6
		50	0.8	40.0
130	330	224	$\bar{g} = 0.79$	177.0

SR (actual) $= \frac{130+330}{224} \cong 2.05$;
SR (allowable) $\cong 6:1$.

(b) Right hand side

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
143				
	283			
		48	0.6	28.8
	13			
		48	1.0	48.0
		70	0.9	63.0
		11	0.8	8.8
		57	0.9	51.3
		26	1.0	26.0
		39	1.0	39.0
		11	0.8	8.8
143	296	310	$\bar{g} = 0.88$	273.7

SR (actual) $= \frac{143+296}{310} \cong 1.42:1$;
SR (allowable) $\cong 7:1$;
Conclusion: the pit can be 'floated' considerable deeper.

If the waste/overburden have densities different from the ore, then the calculation of stripping ratio using simple length ratios does not work. The generalized stripping ratio calculation becomes

$$\mathrm{SR} = \frac{l_{ob}\rho_{ob} + l_w\rho_w}{l_o\rho_o} \tag{5.8}$$

where ρ_{ob} is the overburden density, ρ_w is the waste density, and ρ_o is the ore density. If the mineability characteristics of the overburden and waste rock are different, then the costs involved in their removal will also be different. It will be recalled that a single waste mining

Table 5.6. Bottom in ore: pit limit guess #2 (Line G2).

(a) Left hand side

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
139				
	626			
		52	0.5	26.0
		52	0.6	31.2
		52	0.6	31.2
		52	0.6	31.2
		52	0.6	31.2
		52	0.6	31.2
139	626	258	$\bar{g} = 0.58$	150.8

SR (actual) $= \frac{139+626}{258} \cong 2.97:1$;
SR (allowable) $\cong 2.95:1$.

(b) Right hand side

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
157				
	617			
		65	0.6	39.0
		61	0.6	36.6
		65	0.5	32.5
		61	0.5	30.5
		50	0.6	30.0
157	617	302	$\bar{g} = 0.56$	168.6

SR (actual) $= \frac{157+617}{302} \cong 2.56:1$;
SR (allowable) $\cong 2.6:1$;
Conclusion: guess #2 is close to the pit slope location.

cost was used in the development of the SR – grade curves. Assume that the given cost (C_w/ton) applies to waste rock and that the overburden removal cost is αC_w/ton. The factor α is the relative mining cost of the overburden to that of the waste rock. The stripping ratio formula can be further modified to

$$\mathrm{SR} = \frac{\alpha l_{ob}\rho_{ob} + l_w\rho_w}{l_o\rho_o} \tag{5.9}$$

If the cost/ton to remove the overburden is only half that of the waste rock then $\alpha = 0.5$. This factor changes the overburden into an equivalent waste rock. If the cost used to develop the SR – grade curve had been based on overburden, then one needs to convert waste rock into equivalent overburden.

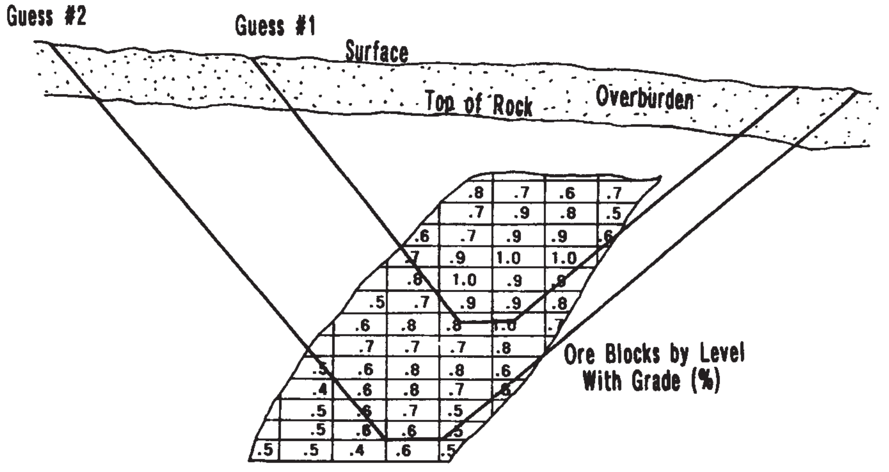


Figure 5.12. Final pit limits with bottom in ore.

5.2.4 *Location of pit limits – pit bottom in ore*

Figure 5.12 shows the case when the orebody continues to depth. Assume as before that

Left hand slope = 50°

Right hand slope = 40°

Minimum pit bottom width = 100 ft

Since the pit bottom is now in ore, the costs for stripping can be paid for by the ore in the strip along the pit bottom as well as that along the pit sides. A 50 ft wide allocation is made to both the right and left hand sides. The procedure is similar to that described with the pit bottom in waste. However, now both sides and the bottom must be examined at the same time. The procedure is as follows:

Step 1. Draw to scale a final pit profile using the appropriate left and right hand slopes as well as the minimum pit bottom on a piece of tracing paper. Superimpose this on the section. Guess an initial position.

Step 2. Calculate the average ore grades and stripping ratios for the left and right hand sides. Compare these to the allowable values. For simplicity it will be assumed that the densities and mineabilities of the materials involved are the same.

Because of the freedom of the pit to 'float' both vertically and horizontally, the iterative procedure can be quite time consuming.

5.2.5 *Location of pit limits – one side plus pit bottom in ore*

A third possible situation is one in which one of the pit slopes is in ore. In Figure 5.13 it will be assumed that the right hand slope follows the ore-waste contact, the left hand slope is at 50°, and a minimum pit bottom width is 100 ft. In this case the ore along the pit bottom

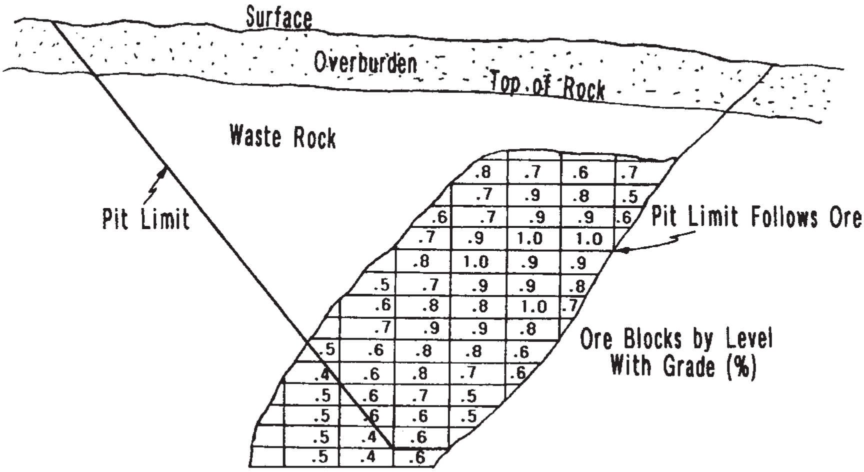


Figure 5.13. Final pit limits with right hand side in ore.

Table 5.7. Final pit limit location for one side plus pit bottom in ore (Fig. 5.13).

Length (ft)			Ore grade (g_{oi})	Ore length × ore grade ($l_{oi}g_{oi}$)
Overburden (l_{ob})	Waste (l_w)	Ore (l_{oi})		
130				
	626			
		43	0.5	12.5
		52	0.4	20.8
		52	0.6	31.2
		52	0.6	31.2
		52	0.4	20.8
		100	0.6	60.0
130	626	351	$\bar{g} = 0.53$	185.5

SR (actual) $= \frac{130+626}{351} \cong 2.15$;
SR (allowable) $\cong 2.21$;
Conclusion: this is the final pit location.

contributes to the cost of stripping the left wall. The approximate position of the final pit is shown superimposed on the figure. The corresponding calculation is given in Table 5.7.

5.2.6 *Radial sections*

The types of sections used depends upon the shape of the orebody. For the elongated orebody shown in Figure 5.14, transverse sections yield the best representation in the central portion. Along the axis of the orebody, a longitudinal section may be taken.

Transverse sections such as 1-1′, 2-2′, etc. in Figure 5.14 have been constructed parallel to one another and normal to the orebody axis. The influence of these sections is assumed to extend halfway to the neighboring sections. They are of constant thickness by construction

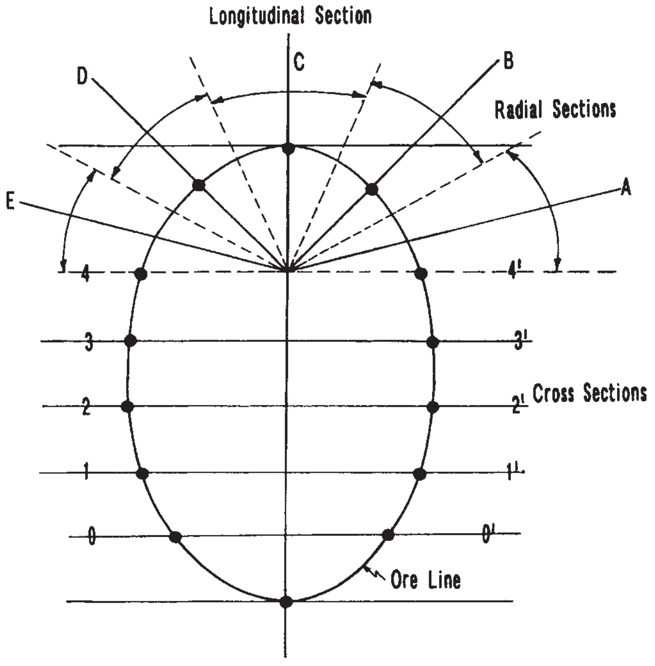


Figure 5.14. Plan view of orebody illustrating the different section types (Koskiniemi, 1979).

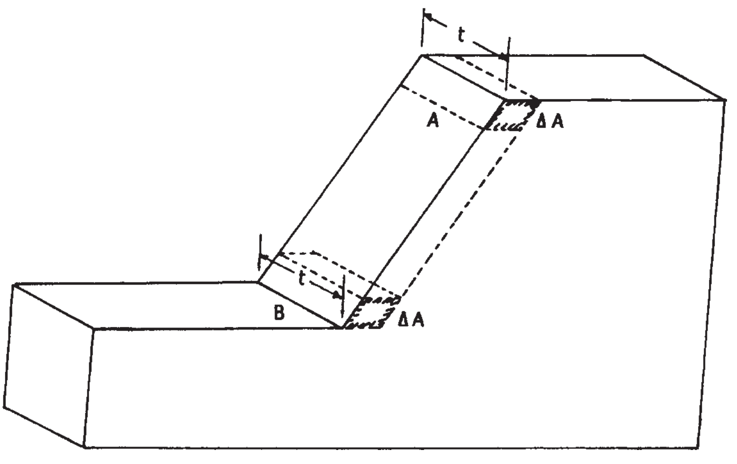


Figure 5.15. Isometric view of a parallel section.

(Fig. 5.15). A small face area (ΔA) at location A at the crest of the pit represents the same volume as the same area located at the toe (location B).

The pit location procedures described in Subsections 5.2.3 through 5.2.5 apply without modification to parallel cross-sections and longitudinal sections. As can be seen in Figure 5.14, radial sections are often needed to describe pit ends. For radial sections such

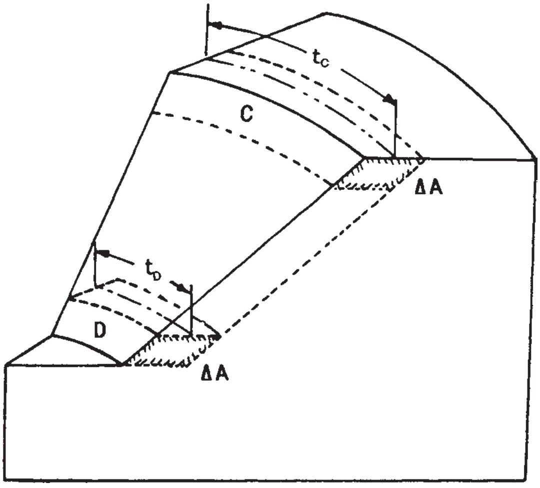


Figure 5.16. Isometric view of a radial section.

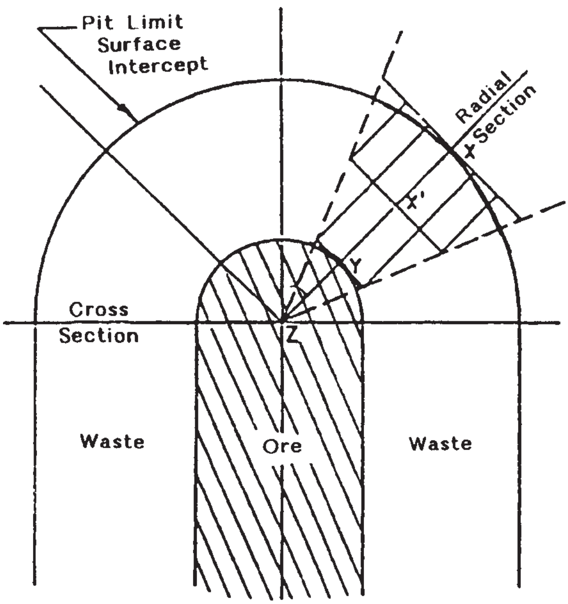


Figure 5.17. Plan representation showing the area of influence for a radial section (Koskiniemi, 1979).

as shown in Figure 5.16, the volume represented by an area ΔA at the crest (location C) is much greater than one at D due to the varying section thickness.

A modification in the procedure used to locate the final pit limit is required. This is accomplished through the development of a curve relating the apparent stripping ratio as measured on the radial section to the true stripping ratio.

Figure 5.17 is a plan view showing the region at the end of the pit in which radial sections are being used.

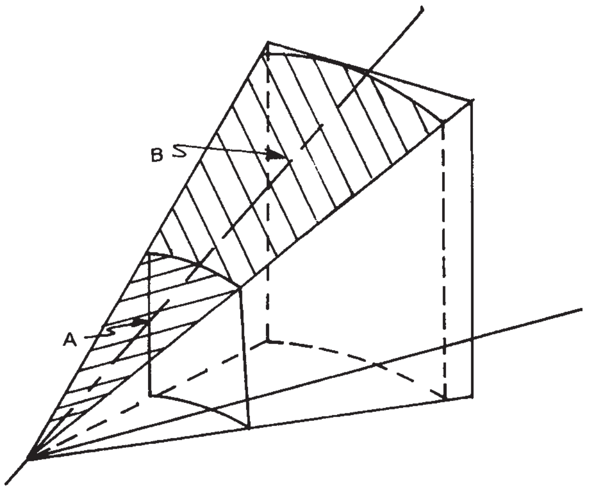


Figure 5.18. Isometric view of the radial sector.

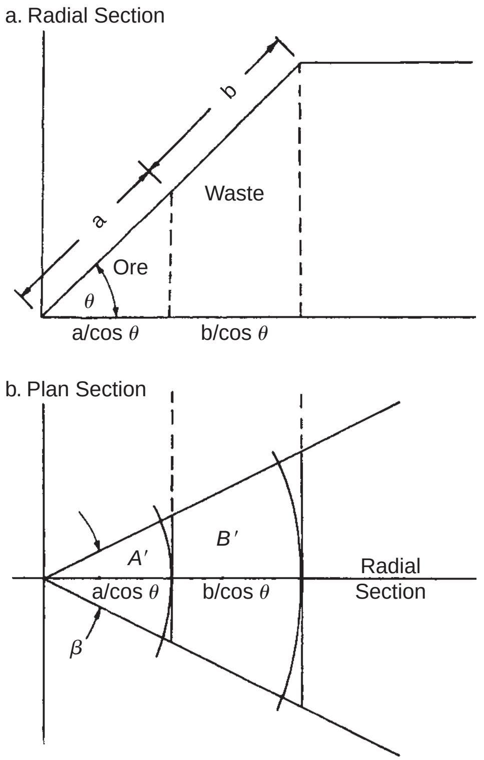


Figure 5.19. Plan and section views of the radial sector shown in Figure 5.18.

Figure 5.18 is an isometric view of the sector in question. The exposed ore area is identified as *A* and that of waste as *B*. The apparent (measured) stripping ratio for the radial section as shown on Figure 5.19a would be

$$\text{SR (measured)} = \frac{b}{a} \tag{5.10}$$

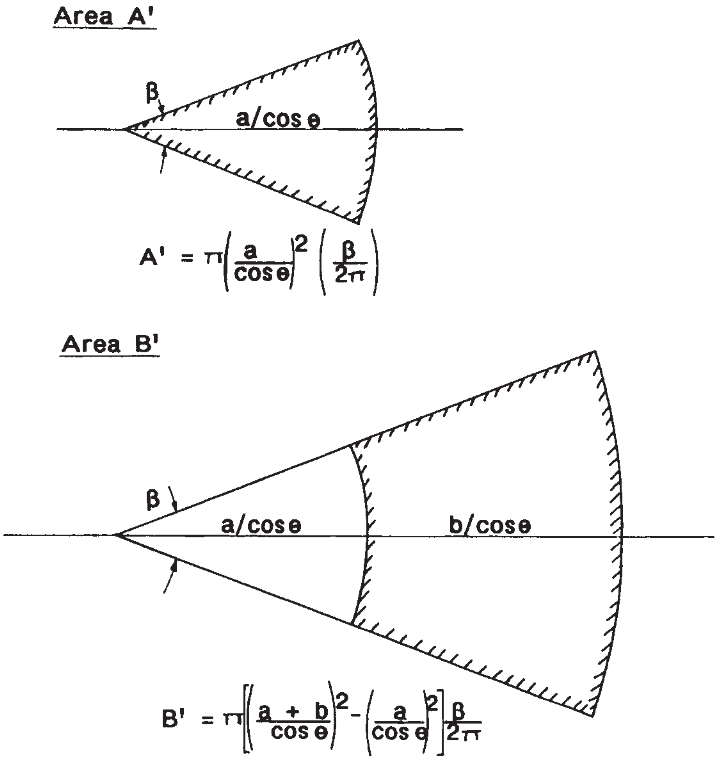


Figure 5.20. The ore and waste areas involved.

From the plan projection of the wall surface for this sector as shown in Figure 5.19b, the area of ore is A' and that of waste B'. Since the included angle for the sector is β, the ore radius in this projection is $a/\cos\,\beta$ and the thickness of the waste zone is $b/\cos\,\theta$.

The calculation of the plan projected areas is shown in Figure 5.20. The true stripping ratio can be expressed as

$$\text{SR (true)} = \frac{B'}{A'} = \frac{(a+b)^2 - a^2}{a^2} = \left(1 + \frac{b}{a}\right)^2 - 1$$

Since the measured stripping ratio on the section is as shown in Equation (5.10) then

$$\text{SR (true)} = (1 + \text{SR (measured)})^2 - 1 \tag{5.11}$$

As can be seen, this relationship is independent of both the slope angle and the included angle.

Values of SR (true) and SR (measured) are presented in Table 5.8 and plotted in Figure 5.21. The steps in the location of the final pit limit on a radial section are outlined below:

Step 1. As with parallel sections, one guesses the location of the final pit slope and calculates the average ore grade $\bar{g}$ and the stripping ratio. Assume that $\bar{g} = 0.8$ and the measured stripping ratio is 2 : 1.

Table 5.8. Comparison of true versus measured stripping ratios for radial sections.

Stripping ratio	
Measured	True
0	0
0.25	0.56
0.50	1.25
0.75	2.06
1.0	3.0
1.25	4.06
1.5	5.25
2.0	8.0
2.5	11.25
3.0	15.0

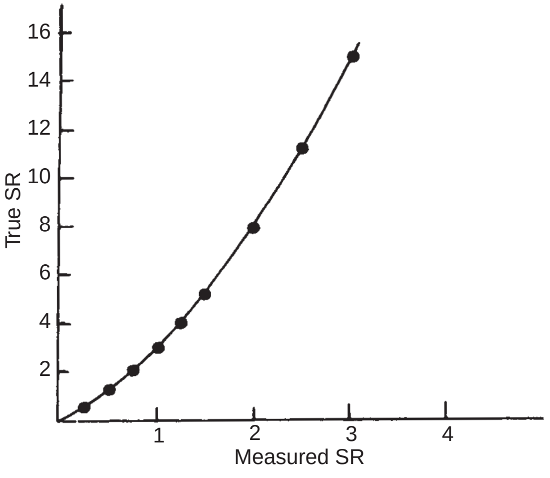


Figure 5.21. True versus measured stripping ratios (Koskiniemi, 1979).

Step 2. With parallel sections one would go directly to the SR – %Cu curve to see what stripping ratio this grade would support. In the case shown in Figure 5.22, it is about 6 : 1 indicating that the limit could be moved outward. For radial sections one must proceed to Step 3.

Step 3. The measured stripping ratio must first be converted to a true stripping ratio through the use of the conversion curve (Fig. 5.21). This indicates that for a measured stripping ratio of 2 : 1, the true stripping ratio is 8 : 1.

Step 4. Returning to Figure 5.22, one finds that the grade required to support a stripping ratio of 8 : 1 is 0.94. This is higher than the 0.8 present and hence the next guess of the final pit limit should be moved toward the pit.

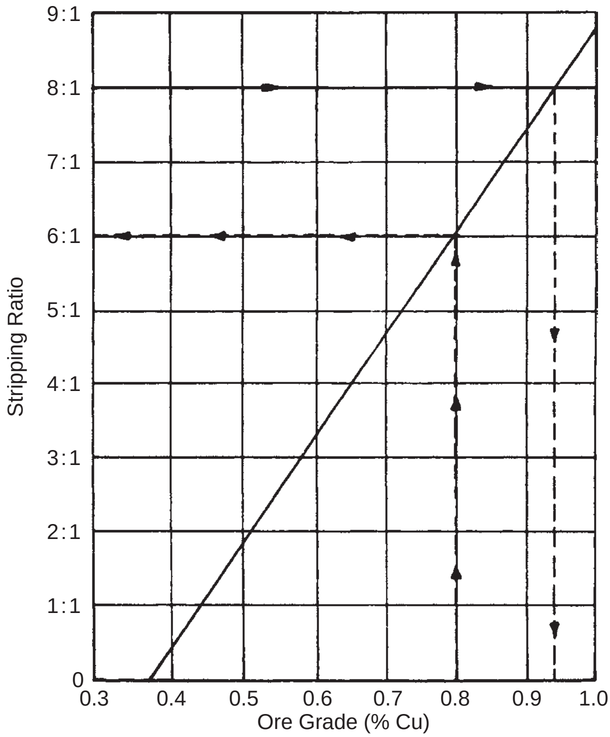


Figure 5.22. Example of stripping ratio calculation for a radial section.

In summary, the treatment of radial sections is done in exactly the same way as with parallel sections except that the use of an intermediate curve to convert measured SR to true SR is required. It should be pointed out that the ore region is pie-shaped and strictly speaking a weighted average approach should be used to calculate an average ore grade.

5.2.7 *Generating a final pit outline*

Once final pit profiles have been located on the individual sections, they must be evaluated as a group to examine how they fit together relative to one another. One section may suggest a very narrow pit, for example, whereas the adjacent ones yield a wide one (see Fig. 5.23). This smoothing requires adjustments to be made to the various sections. The easiest way of visualizing the pit and performing this task is through the use of level plans (horizontal sections). The final ore reserve estimation will also be done using plans. Thus to proceed requires the development of a composite mine plan map from the vertical sections.

The steps are outlined below (Koskiniemi, 1979):

Step 1. Transfer of intercepts to plan map. The locations of the pit bottom and the surface intercepts of the pit limits are transferred from the vertical sections to the plan map.

Step 2. Handling of discontinuous slopes. If a vertical section does not have a continuous slope line from the pit bottom to the surface intercept this is also shown.

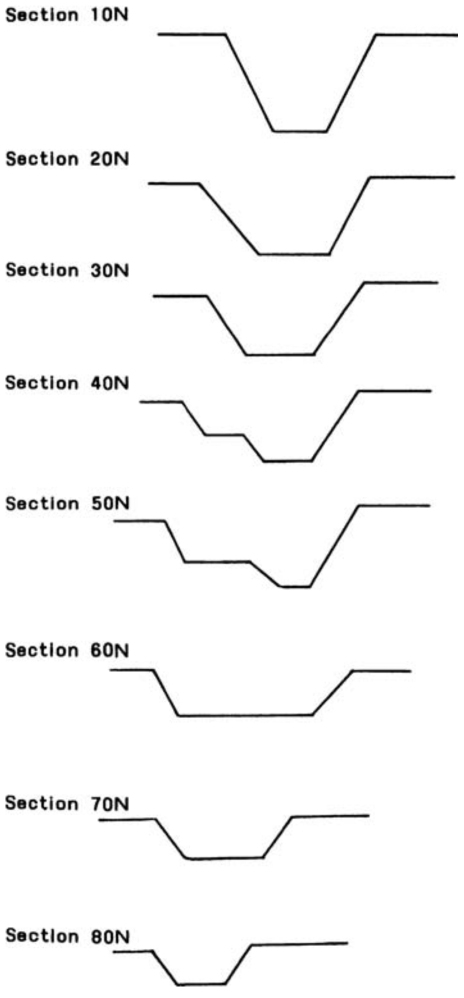


Figure 5.23. A series of initial pit outline sections for an orebody.

Step 3. Deciding on initial bench design location. The actual design of the composite plan generally begins with the pit bottom working upward. (Some designers prefer however to begin the design with a middle bench and work both upwards and downwards.)

Step 4. Smoothing. The points from the sections usually present a very irregular pattern, both vertically and horizontally. In smoothing these and designing the bottom bench several things should be kept in mind (Koskiniemi, 1979):

– Averaging the break-even stripping ratios for adjoining sections. If one section is moved significantly inward or outward, major changes in stripping/ore reserves may result. Thus the sections are carefully evaluated with each change.

– Use of simple geometric patterns for ease of design. The simpler the geometric shape, the easier it is to design the remainder of the pit.

– Location of ramp to pit bottom.

– Watching for patterns that might lead to slope stability problems. For example bulges or noses in the pit often are sources of problems.

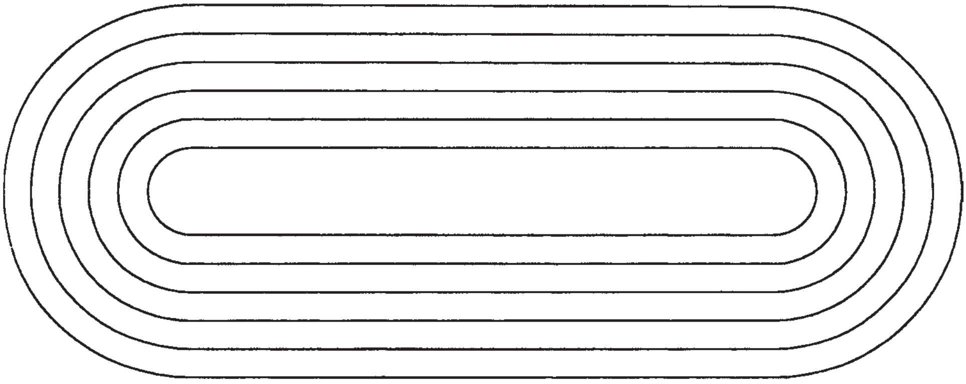


Figure 5.24. Plan view of a pit with constant slope angle.

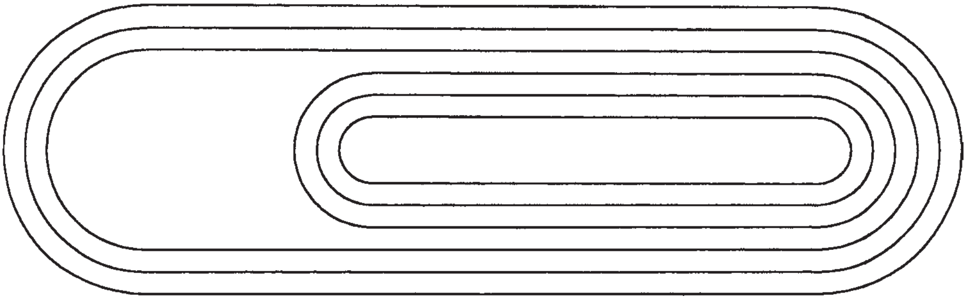


Figure 5.25. Plan view of a pit containing a wide bench.

Step 5. The bottom bench is then drawn. As has been discussed in Chapter 4, there are several possibilities for depicting the geometry. One way is to show both the toe and crest lines. Depending upon map scale and the number of benches, this may or may not be good. A second possibility is to display just one (either the toe or the crest) since the position of the other can be easily obtained knowing the bench height and the bench face angle.

The third possibility is to show the median elevation line (half the distance up the bench face). This is the representation most commonly used and that employed here.

Step 6. Addition of the median lines for the overlying benches. In preliminary designs the roads are often not shown. In such cases, the plan distance between the median lines is:

$$M_{\mathrm{d}} = \frac{H}{\tan\,\theta} \tag{5.12}$$

where M_{d} is the plan distance between median lines, H is the bench height and θ is the overall pit slope angle. An overall pit slope angle is chosen so that the space required for the road is included. For the case shown in Figure 5.24, the slope angle is the same throughout the pit and the slope is continuous from toe to crest and hence the median lines are parallel and equally spaced. Figure 5.25 shows the case when there is a wide bench part way up the pit. Figure 5.26 shows the case when the north and south side slope angles are different.

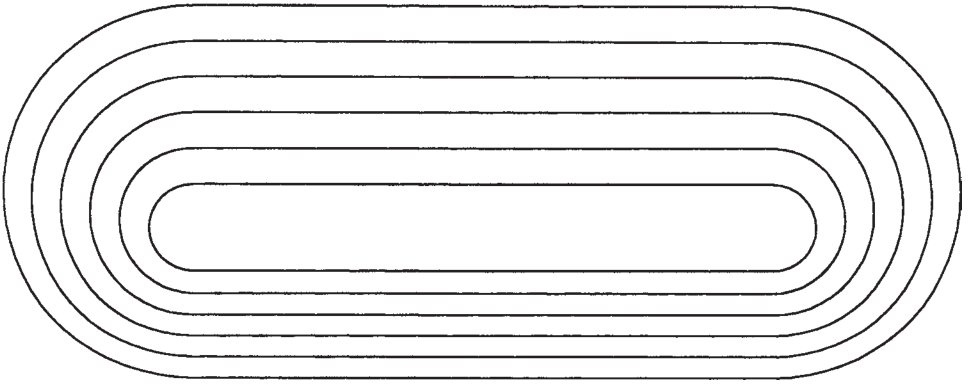


Figure 5.26. Plan view of a pit with different north and south wall slopes.

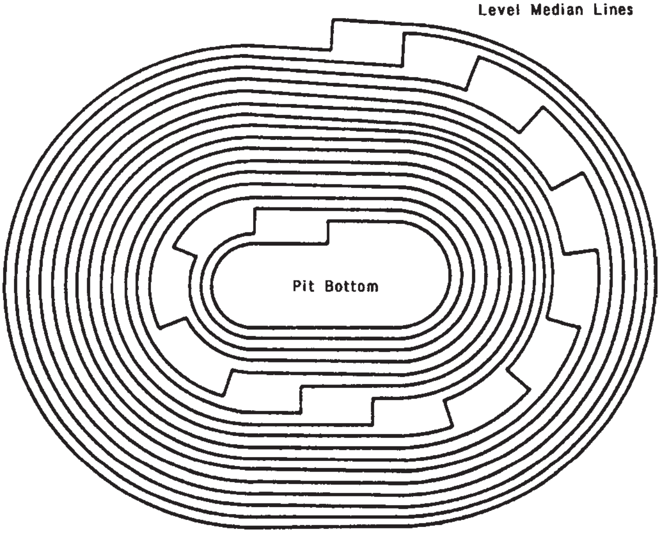


Figure 5.27. Composite ultimate pit plan (Koskiniemi, 1979).

If in the final designs the actual roads are shown, then the horizontal distance M_d between the median lines is equal to

$$M_d = b_w + \frac{H}{\tan \theta_f} \tag{5.13}$$

where b_w is the level berm width, H is the bench height, and θ_f is the bench face angle. A composite ultimate pit plan with a road included is shown in Figure 5.27. The procedure through which a road is added to a pit has been discussed in Chapter 4.

Step 7. Transfer of the pit limits to the individual level plans. When the composite pit plan has been completed, the pit limits are transferred to the individual level plans (Fig. 5.28). Through this process, the design engineer will first check the stripping ratios at the final pit limit. The pit is split into sectors such as shown in Figure 5.29, and the ore-waste relationships checked.

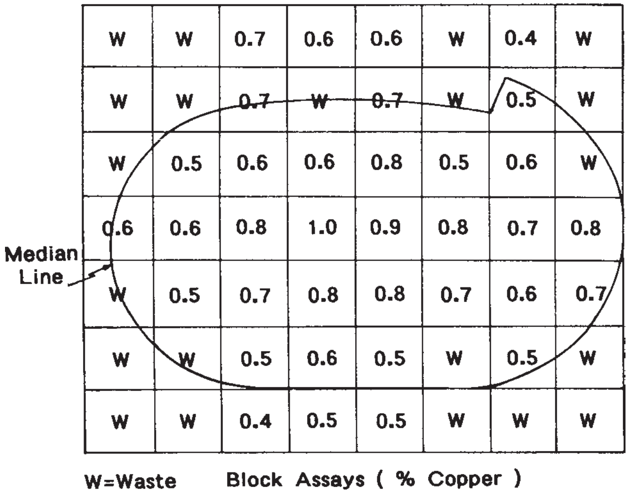


Figure 5.28. Level plan of bench a with the ultimate pit limit superimposed (Koskiniemi, 1979).

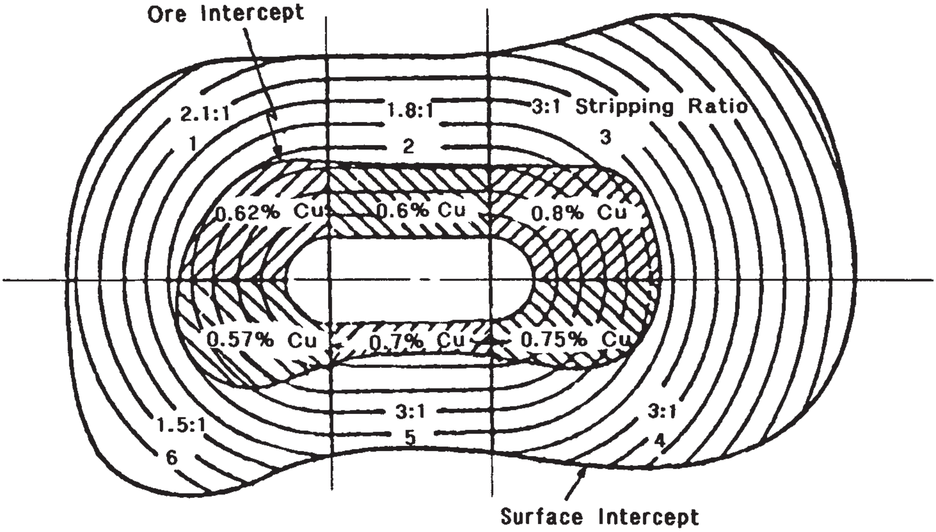


Figure 5.29. Plan of the ultimate pit showing the sectors (Soderberg & Rausch, 1968).

Figure 5.30 shows the median lines and ore grades for a sector taken in the center of the pit. If the pit bottom is in ore then this effect is included simply by adding artificial median lines to this region.

In this case the average grade is 0.7 and the stripping ratio is

$$\mathrm{SR} = \frac{3}{4} = 0.75$$

A radial section is shown in Figure 5.31. In this case

$$\bar{g} = \frac{\sum r_i g_i}{\sum r_i} = \frac{0.9 \times 1 + 0.8 \times 2 + 0.6 \times 3 + 0.5 \times 4}{1 + 2 + 3 + 4} = 0.63$$

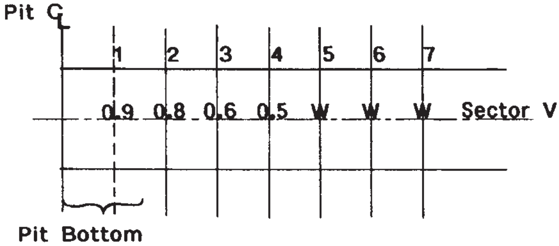


Figure 5.30. Average grade calculation for a parallel sector.

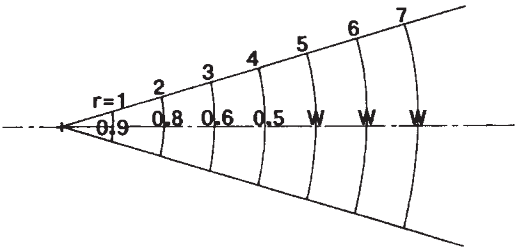


Figure 5.31. Average grade calculation for a radial sector.

The stripping ratio is

$$\mathrm{SR} = \frac{\sum r_{\mathrm{wi}}}{\sum r_{\mathrm{oi}}} = \frac{5+6+7}{1+2+3+4} = 1.8$$

The general procedure is to measure and sum the lengths of waste and ore on each level at the pit limit within a given sector. The ratio gives the stripping ratio. The average grade is the length weighted average. The grade and stripping ratio values for each sector are compared to the SR – grade curve. Anomalous areas are identified and the design reviewed to see what corrections can be made.

Step 8. Calculation of ore reserves by level. The calculation of final ore reserves and overall stripping ratio is done using the level maps. For each level the ore tons and grade and the waste tons are determined within the ultimate pit limits. Assuming that the block size in Figure 5.28 is $100' \times 100' \times 40'$ and that the tonnage factor is 12.5 ft^3/ton, then each whole block contains 32,000 tons. A tabulation of the reserves on this level is presented in Table 5.9. Ore is considered to be that material within the pit limits having a grade equal to or greater than the grade from the stripping curve at a break-even stripping ratio of zero. This is generally called the ore reserve cut off grade. Using a cutoff grade of 0.37%, the total tons of ore and waste are computed. The average ore grade is also obtained.

Step 9. Calculation of final ore reserve and overall stripping ratio. The ore reserves for each level are tabulated such as shown in Table 5.10. From this the ore tons, overall average ore grade, total waste tons and overall stripping ratio can be found. In addition tonnage-grade distribution curves are often plotted both overall and by level to help in mill design and setting of production levels.

5.2.8 *Destinations for in-pit materials*

Once the pit limits have been defined, all of the material within the pit outline will be mined irrespective of its value. Given mined material the decision must be made regarding its

Table 5.9. Reserve summary for bench A.

Grade (%)	Tons ore	Tons waste
<0.30		125,300
0.30–0.36		0
0.37–0.39	0	
0.40–0.49	0	
0.50–0.59	193,340	
0.60–0.69	192,000	
0.70–0.79	144,400	
0.80–0.89	189,000	
0.90–0.99	32,000	
1.00–1.09	32,000	

Table 5.10. Total reserves within the final pit outline.

Bench	Grade									
	<0.30	0.3–0.36	0.37–0.39	0.4–0.49	0.5–0.59	0.6–0.69	0.7–0.79	0.8–0.89	0.9–0.99	1.0–1.09
A	125,300	0	0	0	193,340	192,000	144,400	189,000	32,000	32,000
B										
C										
D										
E										
F										
G										
H										
I										
J										
Totals										

destination. A new cutoff grade based simply upon meeting the processing costs can be calculated. For the example presented in Section 5.2.2, the direct mining cost was given as $1 per ton. If this is subtracted from the total cost then the net value – grade equation becomes

$$y = -4.2 + 14.0\,x$$

Solving for the breakeven cutoff grade (sending to the mill or the dump) one finds that

$$x = 0.30\%\,\mathrm{Cu}$$

Hence for the pit material everything having a grade greater than or equal to 0.30% Cu could be sent to the mill. This assumes, of course, that the transport costs to the dumps and the mill are the same. If not, then the cost differences can be included. Such considerations can be made only if mill capacity is available. Often this is not the case. The economics of this marginal/submarginal material must be evaluated very closely.

Table 5.11. Typical mineral block model data items (Crawford & Davey, 1979).

Copper (Cu)	%
Molybdenite (MoS_2)	%
Gold (Au)	oz/st
Silver (Ag)	oz/st
Copper concentrate recovery	%
Copper concentrate grade	%
Copper smelting recovery	%
Copper in blister	%
Copper refining recovery	%
MoS_2 concentrate recovery	%
MoS_2 conversion recovery	%
Gold concentrate recovery	%
Gold refining recovery	%
Silver concentrate recovery	%
Silver refining recovery	%

5.3 ECONOMIC BLOCK MODELS

The block model representation of orebodies rather than section representation and the storage of the information on high speed computers has offered some new possibilities in open-pit planning. The use of computers allows the rapid updating of plans as well as exploring a wide number of parameters through sensitivity analysis. Although there is still a great deal of opportunity for engineering interaction, much of the tedious work can be done by computer. Both of the major computerized techniques to be described in this chapter: Floating cone and Lerchs-Grossmann, require an initial economic evaluation of the blocks.

Mineral, metallurgical and economic data are combined to assign a net dollar value to each mineral model block. Table 5.11 shows typical mineral block data items for an orebody containing copper, molybdenite, gold and silver. The block size is $50 \times 50 \times 40$ ft and the tonnage factor is 13.5 ft^3/st. The economics format is shown in Table 5.12. All of the costs with the exception of truck haulage and roads are expressed as fixed unit costs for the indicated production quantity unit. All mining and processing costs include operating, maintenance and depreciation costs.

The costs for truck haulage and roads may vary because of haulage profile, length and lift. The costs per hour are first estimated and then converted to unit costs per ton based upon an estimated hourly haulage productivity. The projected haulage productivity may be obtained using haulage simulators. Due to this variation in the haulage and road components, the mining costs for both ore and waste contain a fixed component plus a mining level (bench location) dependent component. This is shown in Table 5.13.

Using Tables 5.11, 5.12, and 5.13, the net value for the block can be determined. In this particular example, the material is considered as either mill feed or waste. Thus the net value calculations are done for these two possibilities. The net value as stripping (NV_{st}) is found by multiplying the block tonnage times the stripping cost.

$$NV_{st} = \text{Block tonnage} \times \text{Stripping cost}_{st} \ (\$/\text{st})$$

As can be seen in Table 5.13, the stripping cost is made up of two items:

- Mining cost ($/st waste), and
- General plant cost ($/st waste).

Table 5.12. Block model mining and processing cost items (Crawford & Davey, 1979).

Drilling	\$/st ore and waste
Blasting	\$/st ore and waste
Loading	\$/st ore and waste
Hauling	\$/truck hour
Haul roads	\$/truck hour
Waste dumps	\$/st waste
Pit pumping	\$/st ore
Mine general	\$/st ore
	\$/st waste
Ore reloading	\$/st ore
Ore haulage	\$/st ore
Concentrating	\$/st ore
Concentrate delivery	\$/st concentrate
Smelting	\$/st concentrate
General plant	\$/st ore
	\$/st waste
	\$/st blister
Blister casting loading and freight	\$/st blister
Refining	\$/st blister
Selling and delivery	\$/lb refined copper
Metal prices	
Copper	\$/lb
MoS_2	\$/lb
Gold	\$/oz
Silver	\$/oz

Table 5.13. Pit limit analysis cost summary (Crawford & Davey, 1979).

	Stripping (waste)	Mill feed (ore)
Mining (\$/st)	$aL^* + b$	$cL^* + d$
Processing and other:		
Ore haulage (\$/st ore)		X
Concentrating (\$/st ore.)		X
Concentrate delivery (\$/st conc.)		X
Smelting (\$/st conc.)		X
Blister casting loading, and freight (\$/st blister)		X
General plant		
\$/st ore		X
\$/st waste	X	X
\$/st blister		X
Refining (\$/st blister)		X
Selling and delivery(\$/lb Cu)		X

* Level elevation.

With regard to the second cost item (general plant), there are many ‘overhead’ types of costs which are independent of whether the material being moved is ‘ore’ or ‘waste’.

Although, in the end, the ‘ore’ must pay for all of the costs, there is some logic for allocating general plant costs to the waste as well (see Subsection 2.4.1).

Table 5.14. Block evaluation calculation format.

Block data		
Block location	Data for block	
Block tonnage	from Table 5.11 and 5.13	
As mill feed*		
Revenue from metal sales ($)		
Copper: Block tonnage × lb saleable Cu/st × Price ($/lb)		$= A$
MoS_2		$= B$
Gold		$= C$
Silver		$= D$
Total revenue		R
Cost of copper ($):		
Block tonnage × lb saleable Cu/st × Cost ($/lb)		$= C'$
Net value as mill feed ($):		
$NV_{mf}(\$) = R - C'$		
Net value as stripping ($):		
$NV_{st}(\$)$ = Block tonnage × Stripping cost ($/st)		

* The block is designated as mill feed and assigned NV_{mf} if net value is greater than NV_{st}, even if negative.

The net value as mill feed (NV_{mf}) requires the calculation of the revenues and costs (excluding any stripping) for the block. Table 5.14 summarizes the calculations. The mill feed and stripping net values are compared for each block and the most positive value is assigned. This net value then becomes the only piece of block data used directly in the mining simulators.

5.4 THE FLOATING CONE TECHNIQUE

In the section on manual techniques for determining the final pit limits, it was shown that for a single pit wall, the line having the slope angle defining the wall was moved back and forth until the actual grade and stripping ratio matched a point on the SR-grade curve. For the case when both walls and the pit bottom were in ore, a geometric figure in the shape of the minimum pit was constructed and moved around on the section (vertically and horizontally) until the stripping ratios and grades measured along the periphery matched that from the curve. For the case of 45° slopes and a 100 ft minimum pit bottom width, the figure to be 'floated' is as shown in Figure 5.32. If this figure is revolved around the axis, the solid generated is called the frustum of a cone. Had the minimum pit width been zero, then the figure would simply be a cone. With sections it was most convenient to use two-dimensional representation and not consider the effect of adjacent sections. This was at least partially done later through the smoothing on plan.

Although there are other techniques for determining the final pit configuration, currently the most popular is through the use of this 'floating cone'. Several changes in the manual process are necessary to accommodate the computer. In the manual process, a net value – grade curve was developed. Knowing the cost of waste removal, this curve was changed into a SR – grade curve. The user then simply evaluated stripping ratio (normally by measuring lengths of ore and waste) and calculated the weighted average grade. Through the use of the SR-grade 'nomograph' one could examine pit expansion.

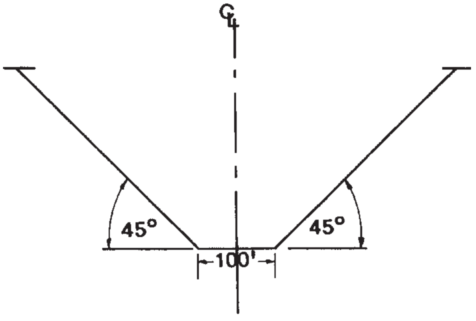


Figure 5.32. Pit profile to be 'floated'.

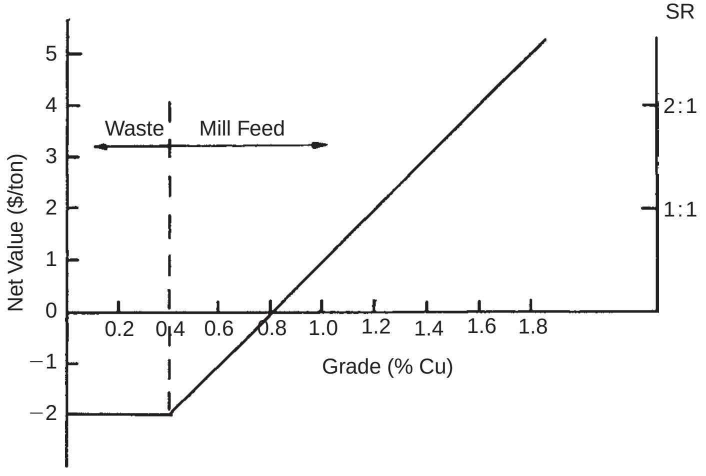


Figure 5.33. Net value – grade curve showing the waste/mill feed split.

When using the computer, it is more convenient to use the net values of the blocks directly. Figure 5.33 shows the type of net value-grade curve which is actually used. Consider the simple example shown in Figure 5.34. This case will be examined using both the manual method based upon grades and stripping ratios, and that in which net values are assigned to the blocks. Three potential pit limits have been superimposed on the grade block model of Figure 5.35. A pit slope of 45° is assumed. The average ore grades and stripping ratios are given. The final pit is represented by case 3.

Using Figure 5.33, the grade block model of Figure 5.34 can be converted into an economic block model. The result is shown in Figure 5.36. By examining the net values of the blocks involved in a particular mining sequence, final pit limits can be determined. Mining is stopped when the net value is negative. The net values for the three cases examined in Figure 5.35 are given in Figure 5.37.

Slice 3 has a NV = 0 and would define the final pit on this section. As can be seen the manual and economic block approaches give equivalent results. The net value approach is by far the easiest to program, particularly when considering a three dimensional array of blocks.

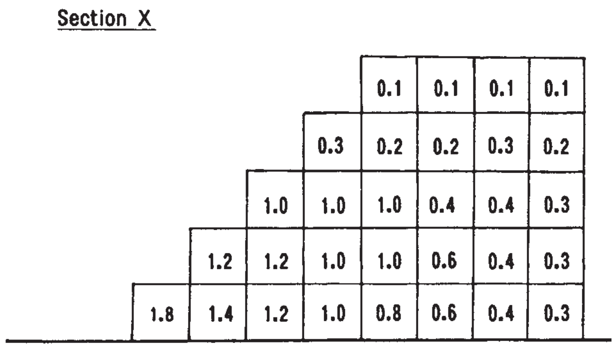


Figure 5.34. Grade block model for pit limit example.

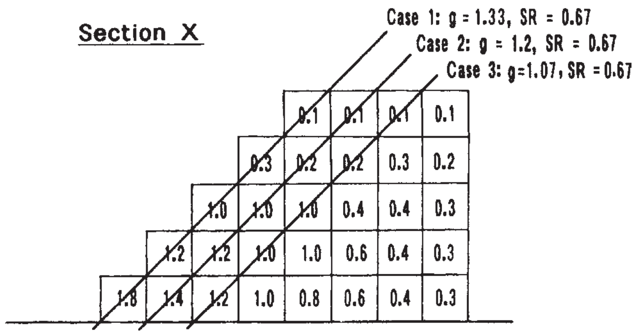


Figure 5.35. Trial final pit limits based on the manual procedure. Case 3 is the final limit.

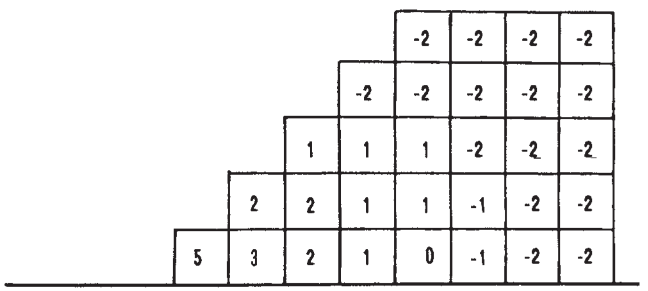


Figure 5.36. Corresponding economic block model.

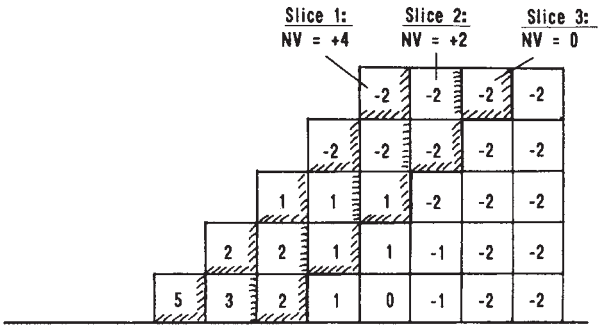


Figure 5.37. Trial final pit limits.

	1	2	3	4	5	6	7
1	−1	−1	−1	−1	−1	+1	−1
2		−2	−2	+4	−2	−2	
3			+7	+1	−3		

Block Value

Figure 5.38. Block model for Example 1 (Barnes, 1982).

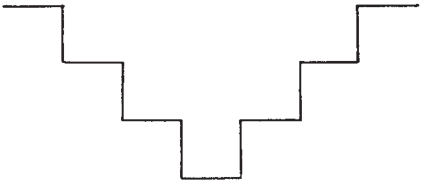


Figure 5.39. Stepped cone pit model.

The principles involved in determining the pit outline, given an entire 2-dimensional array of blocks will now be demonstrated. The examples were originally presented by Barnes, 1982. Figure 5.38 shows the sample section to be considered with the net values given. The blocks are equidimensional and the slope angle will be 45°. This means that slope is formed by going up one and over one block. Figure 5.39 shows the stepped cone which will be used for determining the final pit.

The following steps are used:

Step 1. The cone is 'floated' from left to right along the top row of blocks in the section. If there is a positive block it is removed.

Step 2. After traversing the first row, the apex of the cone is moved to the second row. Starting from the left hand side it 'floats' from left to right stopping when it encounters the first positive block. If the sum of all the blocks falling within the cone is positive (or zero), these blocks are removed (mined). If the sum is negative the blocks are left, and the cone floats to the next positive block on this row. The summing and mining or leaving process is repeated.

Step 3. This floating cone process moving from left to right and top to bottom of the section continues until no more blocks can be removed.

Step 4. The profitability for this section is found by summing the values of the blocks removed.

Step 5. The overall stripping ratio can be determined from the numbers of positive (+) and negative (−) blocks.

These rules can now be applied to the section shown in Figure 5.38. There are four positive blocks in the model hence there are four corresponding cones which must be evaluated. Using a top–down rule, the block at row 1/column 6 would initiate the search. Since there are no overlying blocks, the value of the cone is the value of the block: 1. The value is positive, so the block is mined (Fig. 5.40).

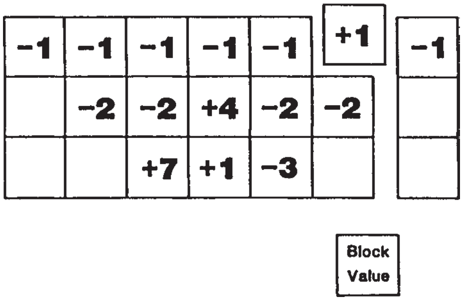


Figure 5.40. First incremental cone – Example 1 (Barnes, 1982).

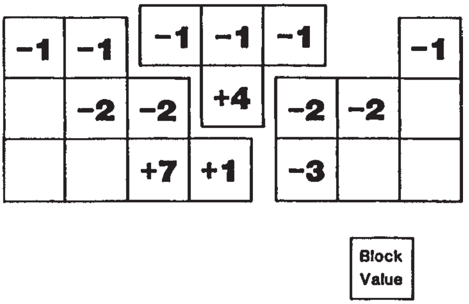


Figure 5.41. Second incremental cone – Example 1 (Barnes, 1982).

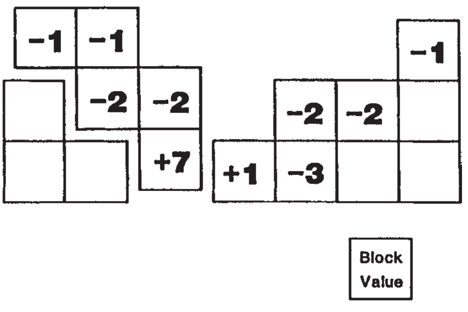


Figure 5.42. Third incremental cone – Example 1 (Barnes, 1982).

The next incremental cone is that defined by the block at row 2/column 4. The value of this cone is

$$-1-1-1+4=+1$$

Since this value is positive, the cone is mined (Fig. 5.41). For the incremental cone defined by the block at row 3/column 3, its value is

$$-1-1-2-2+7=+1$$

Again, since the value is positive, this cone is mined (Fig. 5.42). Finally, the value of the incremental cone defined by the block at row 3/column 4 is

$$-2+1=-1$$

The value of this cone is negative; therefore, the cone is not mined (Fig. 5.43). Figure 5.44 depicts the overall final ultimate pit. The total value of this pit is

$$-1-1-1-1-1+1-2-2+4+7=+3$$

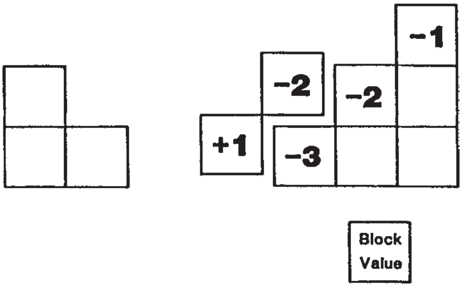


Figure 5.43. Fourth incremental cone – Example 1 (Barnes, 1982).

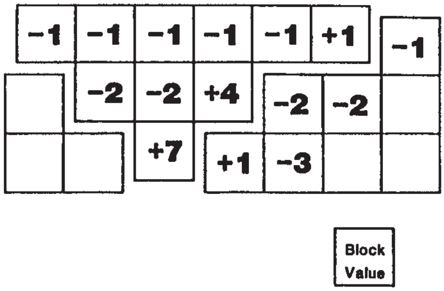


Figure 5.44. Ultimate pit – Example 2 (Barnes, 1982).

The overall stripping ratio is

$$\text{SR (overall)} = \frac{7}{3}$$

In this simple situation, the final pit is 'optimal'. Over the years, considerable effort has been spent in trying to develop procedures which when applied would yield the 'optimal' pit. The 'optimum' might be defined for example, as the one yielding one of the following,

– maximum profit,
– maximum net present value,
– maximum extraction.

Although many contend that the determination of such an 'optimum' pit is largely of academic interest due to changing prices, costs and poorly defined grades, none-the-less efforts continue in this direction. Barnes (1982) has presented three problems representing floating cone situations not leading to an optimum pit.

Problem 1. Missing combinations of profitable blocks

This problem occurs when positive (ore) blocks are investigated individually. A single ore block may not justify the removal of the necessary overburden while combinations of these blocks with overlapping cones are profitable. Johnson (1973) has labelled this 'the mutual support problem'. This is demonstrated in Figures 5.45 to 5.48. The cone defined by the block at row 3/column 3 has a value of

$$-1 - 1 - 1 - 1 - 1 - 2 - 2 - 2 + 10 = -1$$

Since the value is negative, the cone would not be mined by the simple moving cones method (Fig. 5.46). Similarly, for the cone defined by the block at row 3/column 5, the value is

$$-1 - 1 - 1 - 1 - 1 - 2 - 2 - 2 + 10 = -1$$

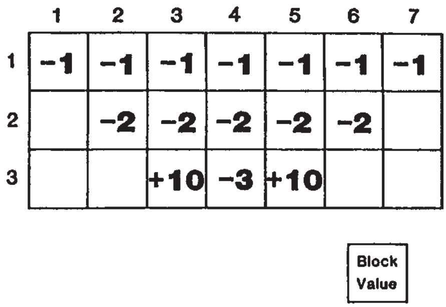


Figure 5.45. Block model for Example 2 (Barnes, 1982).

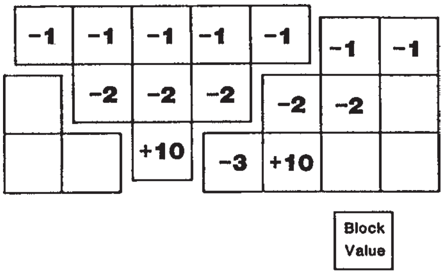


Figure 5.46. Second incremental cone – Example 2 (Barnes, 1982).

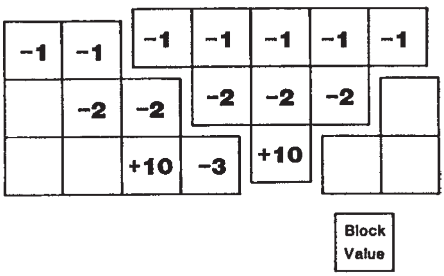


Figure 5.47. Second incremental cone – Example 2 (Barnes, 1982).

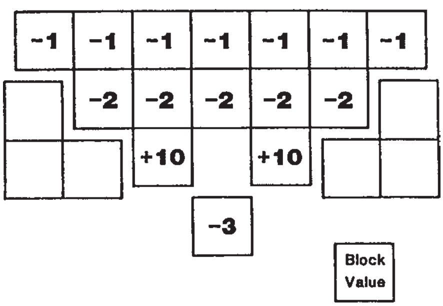


Figure 5.48. True optimal pit – Example 2 (Barnes, 1982).

Again, this cone would not be mined (Fig. 5.47). Therefore, using the simple cone analysis, nothing would be mined. However, due to the overlapping (mutual support) portion of the overburden cones, the value of the composite union is positive

$$-1-1-1-1-1-1-1-2-2-2-2-2+10+10=+3$$

This situation (Fig. 5.48) occurs often in real-world mineral deposits, and a simple moving cones approach misses it.

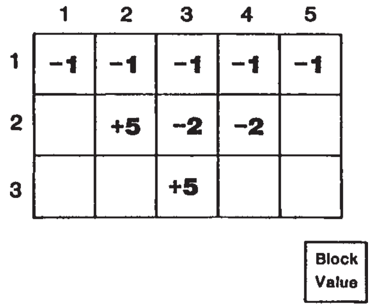


Figure 5.49. Block model for Example 3 (Barnes, 1982).

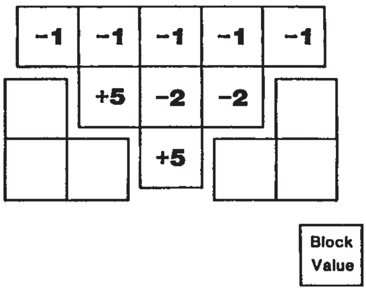


Figure 5.50. Large incremental cone for Example 3 (Barnes, 1982).

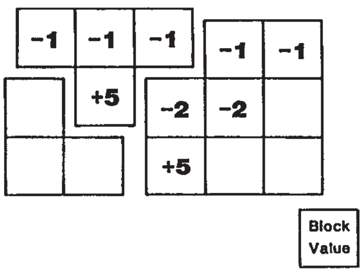


Figure 5.51. Small incremental cone for Example 4 (Barnes, 1982).

Problem 2. Extending the ultimate pit beyond the optimal pit limits
This is the situation where the moving cones algorithm can and often will include non-profitable blocks in the pit design. The inclusion of non-profitable blocks will reduce the net value of the pit. This situation occurs when profitable ore blocks, or profitable combinations of ore blocks, cause a cone defined by an underlying apex to be positive; i.e., the positive values are being extended downward to carry waste below their cones. The two-dimensional block model shown in Figure 5.49 assumes the maximum pit slope to be 45 degrees. The value of the cone defined by the block at row 3/column 3 (Fig. 5.50) is

$$-1-1-1-1-1+5-2-2+5=+1$$

The fact that the value of this cone is positive does not imply that the cone should be mined. As shown on Figure 5.51, the block at row 2/column 2 is carrying this cone. The proper design includes only the block at row 2/column 2 and its three overlying blocks, row 1/columns 1, 2, and 3. The value of the optimal design is

$$-1-1-1+5=+2$$

The value of the small cone is greater than the value of the large cone.

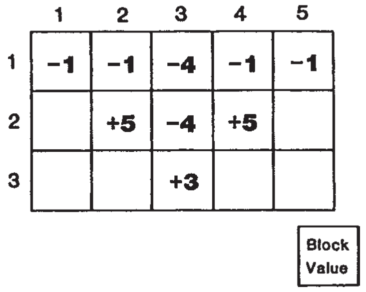


Figure 5.52. Block model for Example 4 (Barnes, 1982).

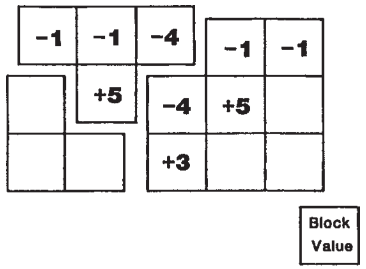


Figure 5.53. First incremental cone for Example 4 (Barnes, 1982).

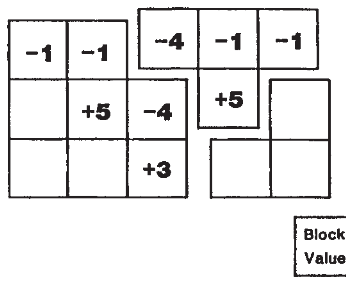


Figure 5.54. Second incremental cone for Example 4 (Barnes, 1982).

Problem 3. Combination of problems 1 and 2

The most common and most difficult situation involving these two problems is their simultaneous occurrence. The two-dimensional block model shown on Figure 5.52 assumes a 45° pit slop. There are three positive blocks, and therefore three possible incremental cones.

The value of the cone defined by the block at row 2/column 2 (Fig. 5.53) is

$$-1 - 1 - 4 + 5 = -1$$

The value of the cone defined by the block at row 2/column 4 (Fig. 5.54) is

$$-4 - 1 - 1 + 5 = -1$$

Yet, the value of the cone defined by the block at row 3/column 3 (Fig. 5.55) is

$$-1 - 1 - 4 - 1 - 1 + 5 - 4 + 5 + 3 = +1$$

This would appear to imply that the pit design shown on Figure 5.55 is optimal; however, this is not the case. The optimal design is shown on Figure 5.56. The value of this pit is

$$-1 - 1 - 4 - 1 - 1 + 5 + 5 = +2$$

This value is one more than the ‘initially apparent’ pit.

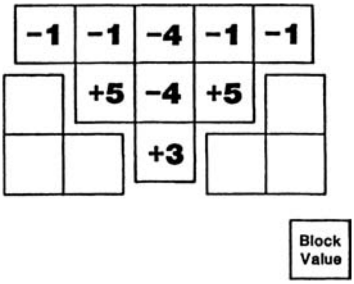


Figure 5.55. Third incremental cone for Example 4 (Barnes, 1982).

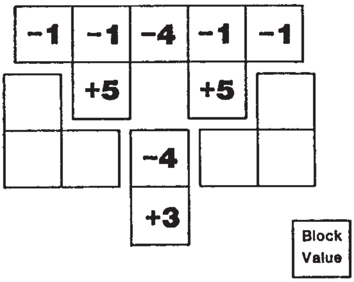


Figure 5.56. Optimal pit design for Example 4 (Barnes, 1982).

In spite of these problems, there are however a number of very positive aspects of the technique which account of its widespread use and popularity (Barnes, 1982):

1. Since the method is a computerization of manual techniques, mining engineers can use the method, understand what they are using, and feel comfortable with the results.

2. Computationally, the algorithm is quite simple. Development and implementation of a moving cones computer program does not require sophisticated knowledge in operations research or computer science. The computer code could be developed in-house, rather than purchased from a software company; thus, a more custom-fitted product can be provided at an operating mine site.

3. The moving cones technique can be used with generalized pit slopes. The single requirement is an unambiguous rule for determining which blocks overlie individual ore blocks.

4. It provides highly useable and sufficiently accurate results for engineering planning.

5.5 THE LERCHS-GROSSMANN 2-D ALGORITHM

In 1965, Lerchs and Grossmann published a paper entitled 'Optimum design of open-pit mines'. In what has become a classic paper they described two numeric methods:

– a simple dynamic programming algorithm for the two-dimensional pit (or a single vertical section of a mine),

– a more elaborate graph algorithm for the general three-dimensional pit.

This section will discuss the first method with the second being described in Section 5.7. The easiest way of presenting the technique is through the use of an example. The mathematics which accompany the actual mechanics will be given at the same time. This example was originally presented by Lerchs & Grossmann (1965) and elaborated upon by Sainsbury (1970). The orebody is as shown in Figure 5.57.

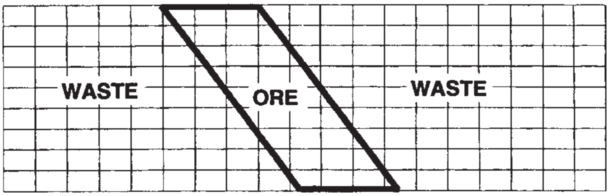


Figure 5.57. Orebody geometry for the Lerchs-Grossmann 2-D example (Sainsbury, 1970; Lerchs & Grossmann, 1965).

The following apply:

$$NV_{st} = -\$4 \times 10^3/\text{block}$$

$$NV_{mf} = \$12 \times 10^3/\text{block}$$

$$\text{Slope angle} = 35.5^\circ$$

$$\text{Bench height} = 40\,\text{ft}$$

$$\text{Tonnage factor} = 12.5\,\text{ft}^3/\text{st}$$

For ease in calculation the values \$12 and $-\$4$ will be used. At the conclusion, the factor of 1000 will be reintroduced. To apply this technique the grid (block geometry) is selected based upon the slope angle. The slope is formed by moving up one block and over one block. Hence for a bench height of 40 ft and a slope angle of 35.5° one finds using Equation (5.14)

$$\alpha = \frac{H}{B} = \tan\theta \tag{5.14}$$

where α is the ratio of block height/block width, H is the block height, B is the block width, and θ is the slope angle, that

$$B = \frac{H}{\tan\theta} = \frac{40}{\tan 35.5^\circ} = 56\,\text{ft}$$

Hence

$$\alpha = \frac{5}{7}$$

This grid system has been superimposed on the section in Figure 5.57. In Figure 5.58 the net values have been added. As can be seen, the boundary blocks contain both ore and waste elements. A weighted averaging procedure has been used to obtain the block model of Figure 5.59. The block positions will be denoted using an i, j numeration system. In keeping with the nomenclature used by Lerchs and Grossmann, i refers to the rows and j to the columns. The first step in this procedure is to calculate cumulative profits for each column of blocks starting from the top and moving downward. Each vertical column of blocks is independent of the others. This process is shown in Figure 5.60 for column $j = 6$.

The equation which describes this process is

$$M_{ij} = \sum_{k=1}^{i} m_{kj} \tag{5.15}$$

-4	-4	-4	-4	-4	-4 / 12	12	12	12 / -4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
	-4	-4	-4	-4	-4 / 12	12	12	12 / -4	-4	-4	-4	-4	-4	-4	-4	-4	-4	
		-4	-4	-4	-4	-4 / 12	12	12	12 / -4	-4	-4	-4	-4	-4	-4	-4		
			-4	-4	-4	-4 / 12	12	12	12 / -4	-4	-4	-4	-4	-4	-4			
				-4	-4	-4	-4 / 12	12	12	12 / -4	-4	-4	-4	-4				
					-4	-4	-4 / 12	12	12	12 / -4	-4	-4	-4					
						-4	-4	-4 / 12	12	12	12 / -4	-4						
							-4	-4 / 12	12	12	12 / -4	-4						
								-4	-4 / 12	12	12	12 / -4						

Figure 5.58. Initial economic block model.

i \ j	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
1	-4	-4	-4	-4	-4	8	12	12	0	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
2		-4	-4	-4	-4	0	12	12	8	-4	-4	-4	-4	-4	-4	-4	-4	-4	
3			-4	-4	-4	-4	8	12	12	0	-4	-4	-4	-4	-4	-4	-4		
4				-4	-4	-4	0	12	12	8	-4	-4	-4	-4	-4	-4			
5					-4	-4	-4	8	12	12	0	-4	-4	-4	-4				
6						-4	-4	0	12	12	8	-4	-4	-4					
7							-4	-4	8	12	12	0	-4						
8								-4	0	12	12	8	-4						
9									-4	12	12	12	0						

Figure 5.59. Final economic block model (Lerchs & Grossmann, 1965).

Row	Current Value	Revised Value
i = 1	12	12
i = 2	12	24 = 12 + 12
i = 3	8	32 = 12 + 12 + 8
i = 4	0	32 = 12 + 12 + 8 + 0
i = 5	-4	28 = 12 + 12 + 8 + 0 - 4
i = 6	-4	24 = 12 + 12 + 8 + 0 - 4 - 4
i = 7	-4	20 = 12 + 12 + 8 + 0 - 4 - 4 - 4
i = 8	-4	16 = 12 + 12 + 8 + 0 - 4 - 4 - 4 - 4

Figure 5.60. Calculation of the cumulative sums for column 6.

where M_{ij} is the profit realized in extracting a single column with block (i, j) at its base and m_{kj} is the net value of block (k, j). Applying the equation to find the value of the column for $j = 6$, $i = 3$

$$M_{36} = \sum_{k=1}^{3} m_{k6} = m_{16} + m_{26} + m_{36}$$
$$= 12 + 12 + 8 = 32$$

Columns

	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1	-4	-4	-4	-4	-4	8	12	12	0	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
2	-8	-8	-8	-8	-8	8	24	24	8	-8	-8	-8	-8	-8	-8	-8	-8	-8	
3	-12		-12	-12	-12	4	32	36	20	8	-12	-12	-12	-12	-12	-12	-12		
4				-16	-16	0	32	48	32	0	-16	-16	-16	-16	-16	-16			
5					-20	-4	28	56	44	12	16	-20	-20	-20	-20				
6						-8	24	56	56	24	-8	-24	-24	-24					
7							20	52	64	36	4	24	-28						
8							16	48	64	48	16	-16	-32						
9									60	56	28	-4	32						

Figure 5.61. Completed cumulative sums (Lerchs & Grossmann, 1965).

The new table of values obtained by applying this process to all columns is shown in Figure 5.61. The next step in the process is to add row $i = 0$ containing zero's. A zero is also added at position ($i = 0, j = 0$). This revised table is shown in Figure 5.61.

It is now desired to develop an overall cumulative sum as one moves laterally from left to right across the section. Beginning with the extreme top left hand real block, the values of three blocks are examined:

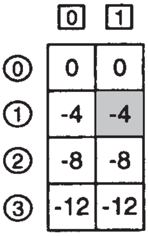
1. One directly above and to the left.
2. One on the left.
3. One directly below and to the left.

Of the three, that block which when its value is added to the block in question yields the most positive sum is selected. An arrow is drawn from the original block to that block. This sum is substituted for that originally assigned and becomes the value used for subsequent calculations.

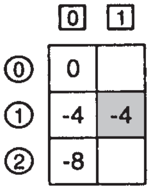
Figure 5.62 shows the process for block (1, 1). This process is continued, working down the first column, then down the next column to the right, until all blocks have been treated. It should be pointed out that the reason some of the blocks on the section have not been filled in is that they fall outside of the bounds of the ultimate pit. Figure 5.63 shows the results when the process has been completed through column 7.

One can examine the relationship between the values in the current table to the initial blocks (Fig. 5.59). If one follows the arrows beginning at the value 32, the pit which results is indicated by the shaded line (Fig. 5.64).

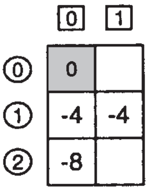
Superimposing the same pit on the block model one finds that the cumulative value of the blocks is 32 (Fig. 5.65). Moving up to the block containing the value 60 in Figure 5.64 one follows the arrows to outline the pit. The value obtained by summing the blocks is also 60 (Fig. 5.65). Therefore the technique provides a running total of the value of the pits defined by following the arrows. At this point in the calculation, the optimum pit has a value of 84. Figure 5.66 shows the results when the summing process is completed. The optimum pit is that which has the maximum cumulative value. To determine this, one moves from right to left along row 1 until the largest value is encountered. The arrows are then followed around to give the optimum pit outline on the section.



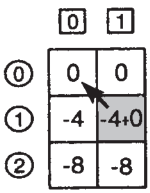
1. Beginning with Block 1,1 (the upper left hand corner of the section) the neighboring blocks in the Row above (Row 0), the same row (Row 1) and the Row below (Row 2) of the column to the left of column [0] are examined.



2. Of these three, the largest positive block is chosen.



3. In this case it is Block 0,0.



4. The sum of Block 1,1 and Block 0,0 is found and substituted for the original value of Block 1,1 in the Figure. In this case it is -4 + 0 or -4. An arrow is drawn between Block 1,1 and the selected block. This process is continued moving down column 1. When completed, column 2 is begun working from the top down.

Figure 5.62. Procedure for determining the cumulative maximum value and maximizing direction.

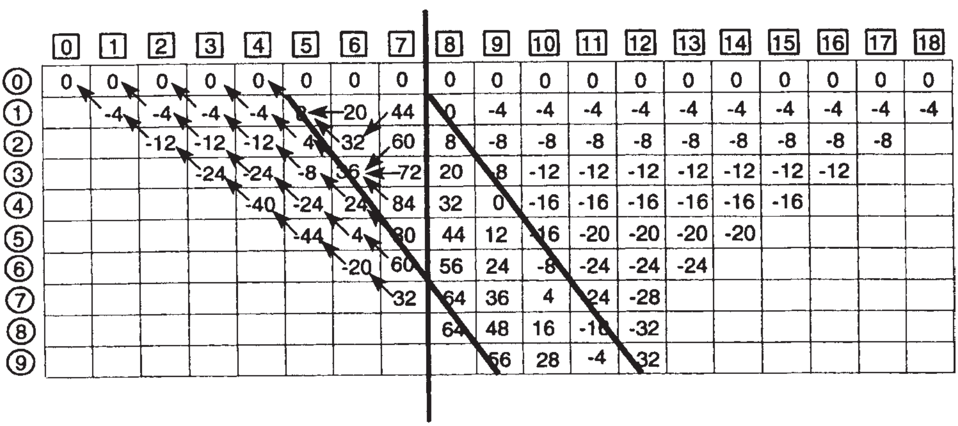


Figure 5.63. Progression of summing process through column 7.

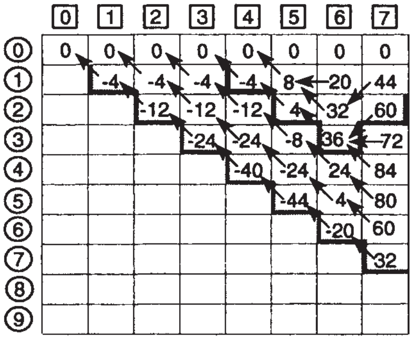


Figure 5.64. Pit determination and total value by following the arrows.

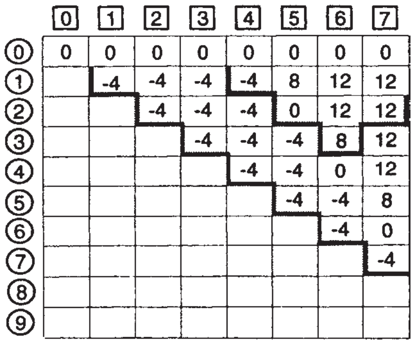


Figure 5.65. Individual block values for the two partial pits.

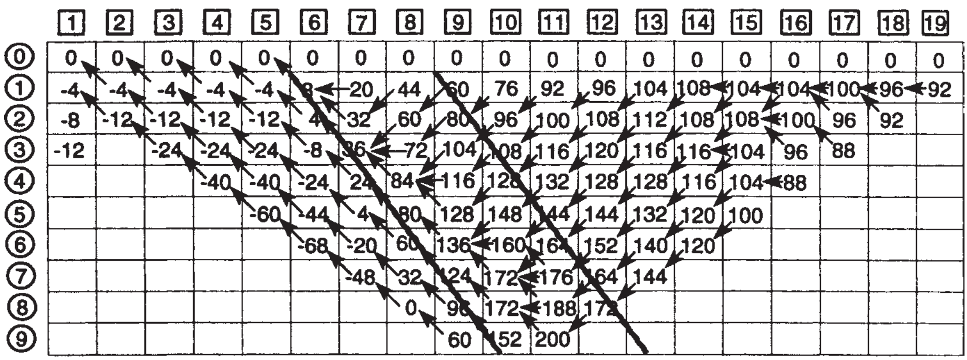


Figure 5.66. The summing process has progressed through the entire section (Lerchs & Grossmann, 1965).

The pit is shown in Figure 5.67. The value is 108. The relationship between the values in Figure 5.67 and the actual block values can be seen by comparing Figures 5.67 and 5.68.

To complete the analysis one calculates:

$$\text{Net value} = 108 \times \$1000 = \$108,000$$

$$\text{Total tons} = 36\,\text{blocks} \times 10,000\ \text{tons/block} = 360,000\,\text{tons}$$

$$\text{Tons ore} = 20\,\text{blocks} \times 10,000 = 200,000\,\text{tons}$$

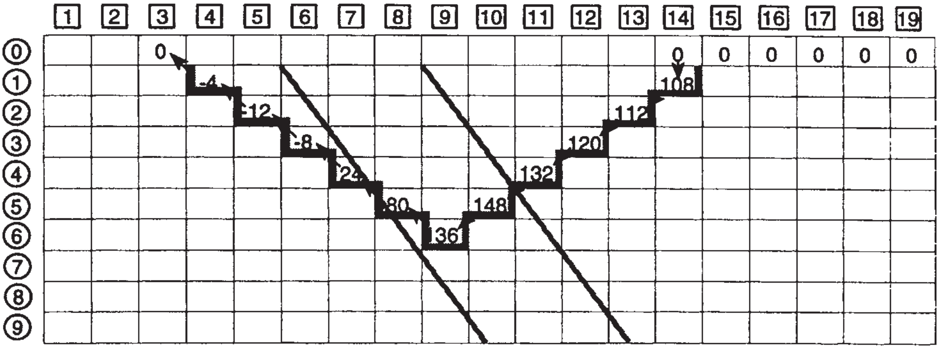


Figure 5.67. Optimum pit determination.

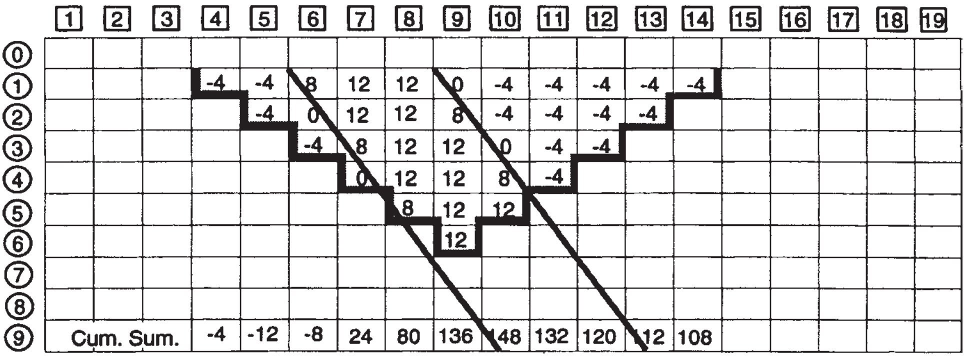


Figure 5.68. Optimum pit limits superimposed on the block model.

$$\text{Tons waste} = 16\,\text{blocks} \times 10,000 = 160,000\,\text{tons}$$

$$\text{Stripping ratio} = \frac{16}{20} = 0.8$$

$$\text{Average profitability/ton} = \frac{\$108,000}{360,000} = \$0.30/\text{ton}$$

The expressions used (Lerchs & Grossmann, 1965) to calculate the 'derived' profit P_{ij} (as given in Fig. 5.66) are

$$P_{ij} = \begin{cases} 0, & i = 0 \\ M_{ij} + \max\limits_{k=-1,0,1} P_{i+k,j-1} & i \neq 0 \end{cases} \tag{5.16}$$

The maximum is indicated by a arrow going from (i, j) to $(i + k, j - 1)$. P_{ij} is the maximum possible contribution of columns 1 to j to any feasible pit that contains (i, j) on its contour. If the element (i, j) is part of the optimum contour, then this contour to the left of element (i, j) can be traced by following the arrows starting from element (i, j). Any feasible pit must contain at least one element of the first row. If the maximum value of P in the first row is positive then the optimum contour is obtained by following the arrows from and to the left of this element. If all elements of the first row are negative, then no contour with positive profit exists.

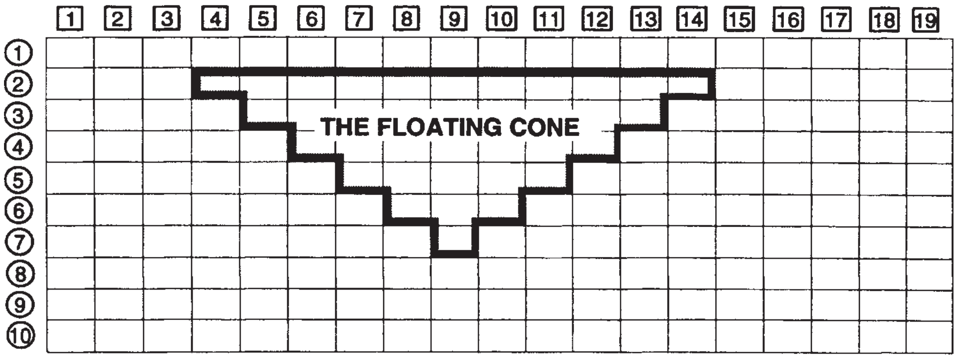


Figure 5.69. The floating cone used to evaluate the final pit limits.

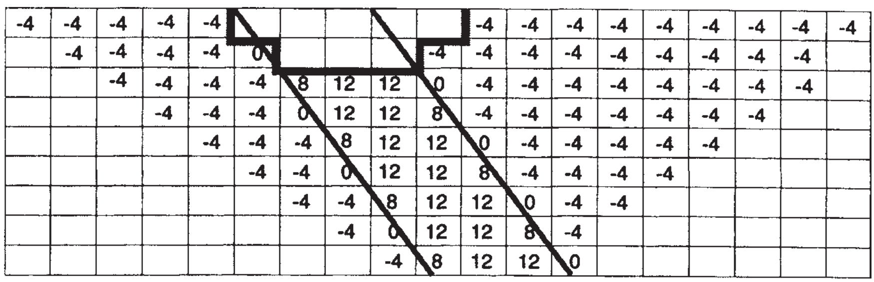


Figure 5.70. Situation after floating down two rows.

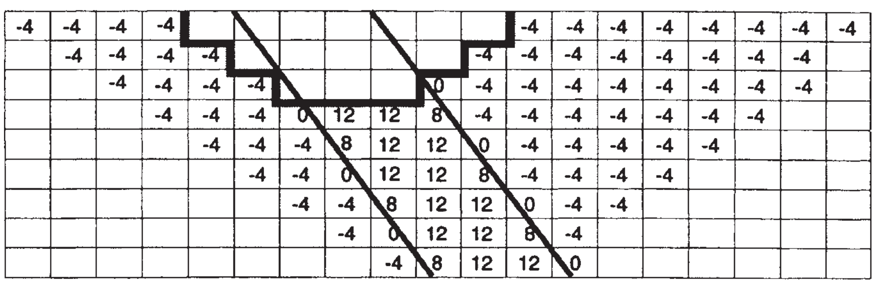


Figure 5.71. Situation after floating down three rows.

This same section (Fig. 5.59) can be evaluated using the floating cone technique. The ‘stepped’ cone of Figure 5.69 has been used to float over the section. Figures 5.70 through 5.73 show the intermediate pits with the final pit given in Figure 5.74. As can be seen the result is the same as when using the Lerchs-Grossmann approach. An advantage of the floating cone procedure is that a variety of slope angles can be modelled.

As shown in Figure 5.75, one can also examine the pit limits by considering the lengths along the perimeter. Consider the left-hand side (LHS) of the pit. The horizontal segments

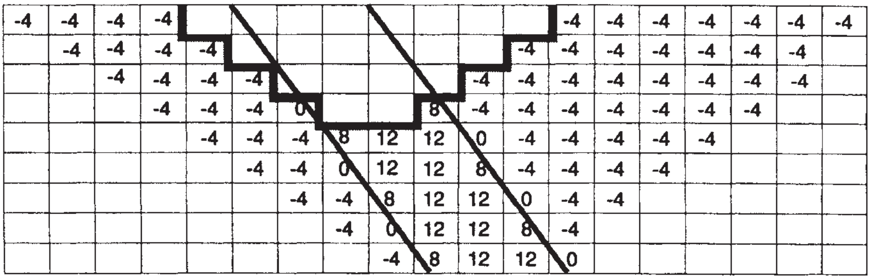


Figure 5.72. Situation after floating down four rows.

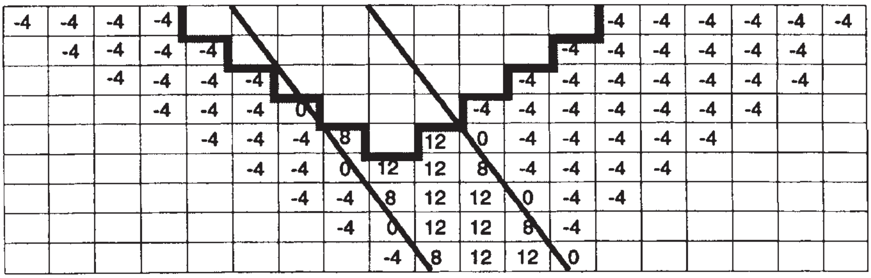


Figure 5.73. Situation after floating down five rows.

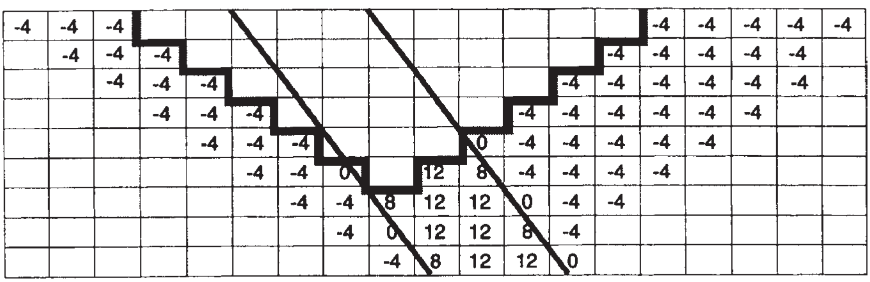


Figure 5.74. Situation after floating down six rows.

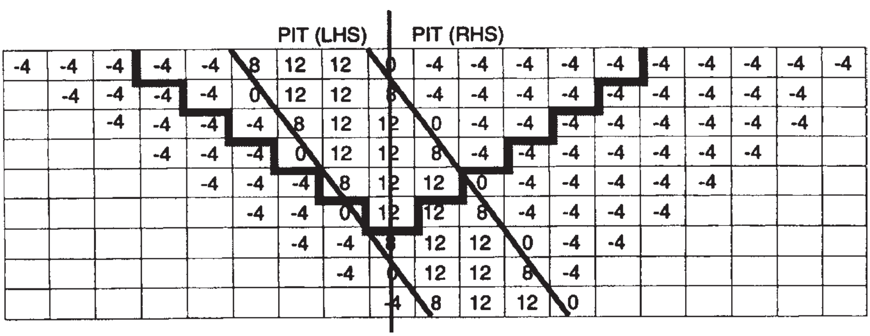


Figure 5.75. Pit resulting from the manual approach using the stripping curves.

are considered first to determine the vertical position. When the pit is slightly above the known true position, the sum of the horizontal segments is

$$-4 - 4 - 4 + 0 + 8 + 6 = +2$$

When it is slightly below than the sum is

$$-4 - 4 - 4 - 4 - 4 + 0 + 4 = -16$$

Examining the vertical segments to find the correct horizontal position is done next. When the contour is slightly inside the correct position the sum is

$$-4 - 4 - 4 + 0 + 8 + 12 = +8$$

When it is slightly outside, the sum is

$$-4 - 4 - 4 - 4 - 4 + 0 = -20$$

Therefore the correct position is as given by the floating cone and Lerchs-Grossmann procedures. The same procedure could be followed on the right hand side.

5.6 MODIFICATION OF THE LERCHS-GROSSMANN 2-D ALGORITHM TO A 2½-D ALGORITHM

The two-dimensional dynamic programming algorithm for determining the optimal configuration of blocks to be mined in cross-section was presented in the previous section. The technique is elegant yet simple. Like all other two-dimensional techniques it has the drawback that extensive effort is usually required to smooth out the pit bottom and the pit ends as well as to make sure that the sections fit with one another. As indicated by Johnson and Sharp (1971), this smoothing seldom results in an optimal three-dimensional pit. The modification which they proposed to the basic algorithm, as described in this section has been referred to as the 2½-D algorithm (Barnes, 1982). An overall view of the block model to be used is shown in Figure 5.76.

The column of blocks representing in the longitudinal section ($i = 4, j = 1$)

2	Level 1
−3	Level 2
1	Level 3
−7	Level 4

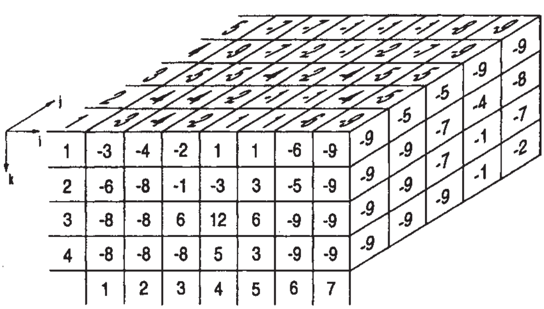


Figure 5.76. Block model used for the 2½-D projection bound (Johnson & Sharp, 1971).

This process is repeated for each of the other sections. The resulting longitudinal section is shown in Figure 5.79. The 2-D algorithm is now applied and the optimal contour becomes that given in Figure 5.80.

Figure 5.77 shows the 5 sections of a block model. Beginning with section 1, the optimum pit outlines are determined, given that one must mine to a given level k. In this case there are four levels to be evaluated. The results are shown in Figure 5.78. Each outline

-3	-4	-2	1	1	-6	-9
-6	-8	-1	-3	3	-5	-9
-8	-8	6	12	6	-9	-9
-8	-8	-8	5	3	-9	-9

a. Section 1 (j=1)

-4	-4	-2	-1	-1	-4	-5
-9	-9	3	4	3	-9	-9
-8	-8	5	5	5	-9	-9
-9	-9	-9	4	4	-9	-9

b. Section 2 (j=2)

-5	-5	-4	-2	-4	-5	-5
-8	-2	4	5	4	-3	-7
-9	-9	3	12	7	-7	-7
-9	-9	-9	7	7	5	-9

c. Section 3 (j=3)

-9	-7	-2	-1	-2	-7	-9
-6	-5	1	5	1	-5	-4
-9	-9	4	5	9	-2	-1
-4	-1	-2	9	3	-2	-1

d. Section 4 (j=4)

-7	-7	-1	-1	-1	-8	-9
-9	-9	-1	-2	-1	-9	-8
-9	-5	1	3	1	-5	-7
-2	2	1	1	2	1	-2

e. Section 5 (j=5)

Figure 5.77. Block sections for the 2½-D model (Johnson & Sharp, 1971).

Level k=1 — S_{ik} value=2

k \ i	1	2	3	4	5	6	7
1	-3	-4	-2	1	1	-6	-9
2							
3							
4							

Level k=3 — S_{ik} value=1

k \ i	1	2	3	4	5	6	7
1	-3	-4	-2	1	1	-6	-9
2	-6	-8	-1	-3	3	-5	-9
3	-8	-8	6	12	6	-9	-9
4							

Level k=2 — S_{ik} value=-3

k \ i	1	2	3	4	5	6	7
1	-3	-4	-2	1	1	-6	-9
2	-6	-8	-1	-3	3	-5	-9
3							
4							

Level k=4 — S_{ik} value=-7

k \ i	1	2	3	4	5	6	7
1	-3	-4	-2	1	1	-6	-9
2	-6	-8	-1	-3	3	-5	-9
3	-8	-8	6	12	6	-9	-9
4	-8	-8	-8	5	3	-9	-9

Figure 5.78. Analysis of section 1.

k \ j	1	2	3	4	5
1	2	-1	-2	-1	-1
2	-3	0	-5	0	-5
3	1	3	5	-7	-14
4	-7	-13	7	-6	-33

Figure 5.79. Longitudinal section (Johnson & Sharp, 1971).

k \ j	1	2	3	4	5
1	2				-1
2	-3	0		0	-5
3	1	3	5	-7	-14
4	-7	-13	7	-6	-33

Figure 5.80. Optimal longitudinal contour (Johnson & Sharp, 1971).

is found using a slightly modified version of the Lerchs-Grossmann 2-D algorithm. The net values for each of the four pits on this section are then determined. As can be seen these are:

Level 1: $S_{11} = 2$

Level 2: $S_{12} = -3$

Level 3: $S_{13} = 1$

Level 4: $S_{14} = -7$

To combine all of the transverse (ik) sections, a longitudinal section will now be taken. In viewing Figure 5.78, it is seen that the deepest mining on section 1 ($j = 1$) occurs at column $i = 4$. A column [$i = 4$, $j = 1$ (section 1)] of blocks will now be formed representing the net value of the cross-section as mined down to the various levels.

The net value for the material contained within the pit is found by summing the block values along the final contour. Here it becomes

$$\text{Pit value} = 2 + 0 + 5 + 0 - 1 = 6$$

Now that the bottom levels for each of the transverse sections have been determined, one goes back to these sections and selects the appropriate one. These are summarized in Figure 5.81.

The data and solution for a somewhat more complicated problem are given in Figures 5.82 and 5.83, respectively. The block height to length ratio along the longitudinal section (the axis of mineralization) is 1 to 2 while for the cross-sections the block height and length are equal (1 to 1). Hence to maintain a maximum pit slope of 45 degrees, the pit slope can change by two blocks per section along the length but by only block per column across the width.

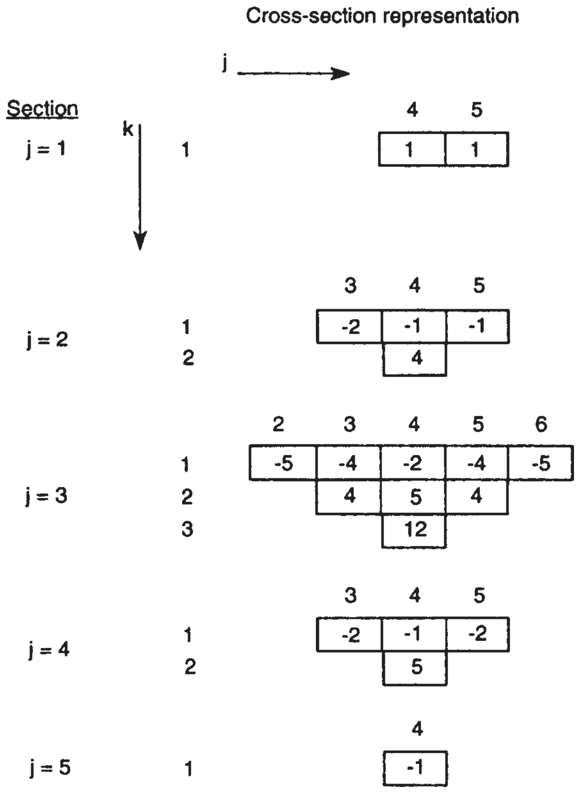


Figure 5.81. Optimal longitudinal pit sections (Johnson & Sharp, 1971).

5.7 THE LERCHS-GROSSMANN 3-D ALGORITHM

5.7.1 *Introduction*

When evaluating final pit dimensions based upon reserves expressed in the form of a grade block model, the overall objective is to find a grouping of blocks such that a selected parameter, for example:

– profit,
– metal content, or
– marginal value

is maximized. In preceding sections, some 2-dimensional approaches have been discussed. This is however a 3-dimensional problem and to obtain a true optimum such an approach is required.

For an orthogonal set of blocks there exist two basic geometries of interest for approximating an open-pit. They are:

– the 1–5 pattern, where 5 blocks are removed to gain access to one block on the level below and

– the 1–9 pattern, where 9 blocks are removed to gain access to one block on the level below.

Section j=1

-1	-1	-1	0	-1	-1	-1	-1	0	0	0	0
-1	-1	-2	-2	-2	-2	-2	-2	-2	-2	-2	-2
-1	-2	-2	-2	-2	-2	-2	-2	-2	-3	-3	-3
-2	-3	-3	-4	-3	-4	-4	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=2

-1	-1	-1	0	0	-1	-1	-1	-1	-1	-1	-1
-1	-1	-2	-2	-2	-1	-2	-2	-2	-2	-2	-2
-2	-2	3	5	4	-2	-2	-2	-2	-2	-2	-2
-3	-3	-3	1	-4	-4	-3	-3	-3	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=3

0	0	0	0	0	-1	-1	0	0	0	0	0
-2	-2	-2	-1	1	0	1	-2	-2	-2	-2	-1
-3	-3	-3	-3	4	6	5	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=4

0	0	0	0	0	0	0	0	0	0	0	0
0	-1	-1	-1	-1	-1	0	-2	-2	-2	-2	-1
-2	-2	-2	-2	3	4	5	4	1	-4	-4	-4
-3	-4	-4	-4	-2	-2	1	-2	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=5

-1	0	0	0	0	0	0	0	0	0	0	0
-2	-2	-2	-2	-2	-2	-1	-2	-2	-1	-1	0
-2	-2	-2	-2	-2	-2	-2	-2	-2	-2	-2	-2
-3	-3	-3	-3	-3	3	4	-4	-4	-3	-3	-3
-4	-4	-4	-4	1	3	6	-3	-1	-2	-4	-4

Section j=6

-1	-1	-1	-1	-1	-1	0	0	0	0	0	0
-2	-2	-2	-2	-2	-1	-2	-2	-2	-2	-2	-1
-3	-2	-2	-3	-4	-4	-4	-4	-3	-4	-4	-4
-4	-4	-4	-4	1	2	3	1	0	-4	-4	-4
-4	-4	-4	-4	0	3	5	2	-4	-4	-4	-4

Section j=7

-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-2	-2
-2	-2	-2	-2	-2	-1	-2	-2	-2	-4	-4	-4
-4	-4	-4	-4	-3	-2	-1	-1	-4	-4	-4	-4
-4	-4	-4	-4	1	8	1	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=8

-1	-1	-2	-2	-2	-2	-2	-1	-2	-2	-2	-1
-2	-2	-4	-4	-4	-4	1	2	1	-4	-4	-2
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
-4	-4	-4	5	-4	-4	-4	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=9

-1	-1	-1	-1	-1	-1	-2	-2	-2	-1	0	-1
-2	-2	-2	-4	-4	-4	-4	-4	2	3	1	-2
-4	-3	-3	-4	-3	-2	-4	-4	-4	-4	-4	-4
-4	-4	-4	5	7	20	12	-4	-4	-4	-4	-4
-4	-4	-4	-4	8	20	9	-4	-4	-4	-4	-4

Section j=10

-3	-2	-3	-3	-3	-2	-3	-4	-1	1	2	1
-4	-4	-4	-4	-4	-4	-4	-4	-4	-3	1	-4
-4	-3	-3	-4	-3	-2	-4	-4	-4	-4	-4	-4
-4	-4	-4	5	7	20	12	-4	-4	-4	-4	-4
-4	-4	-4	-4	8	20	9	-4	-4	-4	-4	-4

Section j=11

-4	-4	-4	-4	-4	-4	-3	-4	-4	2	1	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
-5	-5	-5	-4	-4	-4	-4	-4	-4	-4	-4	-4
-6	-6	-5	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=12

-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4
-4	-4	-4	-4	-4	-4	-3	-4	-4	-4	-4	-4
-5	-5	-4	-4	-4	-4	-4	-4	-3	-4	-4	-4
-6	-5	-5	-4	-4	-4	-4	-4	-4	-4	-4	-4
-7	-6	-5	-4	-4	-4	-4	-4	-4	-4	-4	-4

Section j=13

-5	-4	-4	-4	-4	-3	-4	-4	-4	-4	-4	-4
-6	-5	-4	-4	-4	-4	-3	-4	-4	-4	-4	-4
-7	-6	-5	-4	-4	-4	-4	-4	-4	-4	-4	-4
-8	-7	-6	-5	-4	-4	-4	-4	-4	-4	-4	-4
-9	-8	-7	-6	-5	-4	-4	-4	-4	-4	-4	-4

Section j=14

-7	-6	-5	-4	-4	-4	-3	-4	-4	-4	-4	-4
-8	-7	-6	-5	-4	-4	-4	-4	-4	-4	-4	-4
-9	-8	-7	-6	-5	-4	-4	-4	-4	-4	-4	-4
-10	-9	-8	-7	-6	-5	-4	-4	-4	-4	-4	-4
-11	-10	-9	-8	-7	-6	-5	-4	-5	-4	-4	-4

Figure 5.82. Block sections for a more complicated example (Johnson & Sharp, 1971).

Column (i)	Optimal boundary blocks (level, k) 1	2	3	4	5	6	7	8	9	10	11	12	Yields
Sections (j)													
1	0	0	0	0	0	0	0	0	0	0	0	0	0
2	0	0	0	0	1	1	1	0	0	0	0	0	-2
3	0	0	1	2	3	3	3	2	1	0	0	0	12
4	0	0	1	2	3	3	3	3	2	1	0	0	9
5	0	0	0	0	1	1	1	1	0	0	0	0	0
6	0	0	0	0	0	0	1	0	0	0	0	0	0
7	0	0	0	0	0	0	0	0	0	0	0	0	0
8	0	0	0	0	0	0	0	0	0	0	0	0	0
9	0	0	0	0	0	0	0	0	0	0	1	0	0
10	0	0	0	0	0	0	0	0	0	1	2	1	5
11	0	0	0	0	0	0	0	0	0	1	1	0	3
12	0	0	0	0	0	0	0	0	0	0	0	0	0
13	0	0	0	0	0	0	0	0	0	0	0	0	0
14	0	0	0	0	0	0	0	0	0	0	0	0	0
												Optimal pit yields	27

Figure 5.83. The solution for the Figure 5.82 block model (Johnson & Sharp, 1971).

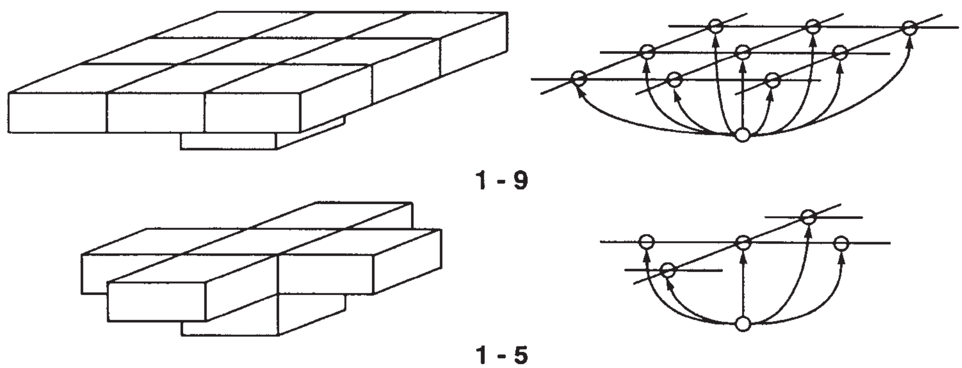


Figure 5.84. Representation of the 1-5 and 1-9 block constraints (Laurent et al., 1977).

The geometric configurations and equivalent graphic representations for these two geometries are shown in Figure 5.84. The nodes represent the physical blocks. The arrows (directed arcs) point toward those blocks immediately above which must first be removed before the underlying block can be mined. Each block has a weight associated with it. In general, the weight assigned is equal to the value of the parameter being maximized. Often this is the net economic value. The weight may be positive or negative.

Lerchs & Grossmann (1965) published the basic algorithm which when applied to a 3-D directed graph (block model) would yield the optimum final pit outline. This section presents in a simplified way, the basic concepts. They are illustrated by examples.

5.7.2 *Definition of some important terms and concepts*

There are a number of terms and concepts which are taken from graph theory (Lerchs & Grossmann, 1965; Laurent et al., 1977). The authors have tried to simplify them and yet retain their basic meaning.

Figure 5.85a is a section through a simple 2-D block model. As can be seen it consists of 6 blocks.

Each block is assigned a number (x_i) which indicates its location within the block model. In the case shown in Figure 5.85a, the block location are x_1, x_2, x_3, x_4, x_5, and x_6. One would know from the construction of the block model that the block designated as x_1 would actually have center coordinates of (2000, 3500, 6800). If there were 100,000 blocks in the block model, then x_i would go from x_1 to $x_{100,000}$.

The file of node locations may be expressed by

$$X = (x_i)$$

Figure 5.85b shows the 6 blocks simply redrawn as circles while maintaining their positions in 2-D space. For the graph theory application each of these circles is now called a 'node'. Straight line elements (called 'edges' in graph theory) are now added connecting the lower nodes to the nearest overlying neighbors. For the 3-D representations shown in Figure 5.84 there would be 9 edges for each underlying block in the 1-9 model and 5 for the 1-5 model. In this 2-D model there are 3 edges for each underlying block. The nearest overlying neighbors for node x_5 as shown in Figure 5.85c are nodes x_1, x_2 and x_3. The physical connection of node x_5 to node x_1 can be expressed either as (x_5, x_1) or (x_1, x_5). In Figure 5.85c, the nodes have been connected by 6 edges:

$$(x_1, x_5) = (x_5, x_1)$$

$$(x_2, x_5) = (x_5, x_2)$$

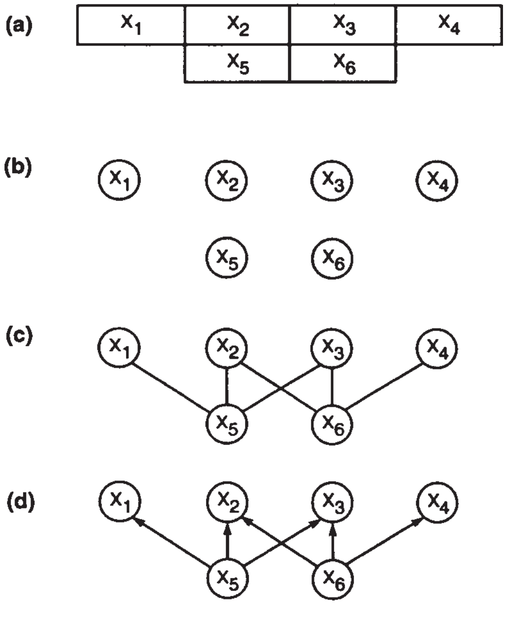


Figure 5.85. Steps in the model building process.

$(x_3, x_5) = (x_5, x_3)$

$(x_2, x_6) = (x_6, x_2)$

$(x_3, x_6) = (x_6, x_3)$

$(x_4, x_6) = (x_6, x_4)$

These edges (e_{ij}) can be described by

$$e_{ij} = (x_i, x_j)$$

The set containing all edges is given the symbol E:

$$E = (e_{ij})$$

A graph is defined as:

Graph: A graph $G = (X, E)$ is defined by a set of nodes x_i connected by ordered pairs of elements called edges $e_{ij} = (x_i, x_j)$.

The next step is to indicate which overlying blocks (nodes) must be removed prior to removing any given underlying block (node). This adds the required sequencing (flow) from the lowest most to the upper most blocks (nodes). To accomplish this, arrows are attached to the edges (lines) connecting the nodes (blocks) pointing in the direction of removal. This has been done in Figure 5.85d. By adding an arrowhead (direction) to an edge, the edge becomes an arc,

$$a_{kl} = (x_k, x_l)$$

An arc denoted by (x_k, x_l) means that the 'flow' is from node x_k to node x_l (the arrowhead is on the x_l end). The set containing all arcs in given the symbol A

$$A = (a_{kl})$$

The graph (G) consisting of nodes $\{X\}$ and arcs $\{A\}$ is called a directed graph.

Directed graph: A directed graph $G = (X, A)$ is defined by a set of nodes x_l connected by ordered pairs of elements $a_{kl} = (x_k, x_l)$ called the arcs of G.

One can consider the entire set of nodes (blocks) and the arcs connecting them (the directed graphs $G(X, A)$) or only a portion of it. A subset (Y) is referred to as a directed subgraph and represented by $G(Y, A_Y)$. An example of a directed subgraphs is shown in Figure 5.86. There are a great number of these subgraphs in the overall graph.

Subgraph: A directed subgraph $G(Y)$ is a subset of the directed graph $G(X, A)$. It is made up of a set Y of nodes and all of the arcs A_Y which connect them.

To this point in the discussion we have considered (1) the physical location of the blocks in space, (2) the connection of the blocks with one another and (3) the fact that overlying

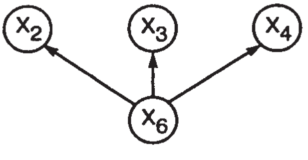


Figure 5.86. Example of a subgraph.

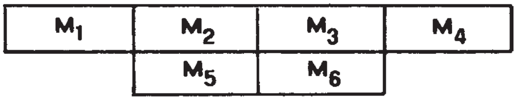


Figure 5.87. The weights assigned to the blocks of Figure 5.85.

blocks must be removed prior to mining lower blocks. Nothing has been said regarding the values of the individual blocks. Each block (x_i) has an associated weight (m_i). This is shown in Figure 5.87 for the blocks in Figure 5.85a. Although we have used net value as the assigned weight (m_i) to this point in the book other measures of worth or content can also be applied (profit, mineral content, etc.).

From a mining view point, the subgraph consisting of the four blocks. x_1, x_2, x_3 and x_5 could form a physically feasible pit. The subgraph consisting of blocks x_2, x_3, x_4 and x_6 could form another. A third possibility of a feasible pit is formed by the six blocks x_1, x_2, x_3, x_4, x_5 and x_6. There are many other feasible combinations. The subgraph x_2, x_3 and x_6 is not feasible since one of the overlying blocks x_4 has not been included. The term 'closure' is used to indicate a feasible subgraph.

Closure: Closure from the viewpoint of a mining engineer is simply a subgraph Y yielding a feasible pit.

Each one of these feasible pits (subgraphs) has an associated total weight (value). The challenge for the mining engineer is to find that one pit (subgraph) out of the great many possible which yields the maximum value. In graph theory this is referred to as finding the directed subgraph of 'maximum closure'.

Maximum closure: Again from the viewpoint of the mining engineer, maximum closure is that closure set, out of all those possible, which yields the maximum sum of block weights, i.e. where $M_Y = \sum m_i$ is a maximum.

A procedure based upon the application of graph theory is used to identify and sort through the various feasible pits in a structured way to find that yielding the maximum value. This corresponds to the optimum pit. To better follow the discussion the following definitions are introduced.

Circuit: A circuit is a path in which the initial node is the same as the find (terminal) node.

Chain: A chain is a sequence of edges in which each edge has one node in common with the succeeding edge.

Cycle: A cycle is a chain in which the initial and final nodes coincide.

Path: A path is a sequence of arcs such that the terminal node of each arcs is the initial node of the succeeding arc.

To illustrate the process, a discussion based upon a tree analogy is used. The terms 'tree', 'root', 'branch' and 'twig' are defined below:

Tree: A tree T is a connected and directed graph containing no cycles. A tree contains one more node than it does arcs. A rooted tree is a tree with a special node, the root.

Root: A root is one node selected from a tree. A tree may have only one root.

Branch: If a tree is cut into two parts by the elimination of one arc a_{kl}, the part of the tree not containing the root is called a branch. A branch is a tree itself. The root of the branch is the node of the branch adjacent to the arc a_{kl}.

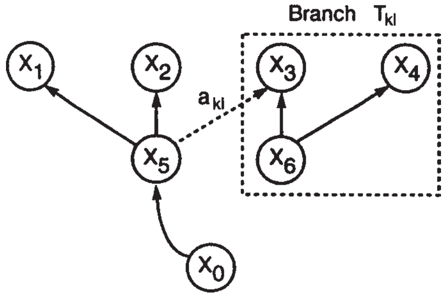


Figure 5.88. Example of a branch.

General Form

X_1	X_2	X_3	X_4
	X_5	X_6	

Assigned Weights

-4	-4	-4	-4
	10	10	

Network Equivalent

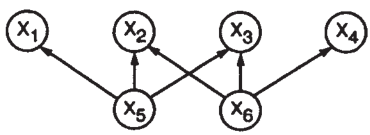


Figure 5.89. Example to be worked using the Lerchs-Grossmann 3-D algorithm.

Each arc a_{kl} of a tree T defines a branch T_{kl}. The weight W_{kl} of a branch T_{kl} is the sum of all weights associated with nodes of T_{kl}. An example of a branch is shown in Figure 5.88.

Twig: A twig is a branch of a branch.

As ‘twigs’ and ‘branches’ are added or cut from the ‘tree’, the value of the tree changes.

The Lerchs-Grossman algorithm is based upon a normalizing procedure in which a number of rules are followed. These will be demonstrated in detail in the next section.

5.7.3 *Two approaches to tree construction*

The algorithm starts with the construction of an initial tree T^0. This tree is then transformed into successive trees $T^1, T^2, \ldots, T^n$ following given rules until no further transformation is possible. The maximum closure is then given by summing the nodes of a set of well identified branches of the final tree. There are two approaches which may be used for generating the initial tree.

Approach 1. Construct an arbitrary tree having one connection to the root.

Approach 2. Construct a tree with each of the nodes connected directly to the root.

The simplest of these is Approach 2. Both approaches will however be applied to the simple example shown in Figure 5.89.

Although elementary, this is an interesting problem since the floating cone approach would suggest no mining at all. By inspection however one would strip the four waste

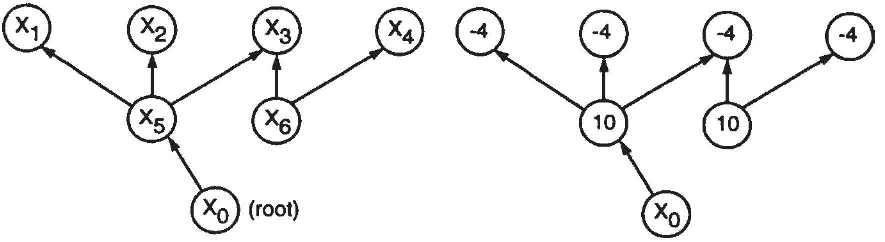


Figure 5.90. Addition of a root.

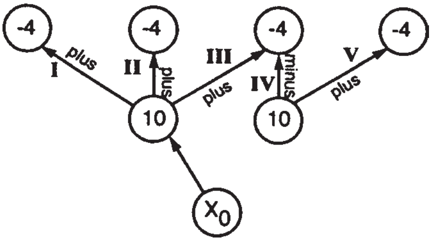


Figure 5.91. Labelling of plus and minus arcs.

blocks to uncover the two ore blocks. The net value for the resulting pit is expected to be +4.

The step-by-step approach used to demonstrate the techniques involved has been adapted from that presented by Laurent et al. (1977).

5.7.4 *The arbitrary tree approach* (*Approach 1*)

This approach (including the common normalizing procedure used with both Approaches 1 and 2) will be presented in step-by-step fashion.

Step 1. Begin by adding a root node x_0 to the directed graph and construct a tree of your choice keeping in mind the connection possibilities

From node x_5: (x_5, x_1)
(x_5, x_2)
(x_5, x_3)
From node x_6: (x_6, x_2)
(x_6, x_3)
(x_6, x_4)

The one chosen is shown in Figure 5.90. Each of the nodes (blocks) is connected to one of the others by a directed arc (arrow). One node is attached to the root.

Step 2. Each of the arcs is labelled with respect to whether it is directed away from the root (plus) or towards the root (minus). This is done in Figure 5.91.

Table 5.15. Labelling guide for arcs.

Case	Direction	Cumulative weight	Label
1	Plus	Positive	Strong
2	Plus	Null or negative	Weak
3	Minus	Positive	Weak
4	Minus	Null or negative	Strong

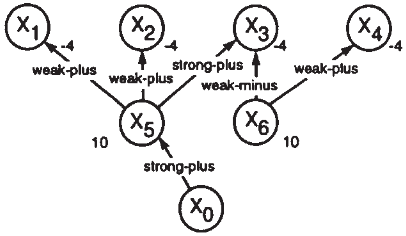


Figure 5.92. Labelling of weak and strong arcs.

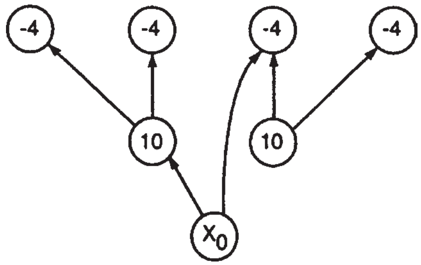


Figure 5.93. Creation of tree T^1.

Step 3. Beginning at the extremities each branch is worked back toward the trunk, summing the weights supported by the individual arcs. The objective is to add the words ‘strong’ or ‘weak’ to each of the arcs. Table 5.15 summarizes the labelling criteria.

We will begin on the left hand side of the tree. Roman numerals have been used to denote the segments for the discussion. For arc I, the direction is plus and the weight is negative. Thus the label to be attached is ‘weak’ (Case 2). For arc II, the same is true. For arc V, the direction is plus and the weight negative. The label is weak. It is also Case 2. When examining arc IV, the direction is minus and the cumulative weight is positive $(10 - 4 = 6)$. Thus the label is weak (Case 3). Continuing to arc III, the direction is plus and the cumulative weight is positive $(10 - 4 - 4 = 2)$. The label is ‘strong’ (Case 1). Figure 5.92 presents the resulting direction and label for each arc of this initial tree.

Step 4. The figure is now examined to identify strong arcs. Two actions are possible:

Action 1. A strong-minus arc: The arc (x_q, x_r) is replaced by a dummy arc (x_0, x_q). The node x_q is connected to the root.

Action 2. A strong-plus arc: The arc (x_k, x_l) is replaced by the dummy arc (x_0, x_l). The node x_l is connected to the root.

In this example there is one strong arc III. Since it is a ‘strong-plus’ arc, action 2 is taken. The arc connecting node x_5 (10) to node x_3 (-4) is removed. An arc connecting the root x_0 to node x_3 (-4) is drawn instead (Fig. 5.93). This becomes tree T^1.

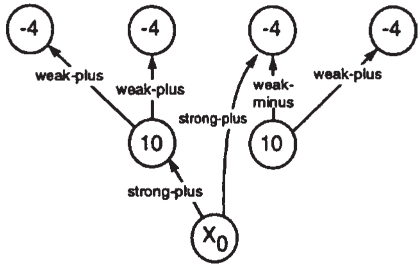


Figure 5.94. Tre1e T^1 with labels attached T^0 arcs.

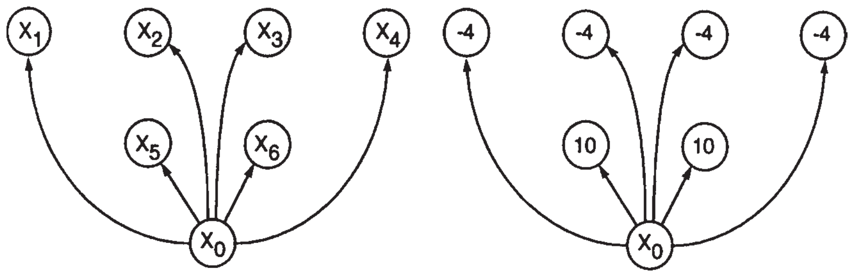


Figure 5.95. Initial tree with all root connection.

Step 5. Tree T^1 is examined in the same way as before labelling the arcs as to whether they are ‘plus’ or ‘minus’ and ‘weak’ or ‘strong’. This has been done in Figure 5.94.

Step 6. Any strong branches of the new tree not directly connected to the root are identified and the procedure discussed in Step 4 is followed. If there are no strong branches not connected to the root, the tree is said to be normalized and the process is over.

Step 7. The maximum closure consists of those nodes connected by strong arcs to the root. In this case the closure is

$$-4 - 4 + 10 - 4 - 4 + 10 = +4$$

5.7.5 *The all root connection approach* (*Approach 2*)

The step-by-step procedure is outlined below.

Step 1. Begin by adding a root node and connecting arcs between the root and each of the other nodes. For the example problem, this initial tree T^0 is shown in Figure 5.95. Note that all of the arcs are ‘plus’.

Step 2. The set (graph) of directed arcs is now split into two groups. Those connected to the root by strong-plus arcs are included in group Y^0. The others are in group X-Y^0. In this case nodes x_5 and x_6 are in group Y^0. Their sum is 20.

Step 3. One must now look at the possible connections between the two groups. Following the sequencing constraints there are 6 directed arcs which can be drawn:

For x_5: (x_5, x_1)
(x_5, x_2)
(x_5, x_3)

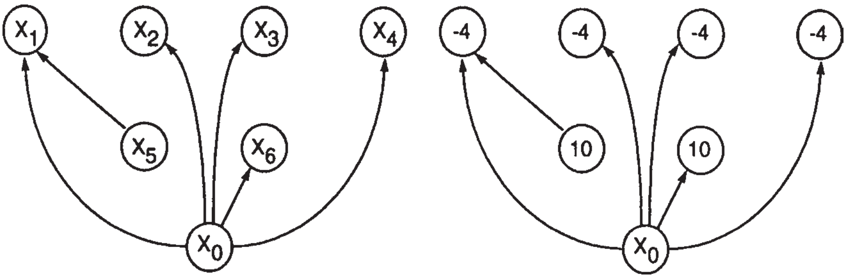


Figure 5.96. Selection of directed arc (x_5, x_1).

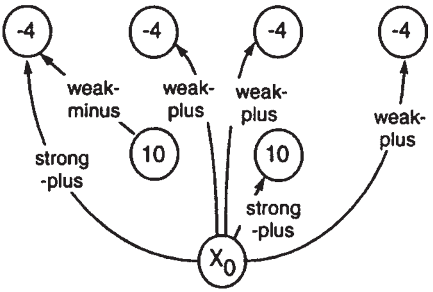


Figure 5.97. Labelling of the resulting arcs.

For x_6: (x_6, x_2)
(x_6, x_3)
(x_6, x_4)

One of these is selected. In this case it will be the connection (x_5, x_1). The directed arc (x_0, x_5) is removed and the directed arc (x_5, x_1) drawn. This is shown in Figure 5.96.

Step 4. The normalizing process is now followed. Each arc is labelled with respect to 'plus' or 'minus' and 'strong' or 'weak'. The result is shown in Figure 5.97. The arc(connection) between x_0 and x_1 is still strong-plus. Hence the members of the Y group are x_1, x_5 and x_6. The value of Y (closure) is 16.

Step 5. One now returns to step 3 to seek additional connections between the Y and X-Y (X without Y) groups. There are 5 feasible arcs

(x_5, x_2)
(x_5, x_3)
(x_6, x_2)
(x_6, x_3)
(x_6, x_4)

The arc (x_5, x_2) will be added to the tree and arc (x_0, x_2) dropped. This is shown in Figure 5.98.

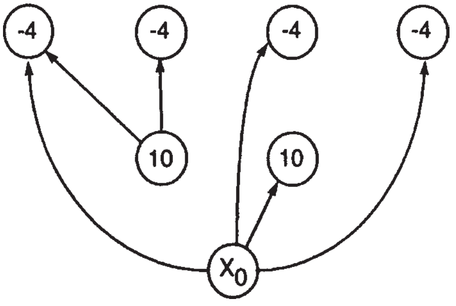


Figure 5.98. Addition of arc (x_5, x_2) and dropping arc (x_0, x_2) from the tree of Figure 5.97.

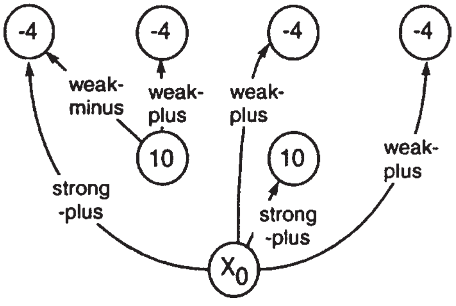


Figure 5.99. Labelling of the resulting arcs.

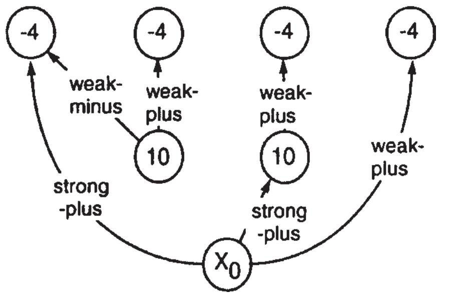


Figure 5.100. Addition of arc (x_6, x_3) and dropping arc (x_0, x_3) from the tree of Figure 5.97.

Step 6. The new tree is now normalized. The result is shown in Figure 5.99. The nodes included in Y are x_1, x_2, x_5 and x_6. The closure Y is 12.

Step 7. Returning to Step 5, there are now 3 possible connections remaining:

(x_5, x_3)

(x_6, x_3)

(x_6, x_4)

Here we will choose to add arc (x_6, x_3) and drop arc (x_0, x_3). (The choice (x_5, x_3) is an interesting one and will be evaluated later.) The resulting normalized tree is shown in Figure 5.100. The arc (x_0, x_6) remains strong-plus and hence the nodes included within Y are x_1, x_2, x_3, x_5, and x_6. The overall closure is 8.

Step 8. Returning to Step 5 there is one possible connection remaining: arc (x_6, x_4). Arc (x_0, x_4) is dropped and arc (x_6, x_4) added. The tree is normalized as before with the result

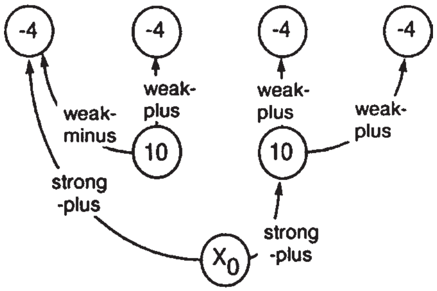


Figure 5.101. Addition of arc (x_6, x_4) and dropping arc (x_0, x_4).

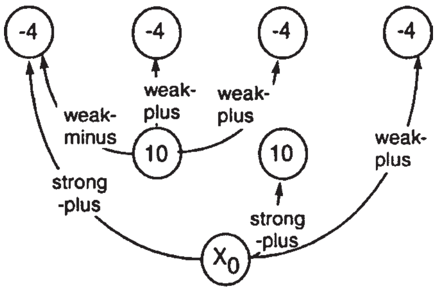


Figure 5.102. The resulting normalized tree for the alternative when arc (x_5, x_3) is added and arc (x_0, x_3) is dropped.

given in Figure 5.101. All of the nodes are now attached directly to the root by chains having one strong edge. There are no more connections to be tried.

Step 9. The maximum closure now is the cumulative sum of the nodes involved. In this case it is +4.

As was indicated, at the stage shown in Figure 5.98, it is possible to select connection (x_5, x_3). We will now return to this point and consider this option.

*Step 7**. Arc (x_5, x_3) will be added and arc (x_0, x_3) dropped. The normalized tree is shown in Figure 5.102. As can be seen, the arc (x_0, x_1) has now become weak-plus.

The only member of the Y group is now x_6. The closure is 10.

*Step 8**. One now considers the possible connections between the X-Y and Y groups. There are two possibilities.

(x_6, x_4)

(x_6, x_3)

Selecting (x_6, x_3) one obtains the normalized tree in Figure 5.103. The members of the Y group are x_6 and x_4. The closure is 6.

*Step 9**. There is one possible connection between the two groups, that one being through arc (x_6, x_3). For the tree in Figure 5.104, arc (x_0, x_1) has been dropped and arc (x_6, x_3) added. The normalized tree is shown. As can be seen all of the nodes are connected to the root through a chain containing one strong edge. There are no more possible connections. The closure is the sum of the nodes which is +4.

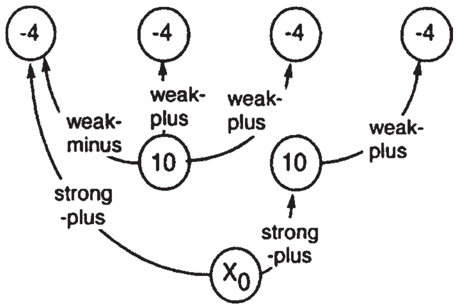


Figure 5.103. Normalized tree for connecting arc (x_6, x_3).

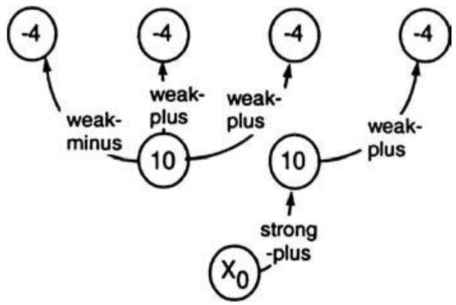


Figure 5.104. Normalized tree when adding arc (x_6, x_3) and dropping arc (x_0, x_1).

General Form

x_1	x_2	x_3	x_4
	x_5	x_6	

Assigned Weights

-10	-2	-2	-10
	10	20	

Figure 5.105. The example used to demonstrate tree cutting.

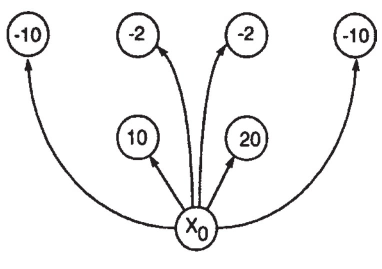


Figure 5.106. The arcs added to form the initial tree.

5.7.6 *The tree 'cutting' process*

The preceding problem was a simple case intended to familiarize the reader with the algorithm. In this example, the process of 'cutting' the tree during normalization will be demonstrated. The problem is as shown in Figure 5.105. The initial tree is as shown in Figure 5.106.

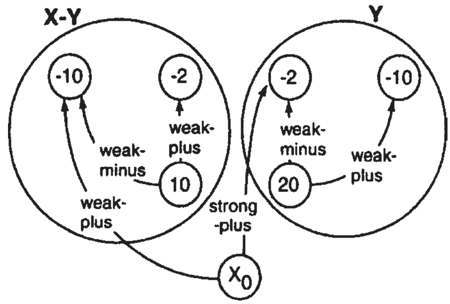


Figure 5.107. The normalized tree after 4 iterations.

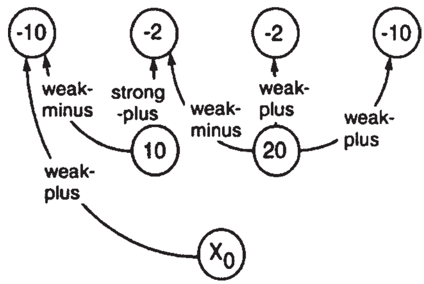


Figure 5.108. The arc (x_0, x_3) is dropped and arc (x_6, x_2) added.

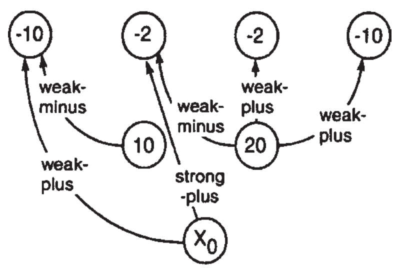


Figure 5.109. The arc (x_5, x_3) is cut and the arc (x_0, x_2) added.

The initial choices for connection are the same as before. They will be done in the following order:

For x_5: (x_5, x_1)

(x_5, x_2)

For x_6: (x_6, x_3)

(x_6, x_4)

After these 4 iterations the normalized tree would appear as in Figure 5.107.

The Y group is shown to consist of nodes x_3, x_4 and x_6. There is one remaining connection to be made between the groups (x_6, x_2). In Figure 5.108, arc (x_0, x_3) is dropped and arc (x_6, x_2) added. The tree is then normalized. As can be seen the arc (x_5, x_2) is strong-plus and not connected directly to the root. It must be cut in order to normalize the tree.

As discussed earlier the arc (x_k, x_l) is replaced by the dummy arc (x_0, x_l). In this case $x_k = x_5$, $x_l = x_2$. Thus the new arc is (x_0, x_2). This is shown in Figure 5.109. The resulting tree is in normalized form. All of the connections have been tried. The final closure is $20 - 10 - 2 - 2 = +6$. It can be seen that if Case 1 in Table 5.15 had been written as 'cumulative weight = null or positive' rather than just as 'cumulative weight = positive', then both nodes x_1 and x_5 would have been included in the maximum closure as well.

5.7.7 *A more complicated example*

Figure 5.110 shows a more complicated directed network to be analyzed using the Lerchs-Grossmann algorithm (Laurent et al., 1977). The results are shown in Figure 5.111. The interested reader is encouraged to select the upper two layers to work as an exercise. As can be seen, the two blocks at the pit bottom $+10$ and -10 would contribute a net of 0 to the section value and hence have not been mined.

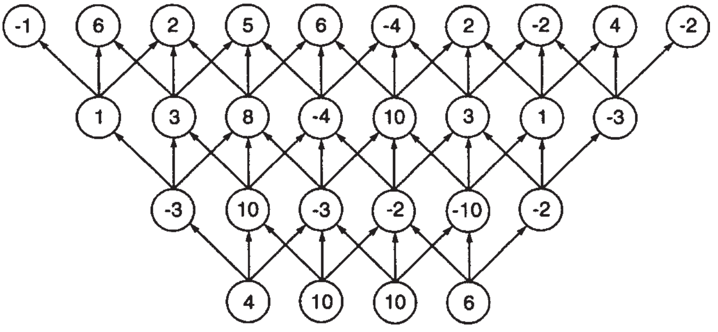


Figure 5.110. A more complicated example on which to practice the algorithm (Laurent et al., 1977).

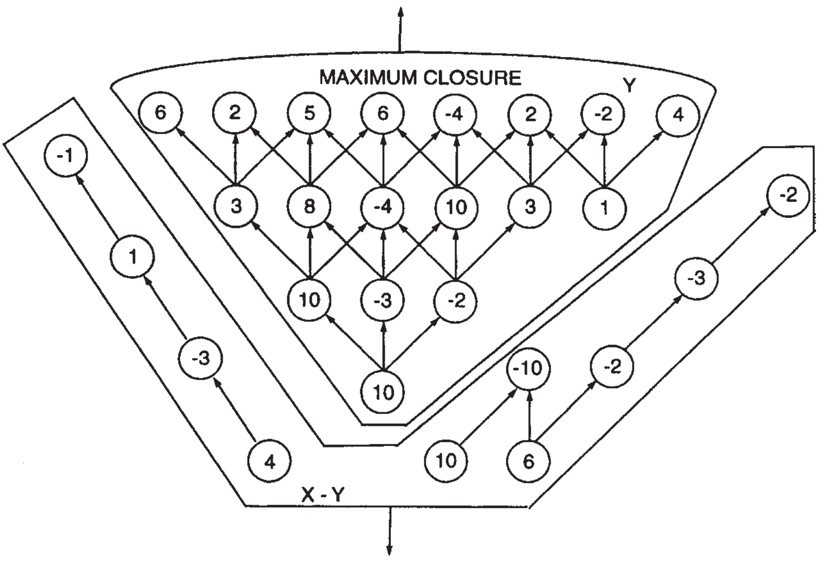


Figure 5.111. Maximum closure for the more complicated example (Laurent et al., 1977).

5.8 COMPUTER ASSISTED METHODS

5.8.1 *The RTZ open-pit generator*

Fairfield & Leigh (1969) presented a paper 'A computer program for the design of open-pits', which outlines a pit planning procedure in use by many mines today. This section will describe, using the material of Fairfield & Leigh (1969), the basic logic involved. The computer techniques using sections do not solve the problem of end sections or the smoothing of perimeters and consequently have definite limitations. The projection of volumes in the form of cones also has some of the same limitations. The approach described here is the projection of plan areas, specifically the projection of perimeter outlines.

The process begins with the development of a block model. Each block as a minimum would be assigned location coordinates and an index character dependent on the rock type or grade type.

The second step in the process is the selection of one or more base perimeters from which to generate the pit. These define the final horizontal extent of the pit at, or close to, the elevation of the pit's final base. Each perimeter so drawn becomes the trial base from which an overall pit is generated.

The information that is required to carry out these calculations is:

1. Size and shape of ore body.
2. Ore and rock types present.
3. Grades of ore.
4. Rock slope stability.
5. Size and shape of base perimeter.
6. Cutoff grades.
7. Depth of pit.
8. Unit working costs.

The trial base perimeter is defined by a series of short chords (Fig. 5.112).

The coordinates of the chord end points are read manually and then entered into the computer or are read and entered directly using a digitizer. Such a file of clockwise read perimeter points is given in Table 5.16.

At every horizon, proceeding in a clockwise order, each defining perimeter chord is considered in turn. For each, the defining end coordinates are located.

To illustrate the basic method, consider the portion of the pit base defined by chords H_1I_1 and I_1J_1 shown in Figure 5.113.

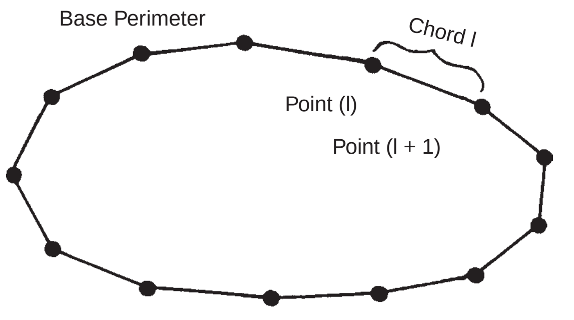


Figure 5.112. Trial based perimeter (Fairfield & Leigh, 1969).

The coordinate pairs H_1, I_1 and J_1 are projected outwards and upwards to the next upper horizon in a direction normal to the perimeter at that point. The next upper horizon occurs at a specified distance above the current horizon. The actual interval is determined by the requirements of the job at hand and is some fractional part (usually one-half) of the bench height. The slope angle used is that permitted in the particular rock type or at that pit position.

Table 5.16. Perimeter coordinates* for trial base (Fairfield & Leigh, 1969).

Perimeter coordinates							
Y	*X*	*Y*	*X*	*Y*	*X*	*Y*	*X*
445.2	601.8	420.5	619.0	404.3	641.7	391.7	659.4
394.3	681.2	397.2	705.8	401.2	726.7	408.6	746.3
410.0	768.1	415.6	796.4	424.6	817.9	428.5	843.3
428.7	869.6	424.5	888.0	432.1	916.6	428.4	945.3
437.0	969.0	415.6	984.7	393.5	1001.5	378.6	1021.8
369.7	1050.1	371.4	1077.4	378.7	1106.8	382.8	1130.0
382.8	1150.3	382.8	1170.8	382.8	1195.7	395.4	1212.5
408.1	1229.5	420.5	1246.0	434.0	1264.0	447.3	1281.7
468.7	1287.3	490.6	1292.9	516.1	1298.5	546.1	1305.2
569.3	1299.1	502.9	1292.0	616.3	1280.2	640.4	1265.8
668.2	1242.7	681.3	1228.7	695.1	1214.9	716.2	1193.8
727.9	1166.9	746.5	1147.0	763.6	1125.1	776.1	1101.4
783.7	1075.3	783.0	1048.0	770.0	1027.6	754.6	1008.0
757.4	984.5	751.0	958.6	748.8	933.5	744.7	912.9
742.4	884.9	744.9	864.6	755.4	841.3	760.6	810.1
756.9	789.2	752.9	766.9	741.7	749.0	723.7	726.2
706.5	711.6	682.8	693.9	661.7	679.5	647.0	667.9
631.1	652.4	619.9	631.5	610.7	610.1	594.9	597.2
571.7	575.8	547.3	559.6	512.2	562.4	488.3	578.3
464.3	598.4						

* Coordinate pairs are connected reading from left to right and top to bottom in the table.

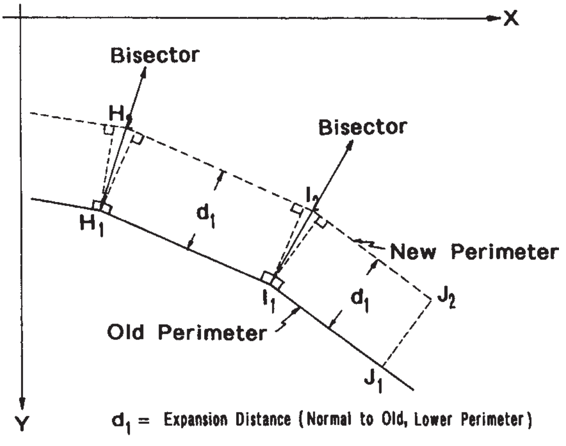


Figure 5.113. Projection of chords and perimeter points along the bisectors (Fairfield & Leigh, 1969).

If the slopes at the chord ends are different then the mean value is used.

$$\text{Slope of chord I} = \frac{1}{2}\,(\text{permitted slope at point (I)} + \text{permitted slope at point } (\text{I}+1)) \tag{5.17}$$

Sufficient information is now available:

(i) the coordinates of the chords defining the current perimeter,
(ii) the average permitted rock slopes of these chord locations,
(iii) the elevation difference to the next upper horizon,

to generate the pit perimeter on the next upper horizon. With this information the program works clockwise around the perimeter, considering pairs of adjacent chords in turn.

Consider the case where adjacent chords HI and IJ have the same permitted rock slopes θ and I is to be projected from Horizon 1 to Horizon 2 through a vertical height V. The situation is shown in plan in Figure 5.113 and in isometric view in Figure 5.114.

The calculation of the new coordinates of I on level 2 (I_2) is given below:

$$\text{Bisector of angle } \text{H}_1\text{I}_1\text{J}_1 = \overrightarrow{\text{I}_1\text{A}}$$

$$\text{Length of } \text{I}_1\text{A} = V\cot\theta$$

$$\text{Bearing angle of } \text{I}_1\text{A} = \alpha$$

$$\text{New coordinates of } \text{I}_2 = (x_1 + \text{d}x, y_1 - \text{d}y)$$
$$= (x_1 + \text{I}_1\text{A}\sin\alpha, \text{y}_1 - \text{I}_1\text{A}\cos\alpha)$$

As an example assume that

$$\theta = 45^\circ$$
$$V = 10\,\text{m}$$
$$x_1 = 510.36$$
$$y_1 = 840.98$$
$$\overrightarrow{\text{I}_1\text{A}} = \text{N}50^\circ\text{E}$$
$$\alpha = 50^\circ$$

The horizontal distance I_1A is given by

$$\text{I}_1\text{A} = V\cot\theta = 10\cot 45^\circ = 10\,\text{m}$$

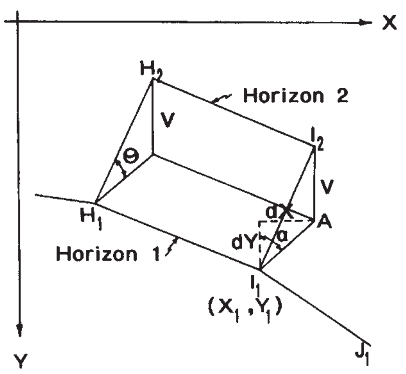


Figure 5.114. Isometric projection of chord HI (Fairfield & Leigh, 1969).

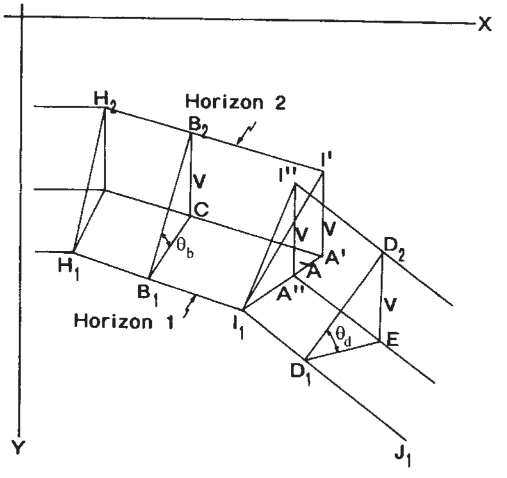


Figure 5.115. Isometric projection of adjacent chords HI and IJ (Fairfield & Leigh, 1969).

Using the bearing angle, one can calculate the distances dx and dy.

$$\mathrm{d}x = \mathrm{I_1A}\sin\alpha = 10\sin 50^\circ = 7.66$$

$$\mathrm{d}y = \mathrm{I_1A}\cos\alpha = 10\cos 50^\circ = 6.43$$

Hence the coordinates (x_2, y_2) of $\mathrm{I_2}$ become

$$x_2 = x_1 + \mathrm{d}x = 510.36 + 7.66 = 518.02$$

$$y_2 = y_1 - \mathrm{d}y = 840.98 - 6.43 = 834.55$$

When adjacent chord slopes are not the same, (Fig. 5.115) an 'averaging' method is used in order to smooth out sharp changes in slope.

(i) The rock slope of any chord is the mean of the slopes as obtained at the chord end points.

$$\text{For chord } \mathrm{H_1I_1}, \text{slope} = \theta_b = \frac{1}{2}(\theta_h + \theta_i)$$

$$\text{For chord } \mathrm{I_1J_1}, \text{slope} = \theta_d = \frac{1}{2}(\theta_i + \theta_j)$$

(ii) The horizontal distance IA (where A is the midpoint of A′A″) is the mean of the distances BC and DE.

Since

$$\mathrm{B_1C} = V\cot\theta_b$$

$$\mathrm{D_1E} = V\cot\theta_d$$

Then

$$\mathrm{I_1A} = \frac{1}{2}(\mathrm{B_1C} + \mathrm{D_1E})$$

or

$$\mathrm{I_1A} = \frac{1}{2}V\left(\cot\frac{\theta_h + \theta_i}{2} + \cot\frac{\theta_i + \theta_j}{2}\right)$$

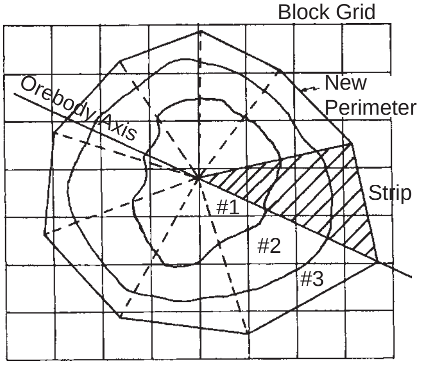


Figure 5.116. Superposition of the bench perimeters on the block model.

The new coordinates of I_2 are given by

$$\text{New coordinates of } I_2 = (x + I_1A \sin\alpha, y - I_1A\cos\alpha)$$

The program then continues around the perimeter and considers each pair of chords in turn and calculates the coordinates of the projected chord end points on the next upper horizon.

When the perimeter on the next upper horizon has thus been generated, the area lying within the new perimeter is scanned and, by reference to the rock type matrix and appropriate vertical interval, the volumes and tonnages of the various rock types between the lower and upper horizons are calculated. The principle adopted is analogous to that of obtaining areas by graph paper rather than by planimeter. Volumes are calculated down to the previous horizon on which a perimeter was generated.

The program works around the perimeter and considers each chord in turn. For each chord, the elemental area between the chord and the axis of the ore body is evaluated in terms of the types of rocks falling within the strip (see Fig. 5.116).

By referring to the current rock matrix as stored in the 'memory' of the computer, the zone between the chord and axis is scanned. Each rock type present is identified and counted in terms of blocks. Then, by reference to the cell dimensions and the level interval, the number of blocks is converted into a volume. In this way all the elements within the pit perimeter are evaluated and progressive totals of each rock type are built up. Table 5.17 shows the type of output which is obtained.

Before continuing to project the perimeter up to the next horizon two checks are made and adjustments carried out if necessary. First the perimeter chords are checked to ensure that the chord length is maintained within a specific range. If the chord length exceeds a certain permitted maximum, an intermediate coordinate point is introduced at the chord's mid point. On the other hand, in certain circumstances, the chord length may become very small and on further projection the end points may actually reverse their order. This will give rise to ambiguous calculations resulting in a badly distorted perimeter, so in order to avoid this situation points are dropped out as chord lengths become less than a certain permitted minimum. The program then scans the perimeter and checks for the development of sharp angles between adjacent perimeter chords, which are compared with a given minimum angle (set with the input data). If the acute chord angle is too sharp, perimeter points are adjusted until the angle becomes larger than the permitted minimum. This part of the program obviates the problem of the development of sharp pit-perimeter curves.

Table 5.17. Typical bench output using the RTZ pit generator (Fairfield & Leigh, 1969).

Test case 3
Base perimeter : 720 ft
Bench No: 11 (Elevation = 440 ft)
Footwall slope angle = 45°
Hangingwall slope angle = 55°

Rock type	1	2	3	4	5	6	7	8
Area (ft^2)	4423.4	29,537.0	59,059.9	45,353.7	14,685.8	1440.0	73,175.6	0.0
Volume (ft^3)	146,627.0	998,383.0	1,793,687.0	1,674,718.0	358,986.0	29,038.0	2,327,491.0	0.0

Total area = 227,675.8 ft^2
Total volume = 7,328,930 ft^3

Table 5.18. Typical total pit output using the RTZ pit generator (Fairfield & Leigh, 1969).

Test case 3: Final summary
Base perimeter : 720 ft
Footwall slope angle: 45°
Hangingwall slope angle: 55°

Rock type	1	2	3	4	5	6	7	8
Volume (ft^3)	3,348,438	5,979,359	17,496,752	13,867,850	9,031,000	1,029,359	51,243,824	7,470,351
Tonnage	304,403	519,944	1,458,062	1,109,428	694,692	76,249	3,660,273	
Cutoff grade (%)					0.6	0.5	0.4	0.3
Volumetric stripping ratio W : O					10.0	2.8	1.5	1.1
Total tonnage of ore (tons)					824,347	2,282,409	3,391,837	4,086,529
Total tonnage of waste (tons)					6,998,704	5,540,642	4,431,214	3,726,522
Average mill feed grade (%)					0.64	0.55	0.50	0.47
Total working costs (units)					27,143,601	32,642,173	34,924,076	36,024,585

Having carried out these checks the program continues to the next horizon where the complete cycle is repeated. Then so on until the final pit perimeter is reached.

Table 5.18 is a typical final summary output from the program. The following are given:

(i) Pit perimeter coordinates.
(ii) Volume and tonnage of ore.
(iii) Volume and tonnage of waste.
(iv) Stripping ratio.
(v) Average grade of mill feed.
(vi) Total working costs.

In cases where working costs and revenues are required additional input to the computer is necessary. Costs are fed in under headings: ‘fixed costs per rock type unit’ (that is, drilling, blasting, loading), ‘depth variable costs’ (that is, transport from various working levels), and ‘fixed cost per unit of rock’ (that is, supervision, services).

The basic pit design method described so far develops a single pit from a single base. Having completed a design for a single pit, the user can continue to add on additional pits either as incremental expansions to the first pit by incorporating a new base which joins the

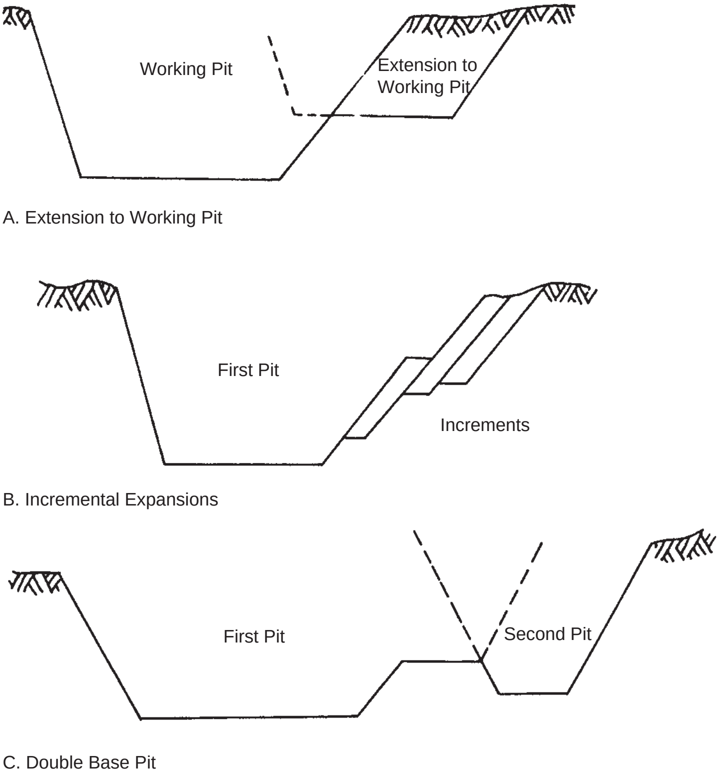


Figure 5.117. More complicated pit expansions (Golder Associates, 1981).

first pit, or by specifying a second base lying outside the first pit thereby creating a double base pit. The same procedure is adopted. A trial base perimeter for the second pit is input to the computer in the same form, and the program repeats the cycle of projection from level to level until the surface is reached.

The perimeter of this second pit will probably encroach on that of the first pit. However, with the aid of its memory the computer will take into consideration the fact that one pit has already been generated. As soon as encroachment occurs the complete volume bounded by the first pit perimeter will be treated as being mined out and hence air. This prevents any duplication that would otherwise occur. Third and further bases can be added by similar means. A second or further base can represent:

1. A working extension to the first pit (Fig. 5.117A).
2. An incremental expansion of the pit to test the sensitivity of the first pit to profitability (Fig. 5.117B).
3. A second pit which may merge with the first at the upper benches (Fig. 5.117C). A typical computer assisted pit design is shown in Figure 5.118.

5.8.2 *Computer assisted pit design based upon sections*

Open-pit design computerization which began in the early 1960's was driven largely by those involved in the mining of large, low grade deposits. The choice of a block model to

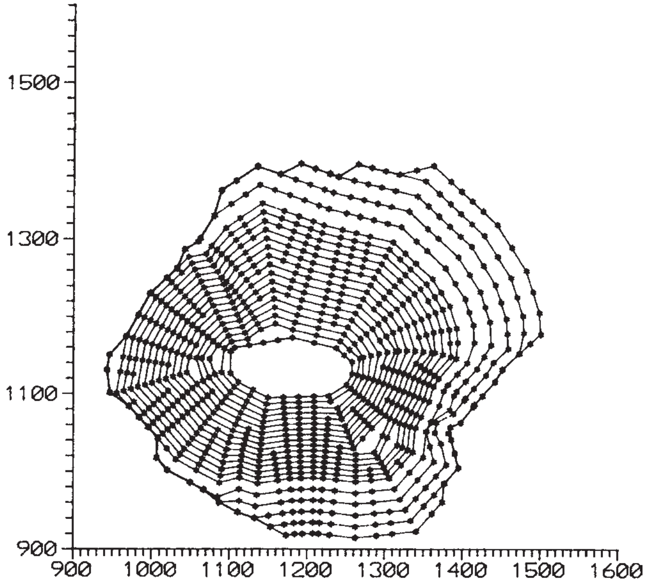


Figure 5.118. A typical pit design (Motta, 1988).

represent porphyry copper deposits, for example, was appropriate. The block size could be made still quite large and yet be small with respect to the size of the deposit and the scale over which the major grade changes take place. The distinction between ore and waste was made on the basis of cut off grade rather than on a sharp boundary.

There are a number of applications where block models are less satisfactory. Steeply dipping, relatively narrow vein/strata form deposits being one example. Here the ore is contained within definite, well defined boundaries. Grade variation within the ore zone is such that an average ore grade may be assigned. In such cases, the use of vertical sections still plays an important role. One computer assisted technique for dealing with such situations has been described by Luke (1972). The example which he presented forms the basis for this section.

Figure 5.119 illustrates the topography and geologic interpretation of a single cross-section through a steeply dipping iron orebody. A thin layer of overburden (alluvium, sand, gravel) overlies the bedrock. There is a sharp transition between the ore zone and the adjacent hanging- and footwall waste zones.

In the manual method of pit design using sections described in an earlier section, it will be recalled that the final pit limits were obtained through an iterative process by which the pit shape was 'floated' around on the section until the actual stripping ratios and average ore grades matched a point on the SR-grade curve.

Such a trial pit for this section is shown in Figure 5.120. The pit bottom has a width of 120 ft, the slope of the right hand slope (RHS) is 60° whereas the left hand slope (LHS) is reduced to 57° to include the presence of a 60 ft wide ramp segment. The straight line approximations for the slopes are used to simplify the process. Once the 'final' best position is determined then the functional mining parameters such as:

– the ramp(s),
– working bench heights,

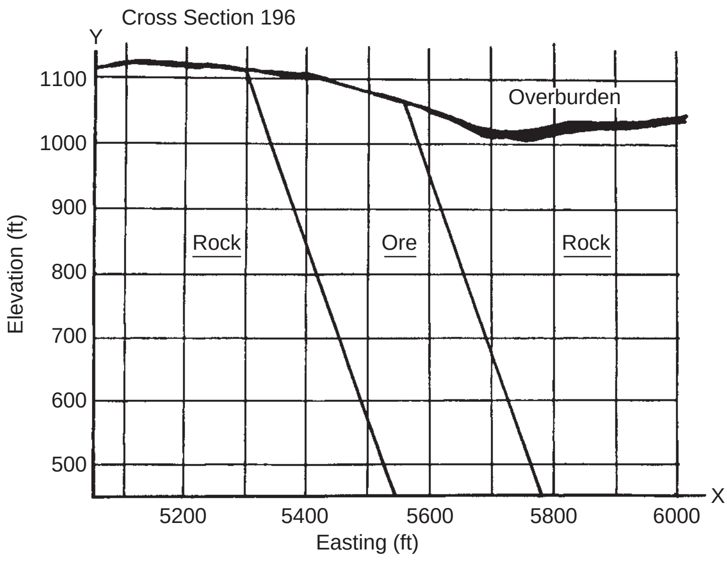


Figure 5.119. Topography and geologic interpretation of a single cross-section for a steeply inclined orebody (Luke, 1972).

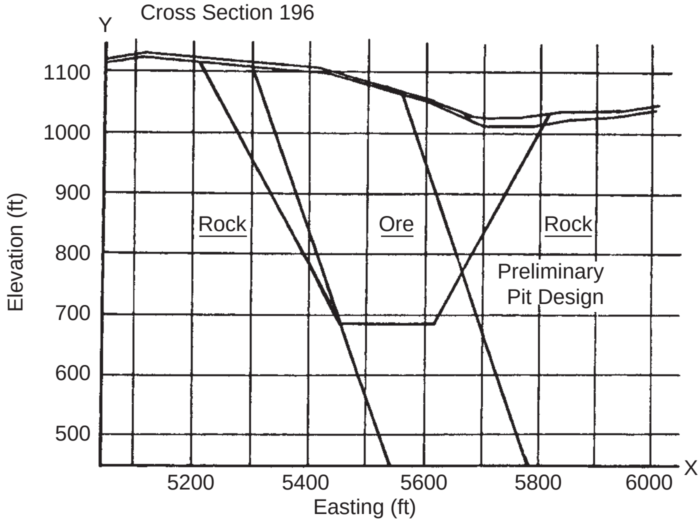


Figure 5.120. Preliminary pit design for example cross-section (Luke, 1972).

– berm widths, and
– bench face angles

are added (Fig. 5.121).

The design is then reexamined. Often significant changes occur between the simplified and actual pit designs. When the 'final' design on this section has been located, the areas of

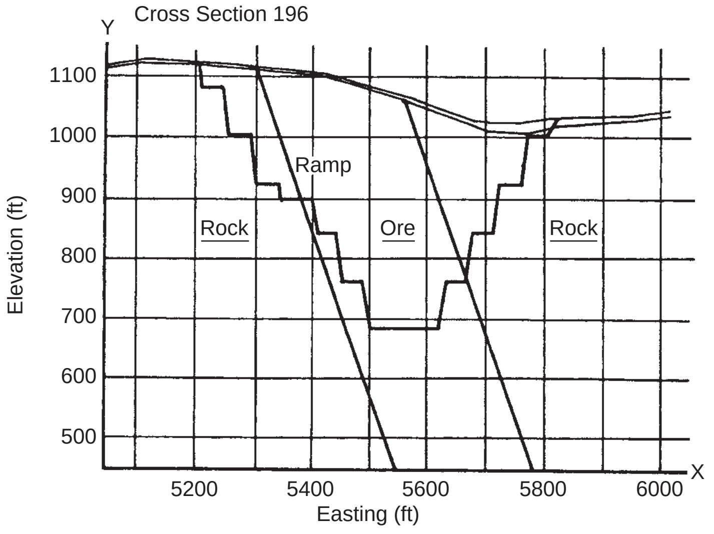


Figure 5.121. Preliminary pit design for example cross-section (Luke, 1972).

overburden, ore and waste are obtained by planimetering. Overall stripping ratios, etc. can then be calculated.

Design changes such as varying

– minimum pit width,
– slope angles,
– position/width of the ramp, and
– limiting stripping ratio

would require the entire process to be repeated.

Computer techniques have been developed to assist in the preparation of the sections and in making the necessary calculations. In this way a large number of potential designs can be evaluated quickly and inexpensively. Since the procedure largely follows the manual process, it is easy for the mine planner to understand what the program is accomplishing and to actively participate in computer-assisted design. The output of the process is a functional mine plan together with a working set of cross-sections. This section will discuss via an iron-ore example, originally presented by Luke (1972) the logic involved in the process.

The process begins with a description of the topography and the geologic data. The topographic relief and the overburden-bedrock contact are defined by a series of straight line segments. The end points of each segment making up the individual contour line (string) are defined either by hand or using a digitizer. In this case the surface relief will be denoted as the surface contour and the top of bedrock as the overburden contour. The values for cross-section 196 are given in Table 5.19.

From the cross-section (Fig. 5.119) it is seen that there are four material types present. Since they adjoin one another, they can be defined by four zones. If, for example, a pod of hangingwall waste exists in the ore then five zones would be needed. These zones must be described. In this simple case the ore-footwall and ore-hangingwall contacts can be described using end point coordinates (Table 5.20).

Table 5.19. Surface and overburden contour strings for cross-section 196 (Luke, 1972).

Surface contour		Overburden contour	
X	*Y*	*X*	*Y*
4870	1100	4870	1097
5000	1106	5000	1103
5114	1125	5111	1122
5251	1114	5222	1113
5298	1114	5305	1110
5416	1102	5406	1100
5539	1073	5419	1100
5613	1050	5523	1073
5673	1024	5590	1052
5723	1021	5710	1010
5743	1020	5770	1008
5838	1035	5842	1023
5940	1034	5940	1028
6000	1043	6057	1043
6100	1050	6121	1043
6121	1049	6200	1030
6200	1035		

Table 5.20. End points for zone lines (Luke, 1972).

	X_1	Y_1	X_2	Y_2
Ore-footwall contact	5298	1112	5550	440
Ore-hangingwall contact	5558	1065	5787	440

Table 5.21. Material descriptors (Luke, 1972).

Zone	Material type	Material description	Divisor
1	50	Rock waste	27.0
2	1	Ore	11.5
3	50	Rock waste	27.0
4	51	Overburden	64.8

The overburden zone lies between the surface and the overburden contour lines. The footwall (hangingwall) waste zone lies to the left (right) of the ore-footwall (hangingwall) zone line and below the overburden contour line. The ore zone lies between the contact lines and beneath the overburden contour line. Each zone corresponds to a particular material type which in turn has certain properties. In this case the area of each material included in the pit will first be determined. This will be changed to a volume by multiplying by the given section thickness (i.e. 100 ft). To convert this volume into the desired units of tons, cubic yards, cubic yards of equivalent rock, etc., certain factors are required. A table of such factors is given in Table 5.21. In this case the factor of 11.5 is used to convert volume (ft^3)

Table 5.22. Pit wall specifications (Luke, 1972).

	Bench height	Berm width	Wall angle	Valid berm elevation	Overburden angle	Toe coordinates	
						X	*Y*
Left	80	35	82	1000	45°	5500	680
Right	80	35	82	1000	45°	5620	680

into long tons of crude ore. The factor of 27 converts volume (ft^3) into cubic yards of rock waste. Since the overburden is much easier to remove than the rock waste, a factor of 2.4 is first introduced to convert volume of overburden (ft^3) into an equivalent volume (ft^3) of rock waste. The factor of 27 is then used to convert ft^3 into yd^3. The overall factor is the product of these two (64.8).

The pit design can now be superimposed upon the basic material-geometry model. As done earlier the pit is defined by a series of connected straight segments. The width and position of the pit bottom is first decided. For Figure 5.120 one can see that the width is 120 ft and the bottom elevation is 680. The end points of this segment form the toe positions of the left and right hand slopes. The walls are defined by:

1. Bench height. The bench elevation differential on the ultimate wall.
2. Bench width.
3. Wall angle. The bedrock wall angle from bench to bench, not the overall slope angle.
4. Valid bench elevation. The elevation of any existing berm if the pit is under current development, or the elevation of the first proposed bench. From this specified bench, the elevation of successive benches is determined from the bench height.
5. Overburden angle. The wall angle that can be maintained in overburden.
6. Toe position. The (X, Y) coordinates for the indicated intersection of the pit wall and pit floor.

The wall and pit bottom specifications for the trial pit are given in Table 5.22. The design specifies the location of the ramp as to

– left or right wall,
– ramp elevation when it crosses the section, and
– ramp width,

For the section shown it has been decided that the ramp should have

$$\begin{aligned} \text{location} &= \text{left wall}, \\ \text{elevation} &= 896\,\text{ft, and} \\ \text{width} &= 60\,\text{ft.} \end{aligned}$$

In manually superimposing the pit onto the section, the designer would locate the pit bottom and then using scale and protractor construct the pit walls. The ramp would be positioned at the proper location. The same procedure is followed with the computer except that end point coordinates of the segments are determined. In addition, the coordinates of the intersections between the string making up the pit and the zone/contour lines are determined as well. Once this has been done, the areas involved in each zone can be obtained. For the ore zone (Zone 2) using a planimeter one might start in the lower left corner (Fig. 5.122) and proceed around the loop in a clockwise motion eventually returning to the starting corner.

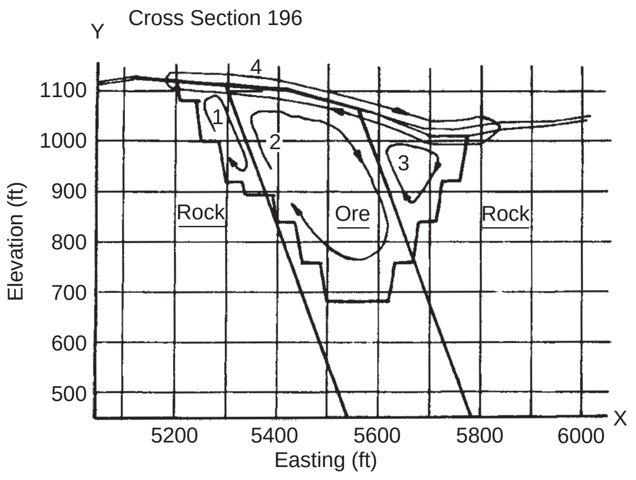


Figure 5.122. Summary of developed data for the example cross-section (Luke, 1972).

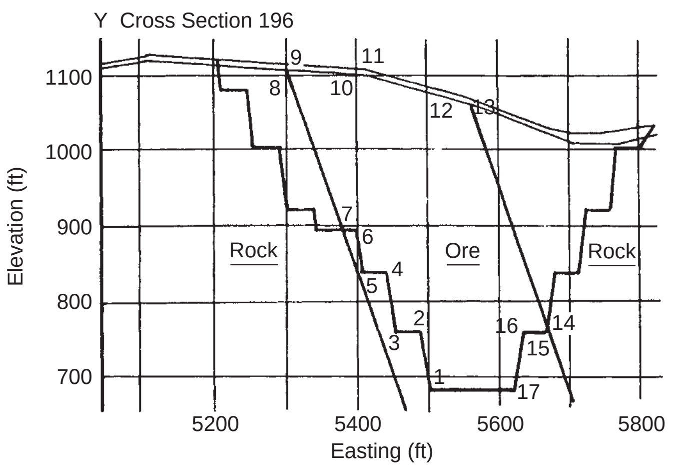


Figure 5.123. Ore area calculation based on the segment end points.

The same basic process is followed knowing the end point coordinates of the segments. Figure 5.123 shows the end point locations and Table 5.23 gives the coordinates.

To demonstrate the process by which areas are found consider the area excavated (A) from the bottom bench in Figure 5.124. The formula used is

$$A = \frac{1}{2}(Y_2 + Y_1)(X_2 - X_1)$$

where Y_1, X_1 are coordinates of initial point of segment and Y_2, X_2 the coordinates of final point of segment.

Table 5.23. Coordinates of the segment end points (Luke, 1972).

Point	X	Y
1	5500	680
2	5488.5	760
3	5453.5	760
4	5442.0	840
5	5407	840
6	5399.1	896
7	5379	896
8	5298	1112
9	5305	1110
10	5406	1100
11	5419	1100
12	5523	1073
13	5558	1065
14	5667.7	766.8
15	5666.6	760
16	5631.5	760
17	5620	680
1	5500	680

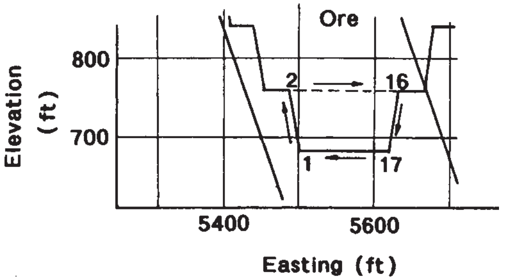


Figure 5.124. Figure used for the example calculation.

In this case, going clockwise around the figure are finds that

Segment 1: 1 to 2

$$A_1 = \frac{1}{2}(760 + 680)(5488.5 - 5500)$$
$$= -8280$$

Segment 2: 2 to 16

$$A_2 = \frac{1}{2}(760 + 760)(5631.5 - 5488.5)$$
$$= 108,680$$

Segment 3: 16 to 17

$$A_3 = \frac{1}{2}(680 + 760)(5620 - 5631.5)$$
$$= -8280$$

Segment 4: 17 to 1

$$A_4 = \frac{1}{2}(680 + 680)(5500 - 5620)$$
$$= -81,600$$

The total area is

$$A = A_1 + A_2 + A_3 + A_4 = -8280 + 108,680 - 8280 - 81,600 = 10,520\,\text{ft}^2$$

Using the formula for the area of the trapezoid one finds that

$$A_{tz} = \frac{1}{2}(b_1 + b_2)h$$

where

$$b_1 = \text{length of lower base} = 120\,\text{ft}$$
$$b_2 = \text{length of upper base} = 143\,\text{ft}$$
$$h = \text{height} = 80\,\text{ft}$$

Substitution yields:

$$A_{tz} = \frac{1}{2}(120 + 143) \times 80 = 10,520\,\text{ft}^2$$

The results from applying this line integration method to zones 1, 2, 3 and 4 in Figure 5.122 are given below

Zone	Material type	Compound area (ft^2)	Section thickness (ft)	Volume (ft^3)
1	50	14,785	100	1,478,500
2	1	87,500	100	8,750,000
3	50	27,931	100	2,793,100
4	51	3264	100	326,400

Using the scaling factors these values are converted into those desired:

Footwall rock waste	$= 54,759\,\text{yd}^3$
Ore	$= 760,870$ long tons (lt)
Hangingwall rock waste	$= 103,448\,\text{yd}^3$
Overburden (equivalent waste rock)	$= 5,037\,\text{yd}^3$

The material totals are:

$$\text{Ore} = 760,870\,\text{lt}$$
$$\text{Waste} = 163,224\,\text{yd}^3$$
$$\text{SR} = 0.215\,\text{yd}^3/\text{lt}$$

Through the use of this pit generator, a variety of pit locations can be tested. In the manual procedure described in Section 5.2, the lengths of the lines in ore and waste were compared. Here the ratio of waste to ore lying between two successive pit positions will be calculated. Two such positions are shown in Figure 5.125.

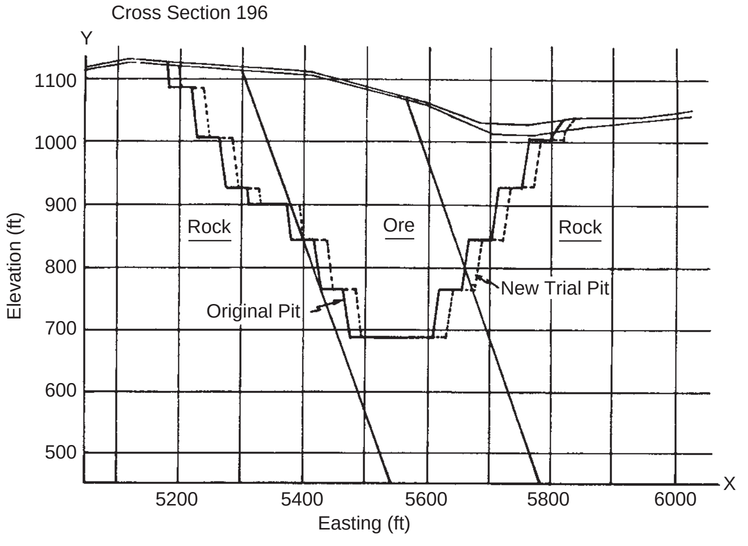


Figure 5.125. Results of the optimization as compared to the preliminary design of Figure 5.121 (Luke, 1972).

The left hand wall was moved 25 ft to the left and the right hand wall 15 ft to the left. For this new position the $(\text{ore})_\text{n}$ and $(\text{waste})_\text{n}$ totals are:

$$(\text{ore})_\text{n} = 787{,}366\,\text{lt}$$

$$(\text{waste})_\text{n} = 172{,}687\,\text{yd}^3$$

The change in the amount of ore between these two pits is

$$\Delta\text{ore} = (\text{ore})_\text{n} - (\text{ore})_\text{o} = 787{,}366 - 760{,}870 = 26{,}496\,\text{lt}$$

Similarly, the change in the amount of waste is

$$\Delta\text{waste} = (\text{waste})_\text{n} - (\text{waste})_\text{o} = 172{,}687 - 163{,}244 = 9443\,\text{yd}^3$$

For this change in geometry the incremental (or differential) stripping ratio (DSR) of the increment is equal to

$$\text{DSR} = \frac{\Delta\text{waste}}{\Delta\text{ore}} = \frac{9443}{26{,}496} = 0.356$$

This value must be compared to the break even (or limiting) stripping ratio (SRL) as applied at the pit periphery. Suppose for example that

$$\text{SRL} = 0.8$$

Since

$$\text{DSR} < \text{SRL}$$

this modification of the pit is desirable. The formula presented by Luke (1972) for use in guiding the changes is

$$\frac{\Delta T - \dfrac{\Delta W}{\text{SLR}}}{|\Delta T|} \geq 0 \tag{5.18}$$

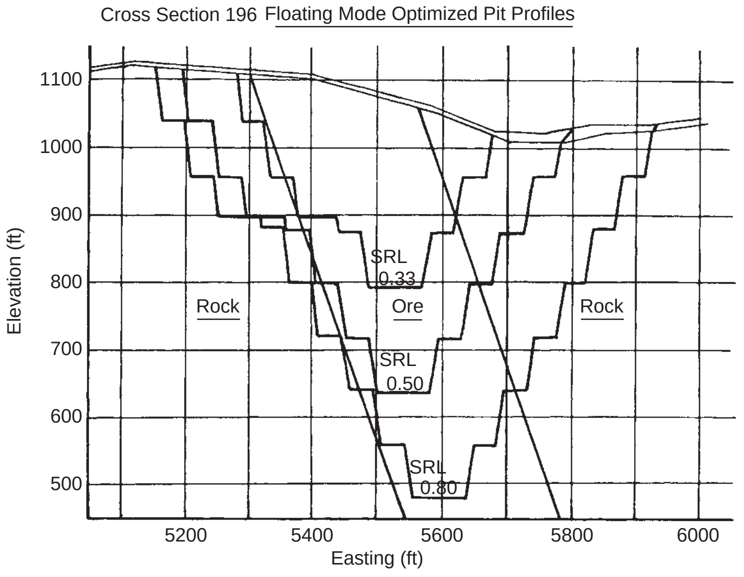


Figure 5.126. Pit profiles for three SRL values (Luke, 1972).

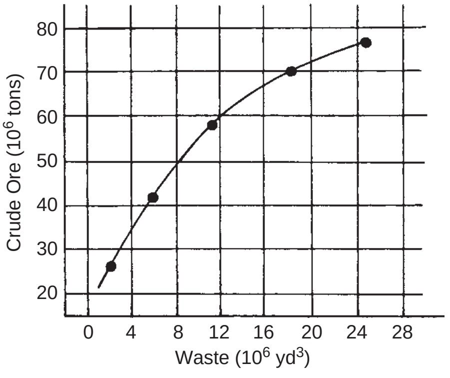


Figure 5.127. Curve of reserve level versus minimum stripping (Luke, 1972).

where

$$\Delta T = T_t - T_b,$$

$$\Delta W = W_t - W_b,$$

$$T = \text{units of ore},$$

W = units of waste,
SRL = incremental stripping ratio,
subscript b refers to the current best position,
subscript t refers to the new trial position.

The optimization can be done under a variety of constraints. One constraint, for example, might be that the pit floor level must remain at a given elevation. When SRL is the only constraint, then the pit can float both vertically and horizontally around the section. Figure 5.126 shows pits for section 196 under different SRL constraints. Figure 5.127 shows the tons of crude ore as a function of stripping for the different SRL scenarios.

Table 5.24. Summary for the optimum pits (Luke, 1972).

Incremental stripping ratio (SRL)	Crude ore (10^6 tons)	Waste (10^6 yd^3)	Stripping ratio
0.33	26.0	2.08	0.080
0.50	42.0	5.87	0.140
0.67	58.0	11.30	0.195
0.80	70.0	18.20	0.260
1.00	76.0	24.70	0.325

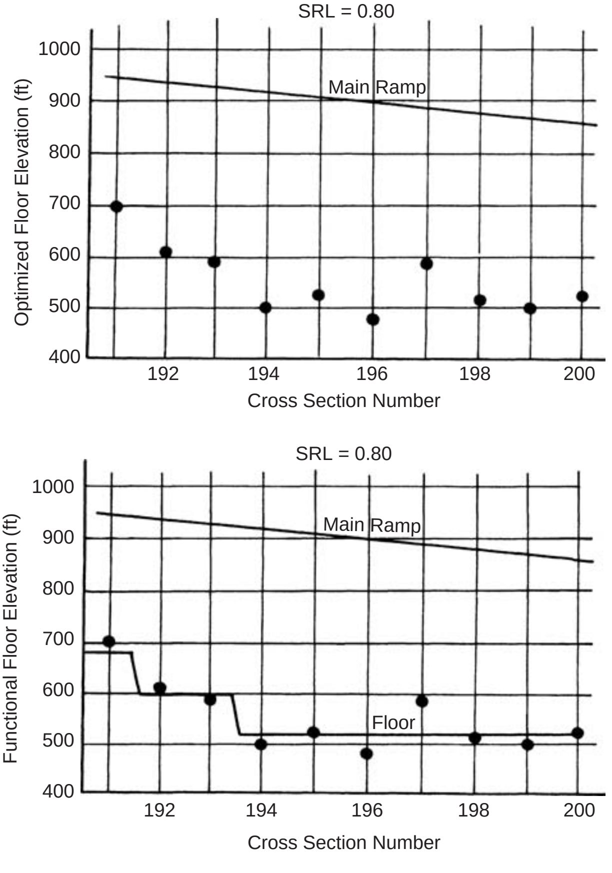


Figure 5.128. Plot of optimum pit floor elevation versus cross-section position in the pit (Luke, 1972).

Table 5.24 presents an overall stripping ratio summary. Such results are very useful for management in examining production decisions.

Figure 5.128 presents the final floor elevations for a series of adjacent sections in which a limit of SRL = 0.8 was imposed. Using this plot, decisions regarding bottom bench location at various locations in the pit can be made. Having decided this elevation the sections can then be rerun to obtain an optimum location. With regard to section 196, the pit bottom should be at 520 ft rather than 480 ft. As can be seen, this use of computer assist, greatly enhances the information base from which the mine planner can make decisions.

REFERENCES AND BIBLIOGRAPHY

Alford, C.G., & J. Whittle. 1986. Application of Lerchs-Grossmann pit optimization to the design of open-pit mines. In: *Large Open-Pit Mining Conference* (J.R. Davidson, editor) 10: 201–208. Australasian Inst. of Min. and Met.

Axelson, A.H. 1964. Apractical approach to computer utilization in mine planning. In: *4th APCOM, Golden, Colorado*: 593–622. Quarterly of the Colorado School of Mines.

Baafi, E.Y., Milawarma, E., & C. Cusack. 1995. Computer aided design and analysis of Sapan Dalam open pit mine. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 1283–1288. Brisbane: AusIMM.

Bakhtavar, E., Namin, F.S., Shahriar, K., & K. Oraee. 2010. Practical final pit depth considering the future underground mining – A case study. Paper 10-033. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.

Barnes, R.J. 1982. Optimizing the ultimate pit. MS Thesis T-2611. Colorado School of Mines.

Barnes, R.J., & T.B. Johnson. 1982. Bounding techniques for the ultimate pit limit problem. In: *Proceedings of the 17th APCOM Symposium* (R.J. Barnes and T.B. Johnson, editors). 263–273. New york: AIME.

Bascetin, A., & A. Nieto. 2007. Determination of optimal cut-off grade policy to optimize NPV using a new approach with optimization factor. JSAIMM. 107(2): 87–94.

Basu, A.J., Fonseca, M., & A.J. Richmond. 2001. Open pit mining risk measured by stochastic simulation. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 87–92. Beijing: Balkema.

Beck, D.A., Arndt, S.M., Reusch, F., & D. Tyler. 2007. Resilient Design of Large Open Pit Slopes, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 3–12.

Berlanga, J.M., Cardona R., & M.A. Ibarra. 1988. Recursive formulae for the floating cone algorithm. In: *Mine Planning and Equipment Selection* (R.K. Singhal, editor): 15–28. Rotterdam: A. A. Balkema.

Boyle, C. 2009. Multivariate conditional simulation of iron ore deposits – Advantages over models made using ordinary kriging. Iron Ore Conference 2009. Perth, WA. July 27–29. pp. 43–50.

Brackebusch, F.W. 1970. Mine planning applications at the Hecla Mining Company. Presented at the SME Fall Meeting, St. Louis, Missouri, Oct. 21–23, 1970: preprint 70-AR-312.

Brackebusch, F.W. 1971a. Hecla open-pit mine planning programs. Personal communication.

Brackebusch, F.W. 1971b. To design Lakeshore open-pit Hecla thinks big with small computer. *E/MJ* 172(7): 72–76.

Braticevic, D.B. 1984. Open-pit optimization method. In: *Proceedings 18th APCOM*: 133–138. London: Inst. Min. Met.

Bye, A. 2007. The application of multi-parametric block models to the mining process. JSAIMM. 107(1): 51–58.

Caccetta, L., & L.M. Giannini. 1986. Optimization techniques for the open-pit limit problem. *Bull. Proc. Australas. Inst. Min. Metall.* (12)291(8): 57–63.

Caccetta, L., & L.M. Giannini. 1988. The generation of minimum search patterns. *The AusIMM Bulletin and Proceedings* (5)293(7): 57–61.

Cai, W.L. 1992. *Sensitivity Analysis of 3-D Model Block Dimensions in the Economic Open-Pit Limit Design*: 475–486. Littleton, CO: SME.

Cai, W.L. 1992. *Sensitivity Analysis of 3-D Model Block Dimensions in the Economic Open Pit Limit Design*. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 475–486. Tucson: SME.

Caldwell, T. 1961. Use of computers in determining pit expansions and pit limits. In: *Transactions of Short Course on Computers and Compute Applications in the Mineral Industry*. Tucson, AZ, April 1961, Vol. II, Section M, Paper M: 1-M10 plus Appendix I.

Camus, J.P. 1992. Open pit optimization considering an underground alternative. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 435–442. Tucson: SME.

Carlson, T.R., Erickson, J.D., O'Brian D.T., & M.T. Pana. 1966. Computer techniques in mine planning. *Mining Engineering* 18(5): 53–56, 80.

Cavender, B. 1992. Determination of the optimum lifetime of a mining project using discounted cash flow and option pricing techniques. *Mining Engineering*. 44(10): 1262–1268.

Celin, E., & P.A. Dowd. 2002. The use of genetic algorithms for multiple cut-off grade optimization. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 769–780. Alaska: SME.

Chanda, E.K. 1999. Maximizing resource utilization in open pit design – A case study. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 359–366. Colorado School of Mines: CSM.

Chen, J., Li, J., Luo, Z., & D. Guo. 2001. Development and application of open pit limits software for the combined mining of surface and underground. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 303–306. Beijing: Balkema.

Chen, T. 1976. 3-D pit design with variable wall slope capabilities. In: *14th APCOM, Penn State, October 1976*: 615–625.

Cherrier, T.E. 1968. A report on the Ceresco Ridge Extension of the Climax molybdenite deposit. MSc Thesis. University of Minnesota.

Crawford, J.T. 1979. Open-pit limit analysis – some observations on its use. In: *16th APCOM* (T.J. O'Neil, editor): 625–634.

Crawford, J.T., & R.K. Davey. 1979. Case study in open-pit limit analysis. In: *Computer Methods for the 80's* (A.Weiss, editor): 310–318.

Dagdelen, K. 1992. Cutoff grade optimization. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 157–168. Tucson: SME.

Dagdelen, K. 1993. An NPV maximization algorithm for open pit mine design. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(257–266). Montreal: CIMM.

Davey, R.K. 1979. Mineral block evaluation criteria. In: *Open-Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 83–96. SME-AIME.

Deraisme, J., & W. Assibey-Bonsu. 2011. Localised uniform conditioning in the multivariate case – An application to a porphyry copper gold deposit. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 131–140.

Diering, J.A.C. 1984. Algorithm for variable slope pit generation. In: *Proceedings 18th APCOM*: 113–121. London: Inst. Min. Met.

Dindiwe, C., & K. Sasaki. 2001. Optimization of pit limit designs by the newly developed BPITC approach. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 249–256. Beijing: Balkema.

Dowd, P.A., Khalokakaie, R., & R.J. Fowell. 2001. A Windows program for incorporating slope design in open pit optimization. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 189–196. Beijing: Balkema.

Drew, D., & E. Baafi. 2002. Ultimate pit limit design using Microsoft Excel Spreadsheet. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 113–121. Alaska: SME.

du Plessis, A., & A.C. Brent. 2006. Development of a risk-based mine closure cost calculation model. JSAIMM. 106(6): 443–450.

Duval Corporation 1976. MINPAC – computerized aid to pit design and extraction scheduling (March).

Erarslan, K., & N. Celebi. 2001. A simulative model for optimum open pit design. CIM Bulletin 94(1055): 59–68.

Erickson, J.D. 1968. Long-range open-pit planning. *Mining Engineering* 20(4): 75–78.

Fairfield, J.D., & R.W. Leigh. 1969. A computer program for the design of open-pits. *Operations Research and Computer Applications in the Mineral Industries* 64(3): 329–340. Quarterly of the Colorado School of Mines.

Ferreira, F.M., & G. De Tomi. 2001. Optimizing ultimate open pit limits with quality constraints. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 261–266. Beijing: Balkema.

Gamache, M., & G. Auger. 2001. Analysis of maximum flow algorithms for ultimate pit contour problems. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 267–272. Beijing: Balkema.

Gauthier, F.J., & R.G. Gray. 1971. Pit design by computer at Gaspe Copper Mines, Limited. *CIM Bulletin* (715)64(11): 95–102.

Grosz, R.W. 1969. The changing economics of surface mining: A case study: In: *A Decade of Digital Computing in the Mineral Industry* (A. Wiess, editor): 401–420. NewYork: AIME.

Hufford, G.A. 1986. Selection of a cutoff grade strategy for an open-pit mine. Society of Mining Engineers of AIME, SME Annual Meeting, New Orleans, March 2–6, 1986: preprint No. 86-49.

Huttagosol, P. 1988. Modified tree graph algorithm for ultimate pit limit analysis. MS Thesis T-3412. Colorado School of Mines.

Huttagosol, P. 1990. Transportation algorithm for ultimate pit limit. PhD Thesis T-3974. Colorado School of Mines.

Huttagosol, P. 1992. Optimizing open-pit limit by transportation algorithm. PhD Thesis. Colorado School of Mines.

Huttagosol, P., & R.E. Cameron. 1992. A computer design of ultimate pit limit by using transportation algorithm. In: *23rd APCOM* (Y.C. Kim, editor): 443–460. Littleton, CO: SME.

Jalali, S.E., Ataee-pour, M., & K. Shahriar. 2006. Pit limit optimization using a stochastic process. CIM Magazine. September/October. pp. 164–174.

Jara, R.M., Couble, A., Emery, X., Magri, E.J., & J.M. Ortiz. 2006. Block size selection and its impact on open-pit design and mine planning. JSAIMM. 106(3): 205–211.

Johnson, T.B. 1973. A comparative study of methods for determining ultimate open-pit mining limits. In: *Proceedings of the 11th APCOM Symposium, Univ. of Arizona, Tucson*.

Johnson, T.B., & D.G. Mickle. 1971. Optimum design of an open-pit – an application in uranium. *Decision Making in the Mineral Industry.* CIM Special Volume 12: 331–326. Discussion: 337–338.

Johnson, T.B., & W.R. Sharp. 1971. A three-dimensional dynamic programming method for optimal ultimate open-pit design. U.S. Bureau of Mines Report of Investigations 7553.

Kelsey, R.D. 1979. Cutoff grade economics. In: *Proceedings 16th APCOM* (T.J. O'Neil, editor): 286–292. SME-AIME.

Khosrowshahi, S., Shaw, W.J., & M.B. McKevitt. 2009. Ore block optimisation for open pit grade control. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 141–145.

Kim, Y.C. 1978. Ultimate pit design methodologies using computer models – the state of the art. *Mining Engineering* 30(10): 1454–1459.

Kim, Y.C., Cai, W., & W.L. Meyer 1988. A comparison of microcomputer based optimum pit limit design algorithms. SME Annual Meeting, Phoenix, AZ, Jan. 25–28, 1988: preprint 88-23.

Koenigsberg, E. 1982. The optimum contours of an open-pit mine: An application of dynamic programming. In: *Proceedings 17th APCOM* (T.B. Johnson and R.J. Barnes, editors): 274–287. SME-AIME.

Korobov, S. 1974. Method for determining optimal open-pit limits. Rapport Technique EP74-R-4. Department of Mineral Engineering, Ecole Polytechnique, Montreal, February 1974.

Korobov, S.D. 1993. Parametric analysis of open pit mines. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(57–66). Montreal: CIMM.

Koskiniemi, B.C. 1979. Hand methods. In: *Open-Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors). SME/AIME: 187–195.

Latorre, E., & T.S. Golosinski. 2011. Definition of economic pit limits taking in consideration time value of money. CIM Journal. 2(3): 162–170.

Laurent, M., Placet J., & W. Sharp. 1977. Optimum design of open-pit mines. Gecamines Rapport No. 04/77. Lubumbashi.

Le Bel, G. 1993. Discussion of the paper. Determination of the optimum lifetime of a mining project using discounted cash flow and option pricing techniques. *Mining Engineering*. 45(11): 1409–1412.

Leigh, R.W., & R.L. Blake. 1971. An iterative approach to the optimal design of open-pits. *Decision-Making in the Mineral Industry*. CIM Special Volume 12: 254–260.

Lemieux, M. 1979. Moving cone optimizing algorithm. In: *Computer Methods for the 80's in the Mineral Industry* (A. Weiss, editor): 329–345. SME-AIME.

Lerchs, H., & I.F. Grossmann 1965. Optimum design of open-pit mines. *CIM Bulletin* 58: 47–54.

Linder, D.E. 1993. Software determines limestone ore body parameters. *Mining Engineering* 45(10): 1275.

Lipkewich, M.P., & L. Borgman. 1969. Two- and three-dimensional pit design optimization techniques. In: *A Decade of Digital Computing in the Mineral Industry* (A. Weiss, editor): 505–524. New york: AIME.

Luke, K.W. 1972. Functional optimization of open-pit mine design utilizing geologic cross-section data. *Trans. of the AIME* 252: 125–131. SME/AIME.

Martínez, L., & R. Wolff. 2006. Modelling the Initial Pit Design – The First Step for Project Valuation, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 269–281.

Mason, P.M., & R.F. Lea. 1978. Practical approach to the use of computers in planning the Nchanga open-pit complex. In: *Proceedings of the Eleventh Commonwealth Mining and Metallurgical Congress* (M.J. Jones, editor): 535–541.

1989 Minerals Yearbook (*Volume I*) *Metals and Minerals*. 1991. U.S. Department of the Interior, Bureau of Mines.Washington, DC: U.S. Government Printing Office.

Motta, R. 1988. Personal communication.

Nilsson, D. 1997. Optimal final pit depth: Once again. *Mining Engineering*. 49(1): 71–72.

O'Brien, N., & F.J. Nowak. 1966. An application of a computer to open-pit mine design. *Pit and Quarry* 58(2): 128–135.

Osborne, W.G. 1976. MINPAC – computerized aid to pit design and extraction scheduling. Spring Meeting Arizona Section. AIME.

Pana, M.T. 1965. The simulation approach to open-pit design. In: *5th APCOM, Tucson Arizona, March 16–19, 1965*: ZZ-1 to ZZ-24.

Pasieka, A.R., & G.V. Sotirow. 1985. Planning and operational cutoff grades based on computerized net present value and net cash flow. *CIM Bulletin* (878)78(6): 47–54.

Phillips, D.A. 1972. Optimum design of an open-pit. In: *10th APCOM* (4): 145–147. Johannesburg: South African Institute of Mining & Metallurgy (SAIMM).

Picard, J-C., & B.T. Smith. 1993. Optimal rate of return in open pit mine design. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(111–118). Montreal: CIMM.

The Planning and Operation of Open-Pit and Strip Mines – Proceedings of an International Conference, Univ. of Pretoria, April 9–13, 1984 (J.P. Deetlefs, editor). Symposium Series S7. Johannesburg: (SAIMM).

Pretorius, J.L. 2007. The Multi-Element Functionality of the Whittle Optimiser – A Different Approach in the Evaluation of Iron Ore Deposits, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 163–173.

Pronk Van Hoogeveen, L.A.J., Cutland. J.R. & M. Weir. 1972. An open-pit design system for stratiform ore-bodies. In: *10th Application of Computers in the Minerals Industries Symposium*. Johannesburg: SAIMM.

Quang, N.C. & D.N. Trung. 2001. Optimization of grouped open pits. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 257–260. Beijing: Balkema.

Reibell, H.V. 1969. Deep open-pit optimization. In: *A Decade of Digital Computing in the Mineral Industry* (A. Weiss, editor): 359–372. New York: AIME.

Richmond, A., Gaze, R., & J. Horton. 2009. De-smoothing block models for underground mine design. Seventh Int. Mining Geology Conference. Perth, WA. August 17–19. pp. 265–267.

Robinson, A. 2007. Performance Driver Trees for Optimising Open Pit Operations, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 113–118.

Robinson, R.H. 1975. Programming the Lerchs-Grossmann algorithm for open-pit design. In: *Proceedings of the 13th APCOM Symposium, Clausthal, Germany, 1975*: B-IV 1–17.

Robinson, R.H., & N.B. Prenn. 1973. An open-pit design model. In: *Proceedings of the 10th APCOM Symposium*: 155–163. Johannesburg: SAIMM.

Rocca, F., Sebbag, M., & T. Taimre. 2007. Buzwagi Open Pit Study, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 119–128.

Sainsbury, G.M. 1970. Computer-based design of open cut mines. *Proc. Aust. Inst. Min. Met.* (6)(234): 49–57.

Santander, E.A., Peña, M.P., & A.R. Bonani. 2007. Optimisation of Slope Angles at Rosario Mine, Collahuasi, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 129–137.

Saydam, S., & E. Yalcin. 2002. Reserve and ultimate pit limit design analysis of Caldagi Nickel Deposit, Turkey. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 121–132. Alaska: SME.

Sens, J., & E. Topal. 2009. A new algorithm for stope boundary optimisation. AusIMM New Leaders Conference. Brisbane, QLD. April 29–30. pp 25–28.

Shahriar, K., Oraee, K., Bakhtavar, E., & P. Flett. 2010. Economic-mathematical analysis of transition from open pit to underground mining. Paper 10-032. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.

Soderberg, A., & D.O. Rausch. 1968. Chapter 4.1. Pit planning and layout. In: *Surface Mining* (E.P. Pfleider, editor): 141–165. NewYork: SME-AIME.

Strangler, R.L., Costa, J.F., & J.C. Koppe. 2001. Risk in stripping ratio estimation. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 81–86. Beijing: Balkema.

Tanays, E., Cojean, R., & D. Hantz. 1992. DEGRES: A software to design open-pit geometry and to draw open-pit plans. *Int. J. of Surface Mining and Reclamation* (2)6: 91–98.

Underwood, R., & B. Tolwinski. 1996. Lerchs Grossman algorithm from a dual simplex viewpoint. 26th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 229–236. Penn. State University: SME.

Wang, Q., & H. Sevim. 1992. Enhanced production planning in open pit mining through intelligent dynamic search. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 461–474. Tucson:SME.

West, F.J. 1966. Open-pit planning at the Adams Mine. *CIM Bulletin* (657)59(3): 392–331.

Wharton, C.L. 1996. What they don't teach you in mining school – Tips and tricks with pit optimizers. In Surface Mining 1996 (H.W. Glen, editor): 17–22. SAIMM.

Whittle, J. 1988. Beyond optimization in open-pit design. 1st Canadian Conference on Computer Applications in the Mineral Industry, Laval University, Quebec City, March 7–9, 1988.

Whittle, J. 1989. The facts and fallacies of open-pit optimization. Whittle Programming Pty. Ltd. (1).

Wick, D.E., & E.M. Sunmoo. 1974. Computer use in determining ore reserves and stripping ratios at Eagle Mountain. In: *Proceedings of the 12th APCOM* (T.B. Johnson and D.W. Gentry, editors), Volume II: H59-H71. Colorado School of Mines.

Williams, C.E. 1974. Computerized year-by-year open-pit mine scheduling. *Trans* 256. SME-AIME.

Williamson, D.R., & E.R. Mueller. 1976. Ore estimation at Cyprus Pima Mine. Paper presented at the AIME Annual Meeting, Las Vegas, Feb. 22–26, 1976: preprint 76-AO-13.

Wright, E.A. MOVING CONE II – A simple algorithm for optimum pit limits design. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 367–374. Colorado School of Mines: CSM.

Xu, C., & P.A. Dowd. 2002. Optimal orebody reconstruction and visualization. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 743–756. Alaska: SME.

Yamatomi, J., Mogi, G., Akaike, A., & U. Yamaguchi 1995. Selective extraction dynamic cone algorithm for three-dimensional open pit designs. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 267–274. Brisbane: AusIMM.

Yegulalp, T.M., & J.A. Arias. 1992. A fast algorithm to solve the ultimate pit limit problem. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 391–398. Tucson: SME.

Yegulalp, T.M., & J.A. Arias. 1992. A fast algorithm to solve the ultimate pit limit problem. In: *Proceedings 23rd APCOM* (Y.C. Kim, editor): 391–397. Littleton, CO: SME.

Zhao, Y., & Y.C. Kim. 1990. A new graph theory algorithm for optimal ultimate pit design. SME Annual Meeting, Salt Lake City, UT, Feb. 26–Mar. 1, 1990: preprint No. 90-9.

Zhao, Y., & Y.C. Kim. 1992. A new optimum pit limit design algorithm. In: *23rd APCOM* (Y.C. Kim, editor): 423–434. Baltimore: Port City Press/SME.

Zhao, Y., & Y.C. Kim. 1992. A new optimum pit limit design algorithm. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 423–434. Tucson: SME.

REVIEW QUESTIONS AND EXERCISES

1. What is the difference between a mineral inventory and an ore reserve?
2. Repeat the example in section 5.2.1 assuming that the wall on the left side of the pit is at 53° rather than 45°. Determine the:
   a. Final pit position
   b. Breakeven stripping ratio
   c. Overall stripping ratio
   d. Value of the section

   Use a slice of zero thickness. Compare the results to those for a slice of thickness 1.4u.
3. Repeat the example in section 5.2.1 assuming that the density of the ore is 1.2 times that of the waste.
4. Summarize the pit limit determination procedure in words.
5. What is meant by the term 'cutoff grade?'
6. What is the practical use of the cutoff grade? Can there be more than one cutoff grade?
7. What is meant by the breakeven cutoff grade?
8. Repeat the net value calculation example in section 5.2.2 assuming a mill recovery of 85% and a mill concentrate grade of 28%. All other factors remain constant. Complete all of the steps including development of the final curve (Figure 5.9).
9. What is meant by the concentration ratio?
10. Summarize the steps required for making a net value calculation.
11. Redo the example of section 5.2.3 assuming that the right hand slope is 45°.
12. The stripping ratio – ore grade curve in Figure 5.9 has been used to determine the final limits for the section shown in Figure 5.10. Repeat the example using the curve obtained in problem 8.
13. Summarize the process of locating the pit limits when the pit bottom is in waste.
14. How is the procedure modified if the ore and waste densities are different? Rework the example in section 5.2.3 if the ore density is 3.0 g/cm^3 and the waste density is 2.5 g/cm^3.
15. How is the procedure modified if there is a difference in the waste and ore mining costs? Rework the example in section 5.2.3 assuming the ore mining cost is $1.10/ton and the cost of mining waste is $0.85/ton.
16. Apply the curve developed in Problem 12 to the determination of the final limits in Figure 5.12.
17. Apply the curve developed in Problem 12 to Figure 5.13.
18. Show the development of equation (5.11) which relates the true stripping ratio for a radial section to the measured.
19. Summarize the steps outlined by Koskiniemi for developing a composite mine plan map from the sections.
20. Assuming that the width of the safety bench in Figure 5.27 is 35 ft, what is the road grade?
21. Using Figure 5.28, check the values given in Table 5.9 for the reserves in Table 5.9. What must have been the block size, block height and material density used?
22. Once the final pit outline has been determined, the material within the pit limits is re-evaluated regarding destination. What is the reason for this? How is the decision made?

23. With the advent of the computer, many companies nowuse block to model their deposits. To develop the mineable reserves, economic values must be applied to each block. Should the G&A costs be assigned to the ore only? To the ore and waste? Discuss. How can this decision be included in CSMine?
24. In a block model can you include depreciation costs? Minimum profit? To which blocks should such values be assigned?
25. As pits get deeper, the haulage costs increase. Assuming that the truck operating cost is \$150/hour. The bench height is 15 m and the road grade is 10%. If the average truck speed is 10 mph uphill loaded and 20 mph down hill unloaded, what should be the assigned incremental haulage cost/level.
26. Discuss the floating cone process described in section 5.4.
27. In the running of a floating cone model to a particular level, the cone returns to the surface and repeats the process. Why are these scavenging runs performed?
28. Describe three problems with regard to the application of the floating cone technique.
29. List the positive aspects of the floating cone technique.
30. What is meant by the 'optimal' pit?
31. Summarize the steps in the Lerchs-Grossmann 2D algorithm.
32. Redo the Lerchs-Grossmann 2D assuming ore value $= +16$ and the waste value $= -4$.
33. Redo problem 32 using the floating cone technique.
34. Summarize the 2½-D technique described in section 5.6.
35. Apply the 2½-D process to sections 2 through 5. Compare your results to those given in Figure 5.81.
36. On section 4 the block at position $i = 5$, $k = 4$ has a value of 9 but is not being mined. Apply the 1–9 constraint model of Figure 5.84 to show why this result is correct.
37. Apply the 2½-D procedure to the block sections in Figure 5.82. Check your results versus those given in Figure 5.83.
38. Define the following terms used with respect to the Lerchs-Grossmann 3D algorithm:
    a. Directed arc
    b. Edge
    c. Weight
    d. Node
    e. Graph
    f. Directed graph
    g. Sub graph
    h. Closure
    i. Maximum closure
    j. Circuit
    k. Chain
    l. Cycle
    m. Path
    n. Tree
    o. Root
    p. Branch
    q. Twig
39. Following the steps, apply the 'arbitrary tree approach (Approach 1)' to the example shown in Figure 5.89.

40. Following the steps, apply the 'all root connection approach (Approach 2)' to the example shown in Figure 5.89.
41. Apply the tree 'cutting' process to the problem shown in Figure 5.105.
42. Apply your skills to the more complicated example shown in Figure 5.110.
    a. Start with doing the first two layers.
    b. For more practice, choose the first three layers.
    c. Apply your skills to the full four layer problem whose solution is given in Figure 5.111.
43. Summarize the steps used in the RTZ open pit generator.
44. Redo one of the computer-assisted design examples considered by Luke. Check your answers with those given.

CHAPTER 6

Production planning

6.1 INTRODUCTION

In this chapter some of the production planning activities involved in an open pit mine will be discussed. Specifically, attention will be devoted to mine life – production rate determinations, push back design and sequencing, as well as providing some general guidance regarding both long and short range planning activities.

The basic objectives or goals of extraction planning have been well stated by Mathieson (1982):

– To mine the orebody in such a way that for each year the cost to produce a kilogram of metal is a minimum, i.e., a philosophy of mining the 'next best' ore in sequence.

– To maintain operation viability within the plan through the incorporation of adequate equipment operating room, haulage access to each active bench, etc.

– To incorporate sufficient exposed ore 'insurance' so as to counter the possibility of mis-estimation of ore tonnages and grades in the reserve model. This is particularly true in the early years which are so critical to economic success.

– To defer waste stripping requirements, as much as possible, and yet provide a relatively smooth equipment and manpower build-up.

– To develop a logical and easily achievable start-up schedule with due recognition to manpower training, pioneering activities, equipment deployment, infrastructure and logistical support, thus minimizing the risk of delaying the initiation of positive cash flow from the venture.

– To maximize design pit slope angles in response to adequate geotechnical investigations, and yet through careful planning minimize the adverse impacts of any slope instability, should it occur.

– To properly examine the economic merits of alternative ore production rate and cutoff grade scenarios.

– To thoroughly subject the proposed mining strategy, equipment selection, and mine development plan to 'what if' contingency planning, before a commitment to proceed is made.

Planning is obviously an ongoing activity throughout the life of the mine. Plans are made which apply to different time spans.

There are two kinds of production planning which correspond to different time spans (Couzens, 1979):

– Operational or short-range production planning is necessary for the function of an operating mine.

– Long-range production planning is usually done for feasibility or budget studies. It supplements pit design and reserve estimation work and is an important element in the decision making process.

Couzens (1979) has provided some very useful advice which should be firmly kept in mind by those involved in the longer term planning activities:

How would I plan this if I had to be the mine superintendent and actually make it work?

In guiding the planner, Couzens (1979) has proposed the following five planning 'commandments' or rules:

1. We must keep our objectives clearly defined while realizing that we are dealing with estimates of grade, projections of geology, and guesses about economics. We must be open to change.

2. We must communicate. If planning is not clear to those who must make decisions and to those who must execute plans, then the planning will be either misunderstood or ignored.

3. We must remember that we are dealing with volumes of earth that must be moved in sequence. Geometry is as important to a planner as is arithmetic.

4. We must remember that we are dealing with time. Volumes must be moved in time to realize our production goals. The productive use of time will determine efficiency and cost effectiveness.

5. We must seek acceptance of our plans such that they become the company's goals and not just the planner's ideas.

This chapter will focus on the longer term planning aspects both during feasibility studies and later production.

6.2 SOME BASIC MINE LIFE – PLANT SIZE CONCEPTS

To introduce this very important topic, an example problem will be considered. Assume that a copper orebody has been thoroughly drilled out and a grade block model constructed. Table 6.1 presents an initial estimate for the costs and recoveries.

Since they will be refined later, this step will be called Assumption 1.

The best estimate price (in this case \$1/lb) is also selected (Assumption 2). From these values an economic block model is constructed. The break-even grade for final pit limit

Table 6.1. Costs used to generate the economic block model.

Mining cost (ore)	= \$1.00/ton
Mining cost (waste)	= \$1.00/ton
Milling cost	= \$2.80/ton
G&A cost (mining)	= \$0.17/ton
G&A cost (milling)	= \$0.40/ton ore
Smelting, refining and sales	= \$0.30/lb Cu
Overall metal recovery	= \$78%

Table 6.2. Mineral inventory as a function of grade class interval.

Grade class interval (% Cu)	Tons (10^3)	Grade class interval (% Cu)	Tons (10^3)
>3.2 (Ave = 5.0)	25	1.5–1.6	205
3.1–3.2	7	1.4–1.5	130
3.0–3.1	15	1.3–1.4	270
2.9–3.0	5	1.2–1.3	320
2.8–2.9	5	1.1–1.2	570
2.7–2.8	10	1.0–1.1	460
2.6–2.7	33	0.9–1.0	550
2.5–2.6	40	0.8–0.9	420
2.4–2.5	15	0.7–0.8	950
2.3–2.4	25	0.6–0.7	980
2.2–2.3	30	0.5–0.6	830
2.1–2.2	30	0.4–0.5	1200
2.0–2.1	50	0.3–0.4	1050
1.9–2.0	75	0.2–0.3	1300
1.8–1.9	60	0.1–0.2	2700
1.7–1.8	150	<0.1	18,020
1.6–1.7	170		

determination is

$$g(\%\ \text{Cu}) = \frac{\$1.00 + \$2.80 + \$0.40 + \$0.17}{0.78(1.00 - 0.30)\frac{2000}{100}}$$

$$= \frac{\$4.37}{10.92} \cong 0.40\%(\text{Cu})$$

It is noted that the costs chosen at this stage might be considered as 'typical'. No attempt has been made to include capital related costs nor a 'profit'. Using the floating cone, Lerchs-Grossmann or another technique, a final pit outline is generated. The intersections are transferred onto the corresponding benches of the grade block model. A mineral inventory using appropriate grade class intervals is created bench by bench. These are later combined to form a mineral inventory for the material within the pit. Table 6.2 is the result for the example orebody (Hewlett, 1961, 1962, 1963). The number of tons in each grade class interval (0.1%) are plotted versus grade. This is shown in Figure 6.1. From this curve a plot of total (cumulative) tonnage above a given cutoff grade can be constructed. The result is given in Figure 6.2. As can be seen, the curve is of the shape typical for a low grade copper orebody. Figure 6.3 shows the straight line expected when a lognormal plot is made of the data.

One can also determine the average grade of the material lying above a given cutoff grade. Figure 6.4 presents the results. Table 6.3 contains the data used in constructing Figures 6.2 and 6.3. These two curves are very useful when considering various plant size – mine life options.

At this point two destination options will be considered for the material contained within the pit:

– Destination A: Mill.
– Destination B: Waste dump.

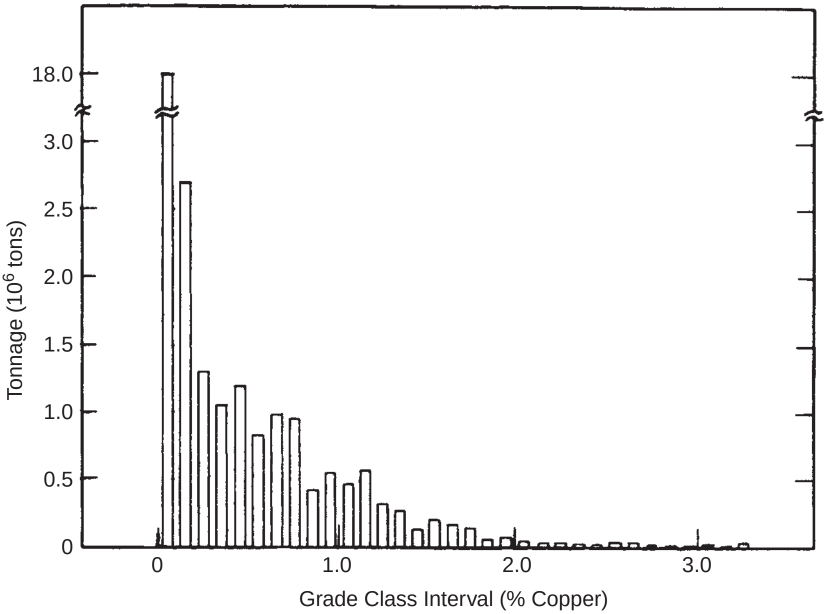


Figure 6.1. Tonnage versus grade class interval for the Silver Bell oxide pit (Hewlett, 1961, 1962).

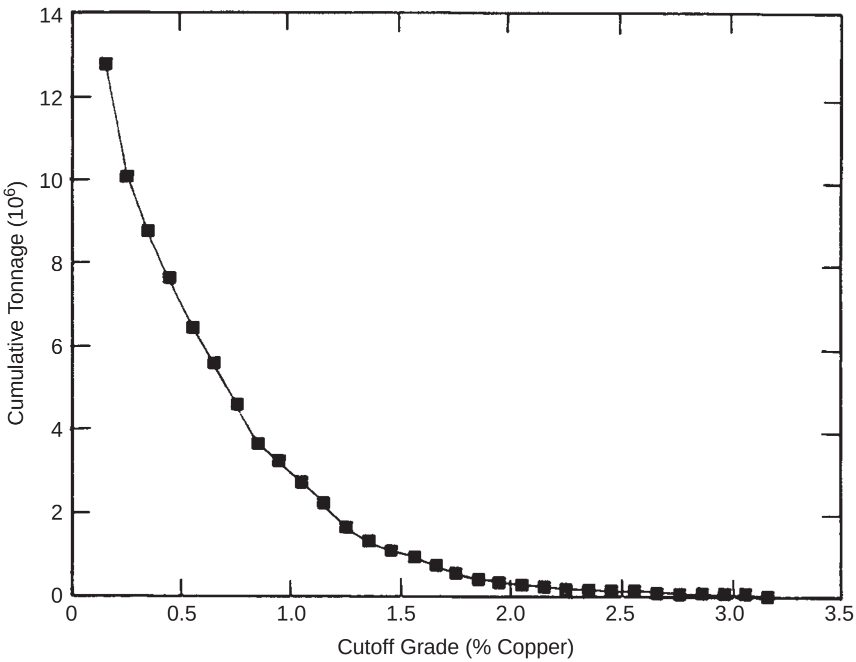


Figure 6.2. Cumulative tonnage versus cutoff grade for the Silver Bell oxide pit.

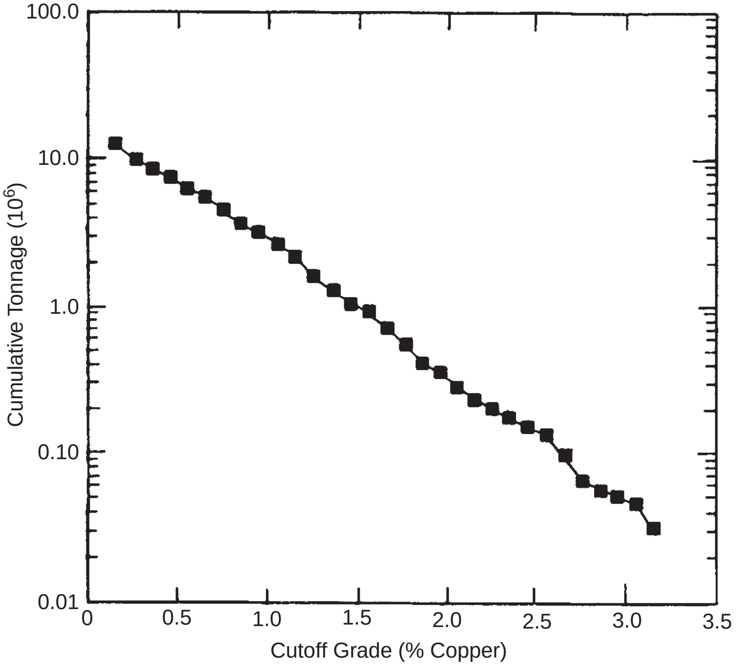


Figure 6.3. Logarithm of cumulative tonnage versus cutoff grade.

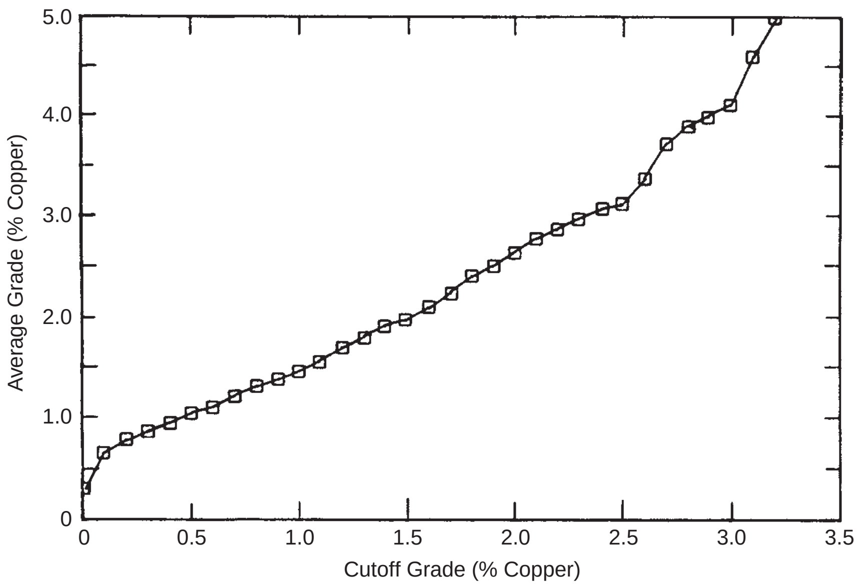


Figure 6.4. Average grade versus cutoff grade.

Table 6.3. Cumulative tons and average grade as a function of cutoff grade.

Grade (% Cu)		
Cutoff	Average	Tons above cutoff (10^3)
0.0	0.30	30,800
0.1	0.65	12,780
0.2	0.78	10,080
0.3	0.86	8780
0.4	0.94	7630
0.5	1.04	6430
0.6	1.11	5600
0.7	1.21	4620
0.8	1.32	3670
0.9	1.38	3250
1.0	1.47	2700
1.1	1.56	2240
1.2	1.70	1670
1.3	1.81	1350
1.4	1.92	1080
1.5	1.99	950
1.6	2.11	745
1.7	2.24	575
1.8	2.41	425
1.9	2.51	365
2.0	2.65	290
2.1	2.78	240
2.2	2.87	210
2.3	2.97	180
2.4	3.07	155
2.5	3.13	140
2.6	3.37	100
2.7	3.72	67
2.8	3.89	57
2.9	3.99	52
3.0	4.10	47
3.1	4.60	32
3.2	5.00	25

There are other possible destinations which can be considered later. This destination stipulation becomes Assumption 3. Since all of the material will eventually have to be removed from the pit there is no question concerning mining or not mining. For the sake of this example it will be assumed that the mineral distribution is uniform throughout the pit. This is generally not true and there will be high and low grade areas of various extent.

A cutoff grade must be selected differentiating ore (that going to the mill) and waste (that going to the dump). Assumption 4 will be that the mill cutoff grade is 0.40% (the same value used in the pit limit determination). From Figure 6.5 one finds that there are 7.8×10^6 tons of ore with an average grade (Fig. 6.6) of 0.92%.

The next question is with regard to the size of plant to be constructed. An equivalent question is 'What is the expected life of the property?' Although there are several ways of approaching this, the one chosen here is market based. It will be assumed (Assumption 5)

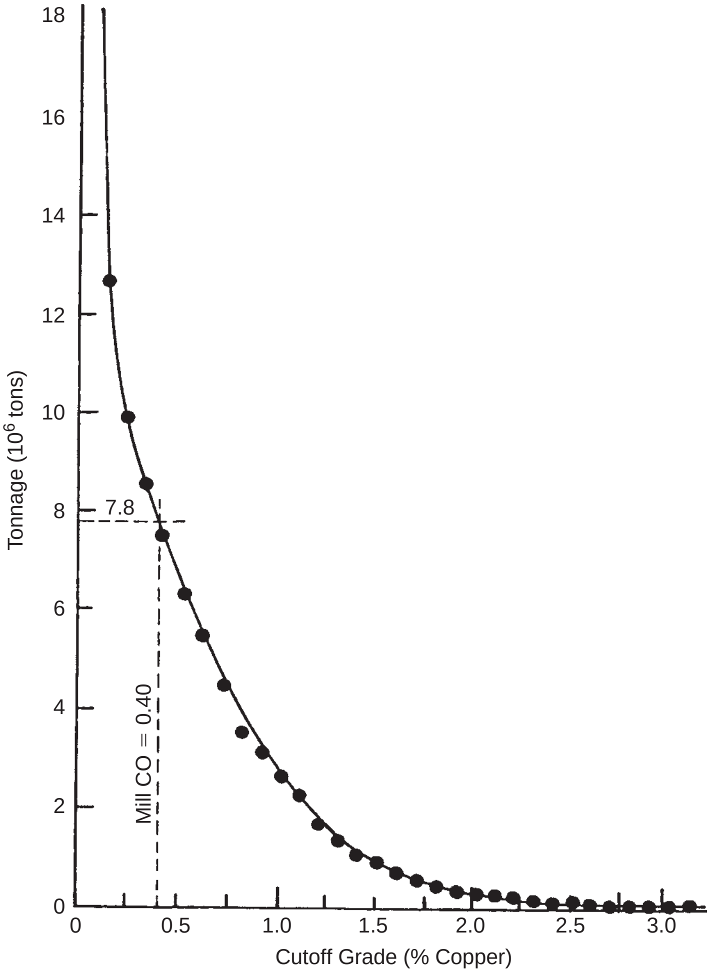


Figure 6.5. Mill tonnage for a 0.4 cutoff grade.

that a market survey has suggested that 5000 tons of copper metal can be sold every year. The yearly and daily production rates as well as the mine life can now be computed.

Assuming that:

Mill recovery = 80%

Combined smelter/refinery recovery = 97%

Operating days = 250 days/yr

we obtain the following milling rates (R_{mill})

$$R_{mill} = \frac{5000\,\text{tpy} \times 2000\,\text{lbs/ton}}{\frac{0.92}{100} \times 0.80 \times 0.97 \times 2000\,\text{lbs/ton}}$$

$$= 700{,}360\,\text{tpy}$$

$$R_{mill} = 2801\,\text{tpd}$$

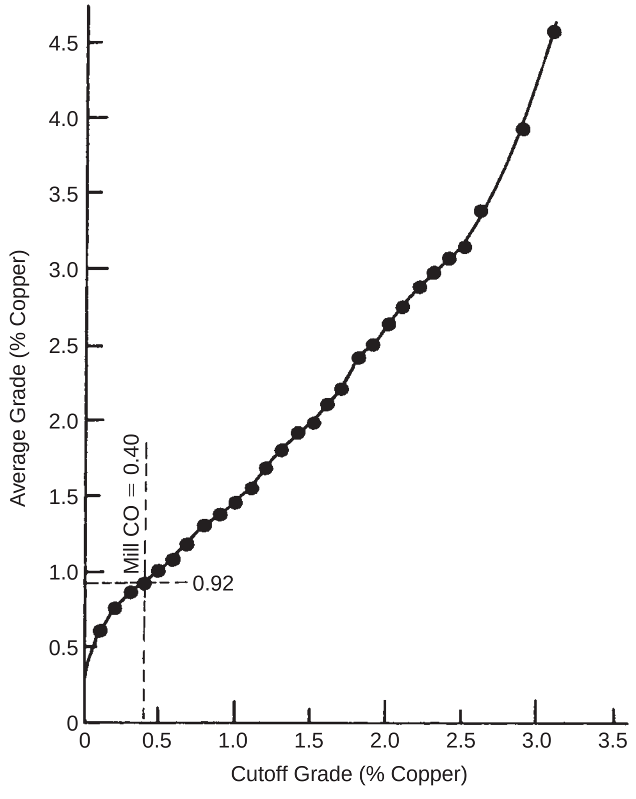


Figure 6.6. Average mill feed grade for a 0.4 cutoff grade.

Knowing the mill production rate and the ore reserves the mill/mine life can be calculated:

$$\text{Mine life (yrs)} = \frac{\text{Ore reserves (tons)}}{\text{Ore production rate (tpy)}} = \frac{7{,}800{,}000}{700{,}360} = 11.1 \text{ years}$$

The required mine production rate (R_{mine}) is:

$$R_{\text{mine}} = \frac{\text{Mineral reserve (tons)}}{\text{Mine life (yrs)}} = \frac{30{,}800{,}000}{11.1} = 2{,}775{,}000 \text{ tpy}$$

$$R_{\text{mine}} = 11{,}100 \text{ tpd}$$

The total amount of copper recovered is

$$\text{Copper recovered} = 55{,}500 \text{ tons}$$

and the overall stripping ratio is

$$\text{Overall SR} = \frac{23{,}000{,}000}{7{,}800{,}000} = 2.95$$

Knowing the mining and milling rates and the mine life

$$\begin{aligned} \text{Milling} &= 2801\,\text{tpd} \\ \text{Mining} &= 11{,}000\,\text{tpd} \\ \text{Mine life} &= 11.1\ \text{years} \end{aligned}$$

one can now go back to Assumption 1 and improve the operating cost estimates. Using these values one can recalculate the economic block values, the final pit limits, the grades-tonnages, etc. Eventually a solution will be found which changes little from run to run.

Knowing the plant size, the required capital investment can be determined. The cash flows are calculated as is the net present value. Obviously other economic indicators such as total profit, internal rate of return, etc., could be calculated as well. One might then return to Assumption 2 and examine the sensitivity with price.

It will be recalled that Assumption 4 dealt with the mill cutoff grade which was chosen as 0.4. If a mill cutoff grade of 0.2 is assumed instead the process can be repeated maintaining the 5000 ton copper output. As Figures 6.7 and 6.8 show, the ore tonnage and average grade become

$$\text{Ore tonnage} = 10.8 \times 10^6\ \text{tons}$$

$$\text{Average ore grade} = 0.77\%\,\text{Cu}$$

With the same assumptions as before the milling rate, mill/mine life and mining rate become:

Mill rate:

$$\begin{aligned} R_{\text{mill}} &= \frac{5000 \times 2000}{\frac{0.77}{100} \times 0.80 \times 0.97 \times 2000} \\ &= 836{,}800\,\text{tpy} \end{aligned}$$

$$R_{\text{mill}} = 3350\,\text{tpd}$$

Mill/mine life:

$$\text{Mill life} = \frac{10{,}800{,}000}{836{,}800} = 12.9\,\text{yrs}$$

Mining rate:

$$R_{\text{mine}} = \frac{30{,}800{,}000}{12.9} = 2{,}387{,}600\,\text{tpy}$$

$$R_{\text{mine}} = 9550\,\text{tpd}$$

The amount of copper recovered increases to

$$\text{Copper recovered} = 64{,}500\,\text{tons}$$

and the overall stripping ratio would drop to

$$\text{Overall SR} = \frac{20{,}000{,}000}{10{,}800{,}000} = 1.85:1$$

A summary of the results for the two mill cutoff grades is given in Table 6.4.

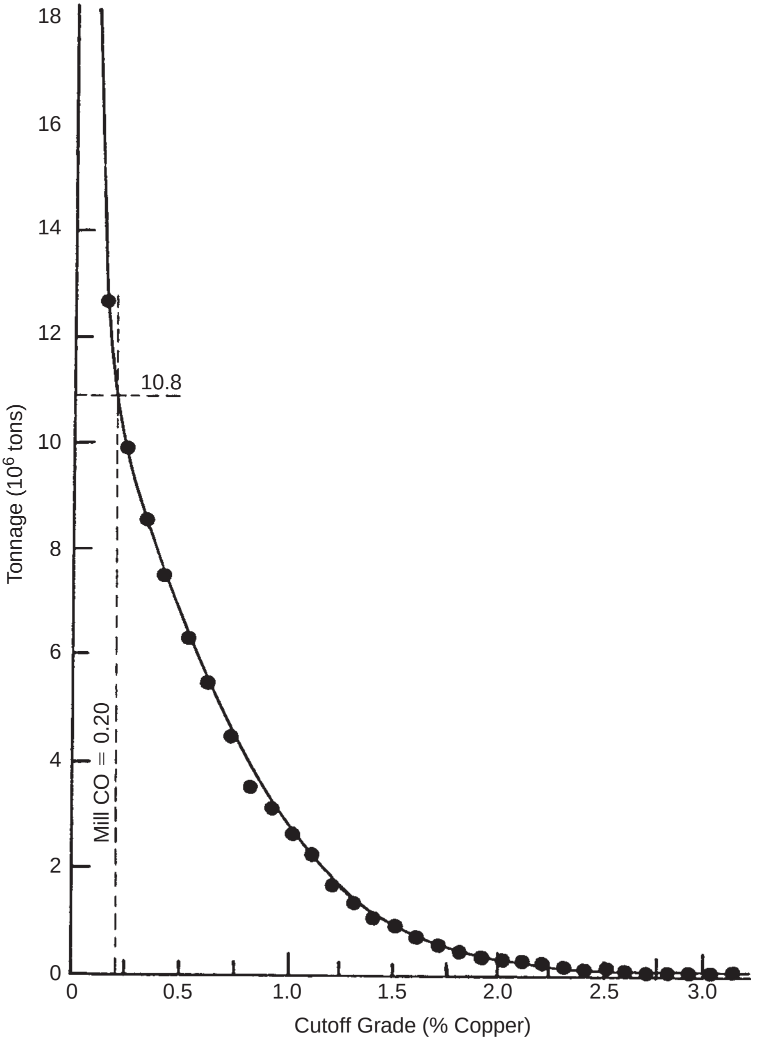


Figure 6.7. Mill tonnage for a 0.3 cutoff grade.

These would be expected to yield different economic results. The incremental financial analysis approach to this type of evaluation will be discussed in Section 6.6.

There are a number of iterations which must be performed as the various assumptions are examined. In this simple example, there were five assumptions made in order to proceed:

- Assumption 1. Cost-recovery values.
- Assumption 2. Commodity price.
- Assumption 3. Destination options.
- Assumption 4. Mill cutoff grade.
- Assumption 5. Yearly product mix.

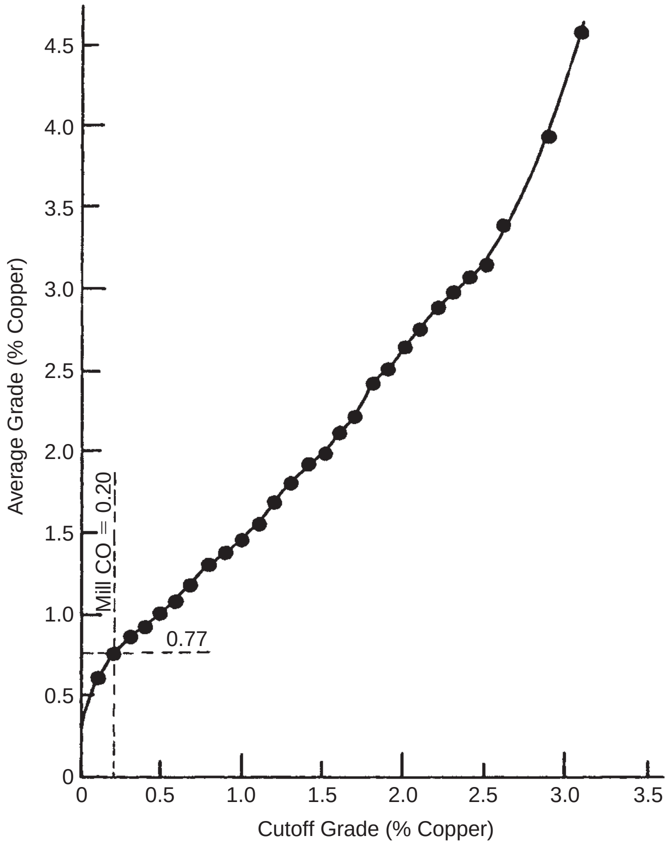


Figure 6.8. Average mill feed grade for a 0.3 cutoff grade.

Table 6.4. Summary for the two mill cutoff grades.

Quantity	Mill cutoff grade (% Cu)	
	0.2	0.4
Tons mined	30.8×10^6	30.8×10^6
Tons milled	10.8×10^6	7.8×10^6
Tons waste	20.0×10^6	23.0×10^6
Avg. ore grade	0.77%	0.92%
Milling rate (tpd)	3350	2800
Mining rate (tpd)	9550	11,100
Mine life (yrs)	12.9	11.1
Overall SR	1.85	2.95
Recovered copper (tons)	64,500	55,500

Clearly, a large number of combinations must be considered, many of which could provide satisfactory results. Because of the many uncertainties associated with the grade, tonnage, price, and cost data, the meaning of some sort of 'optimum' solution is tenuous at best.

6.3 TAYLOR'S MINE LIFE RULE

Taylor (1977, 1986, 1991) has, over the years, provided some very practical and useful advice regarding mine life. This section is based on extractions from his writings. In theory, it is possible to calculate an 'optimum' rate of extraction from an orebody. To do this, however, knowledge or precise assumption of the total tonnage and its sequential grades (including the effects of varying the cutoff grade), and of all costs and product prices throughout the project life is required. This information is unavailable for early studies and may indeed never reach high certainty or even be necessary.

Even with certain knowledge of everything, optimizing theory yields different answers depending on what quantity is selected to be maximized. The maximized quantity might be total profit, total cash flow, the net present value or the internal rate of return. Furthermore, the peaks of such curves are rather flat. Thus when allowing for the practical inaccuracies of data, the calculated results cannot be considered critical. Hence, although valid, a highly mathematical approach to mine life determination is seldom of practical use. Other ways must be found to provide a reasonable first approximation for mine life.

Too low a production rate sacrifices possible economies of scale and defers possible profits too far into the future. Conversely, too high a rate may drive up the project's capital cost beyond any ability to repay within the shortened life. Too high an output may be unsalable, while too short a life for a large enterprise may be wholly undesirable on social grounds. One hazard of short life mines merits special mention. Since base metal prices seem to move in cycles of four to seven years' duration, an operation of under four years' life may find itself depleting all its ore in a trough of the price cycle, and be left with neither ore nor time to recover.

In real life, rates of output are strongly limited or influenced by practical problems. One of the most important of these is working space. A mine may be able to increase output as it gets older solely because its ever expanding workings offer more points of attack.

In an open pit the working space for equipment and hence maximum production rate tends to vary with the area (ft^2) exposed while tonnage varies with volume (ft^3). Thus one might expect the production rate for groups of more-or-less similarly shaped orebodies to be proportional to the two-thirds power of the orebody tonnage. The life would then be proportional to the cube root of that tonnage.

Taylor (1977) studied many actual projects (some operating and others only planned) involving a wide range of orebody sizes, and shapes (other than thin deposits of very large lateral extent), for which the total ore reserves were reasonably well known before major design commenced. He found that the extraction rates seemed proportional to the three-quarters power of the ore tonnage rather than the two-thirds power. The designed lives were proportional to the fourth root of the tonnage.

This lead to the formulation of Taylor's rule, a simple and useful guide that states:

$$\text{Life (years)} \cong 0.2 \times \sqrt[4]{\text{Expected ore tonnage}} \tag{6.1}$$

In this equation, it is immaterial whether short or metric tons are used. It is more convenient to use quantities expressed in millions and except for special conditions, the practical range of variation seems to lie within a factor of 1.2 above and below. The rule can thus be restated as:

$$\text{Life (years)} \cong (1 \pm 0.2) \times 6.5 \times \sqrt[4]{\text{Ore Tonnage in millions}} \tag{6.2}$$

Table 6.5. Mine life as a function of ore tonnage (Taylor, 1977).

Expected ore (10^6 tons)	Median life (years)	Range of lives (years)	Median output (tpd)	Range of outputs (tpd)
0.5	3.5	4.5–3	80	65–100
1.0	6.5	7.5–5.5	450	400–500
5	9.5	11.5–8	1500	1250–1800
10	11.5	14.9–5	2500	2100–3000
25	14	17–12	5000	4200–6000
50	17	21–14	8400	7000–10,000
100	21	25–17	14,000	11,500–17,000
250	26	31–22	27,500	23,000–32,500
350	28	33–24	35,000	30,000–42,000
500	31	37–26	46,000	39,000–55,000
700	33	48–28	60,000	50,000–72,000
1000	36	44–30	80,000	65,000–95,000

At a preliminary stage, 'ore tonnage' could represent a reasonable though not optimistic estimate of the ore potential. Later, it could comprise the total of measured and indicated ore, including probable ore, but excluding possible or conjectural ore.

This empirical formula generates the values presented in Table 6.5.

The rule provides an appropriate provisional output rate for preliminary economic appraisals and will define a range of rates for comparative valuation at the intermediate stage after which a preferred single rate can be selected for use in the feasibility study.

6.4 SEQUENCING BY NESTED PITS

Of the various techniques used to develop mining sequences, the most common is to produce a nest of pits corresponding to various cutoff grades. From a practical point of view this is accomplished by varying the price of the metal (commodity) being extracted. The final pit limit is generally determined using the most likely price. For prices lower than this value, successively smaller pits will be produced. The pits will migrate toward the area of highest grade and/or lowest amount of stripping. This will be illustrated by way of an example.

The topographic map for a molybdenum prospect (Suriel, 1984) is shown in Figure 6.9. As can be seen, the majority of the deposit area has moderate surface relief with the exception of some steeper topography in the north-central project area. The relative position of the deposit is southwest of the hill. It is shown by the dashed lines on the figure.

A grade block model has been prepared using blocks 50 ft × 50 ft × 50 ft. The tonnage factor is 12.5 ft^3/st hence each block contains 10,000 tons. The following data were used in preparing the grade block model and in running the floating cone.

Mining cost = \$0.74/st

Processing cost = \$1.89/st ore

General and administrative (G&A) cost = \$0.67/st ore

Mill recovery = 90%

Selling price = \$6/lb contained molybdenum (F.O.B. mill site)

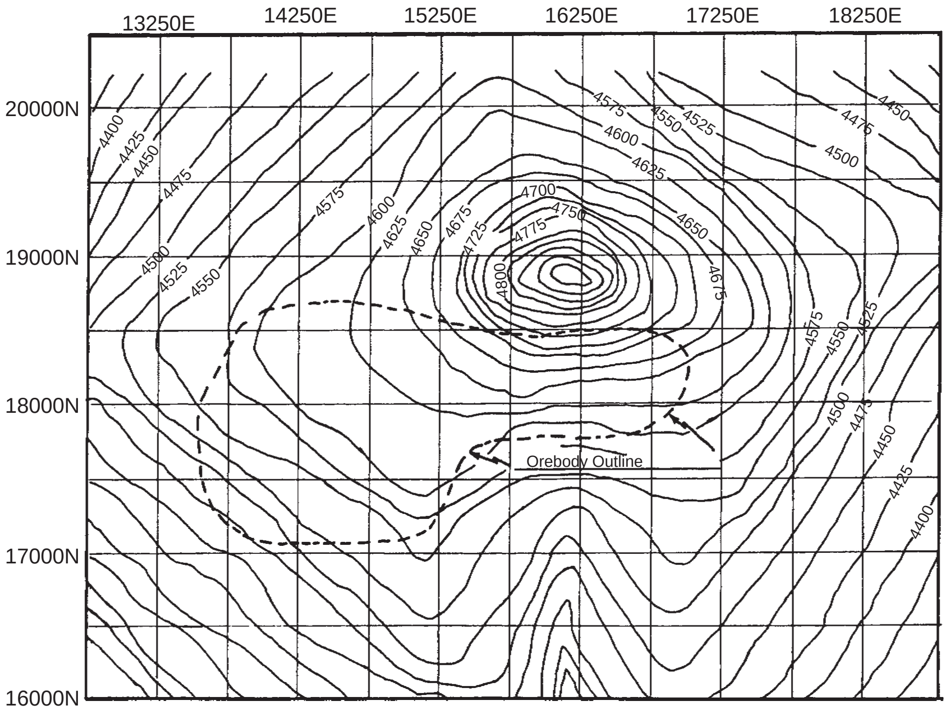


Figure 6.9. Topographic map of the project area (Suriel, 1984).

The ore grade is expressed in terms of % MoS_2. Since one pound of MoS_2 contains 0.60 lbs Mo, the equivalent price is \$3.24/lb MoS_2.

In determining the final pit limits, the costs and revenues involved in mining and processing 1 ton of material containing X% MoS_2 are first determined.

$$\text{Cost (\$/st)} = 0.74 + 0.67 + 1.89 = \$3.30/\text{st}$$

$$\begin{aligned}\text{Revenue (\$/st)} &= \frac{X}{100} \times 2000 \times 0.90 \times \$6.00 \times 0.60 \\ &= 64.8X\end{aligned}$$

Equating costs and revenues

$$64.8X = 3.30$$

one finds that the breakeven grade (X%) is

$$X = 0.05\% MoS_2$$

This cutoff grade corresponds to a commodity price of \$6/lb contained molybdenum. Using these costs and the \$6/lb price one generates the pit shown in plan in Figure 6.10. It can be referred to as the \$6 pit or equivalently the 0.05% cutoff pit.

For this example two smaller pits will be created. If a price of \$2/lb Mo is selected instead, then by redoing the breakeven analysis one finds that

$$X = 0.15\%$$

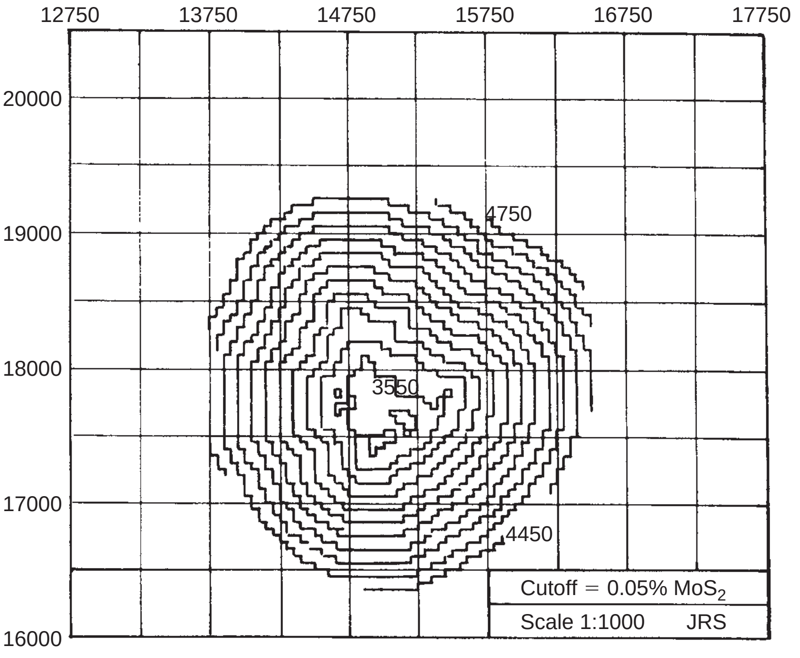


Figure 6.10. Pit outline for a 0.05 cutoff (Suriel, 1984).

The \$2 pit is shown in Figure 6.11. For a price of \$1.50/lb Mo, the breakeven grade is

$$X = 0.20\%$$

Figure 6.12 is a plan view of the resulting \$1.50 pit. The three nested pits are shown on section 18000 N in Figure 6.13.

With this approach one would begin (Phase I) with the mining of the 0.20% MoS_2 (\$1.50) cutoff pit. Phase II involves the material in the 0.15% MoS_2 (\$2) cutoff pit and finally, Phase III, the 0.05% MoS_2 (\$6) cutoff pit. Pits intermediate to these can be found by selecting the appropriate price. As can be seen in Figure 6.13, barren material overlies the orebody. All material down to the 4400 level will be stripped and sent to a waste dump. Its removal requires drilling and blasting. There is also some low grade material running 0–0.05% MoS_2below this level. The grade-tonnage distribution for the overall pit is shown in Table 6.6. There are 102,970,000 tons above the 0.05% cutoff. The average grade is 0.186%. The grade-tonnage distributions for the 3 mining phases are given in Table 6.7.

The average grade of the material above 0.05% for each of the 3 phases is:

$$\text{Phase I}: g = 0.225\%$$

$$\text{Phase II}: g = 0.182\%$$

$$\text{Phase III}: g = 0.176\%$$

An initial decision has been made to mine the orebody over a period of 15 years. Thus the average milling rate will be of the order of 6.9×10^6 tons/year. Assuming that the mill operates 250 days/year the daily milling rate is 27,500 tpd. In reviewing the level plans,

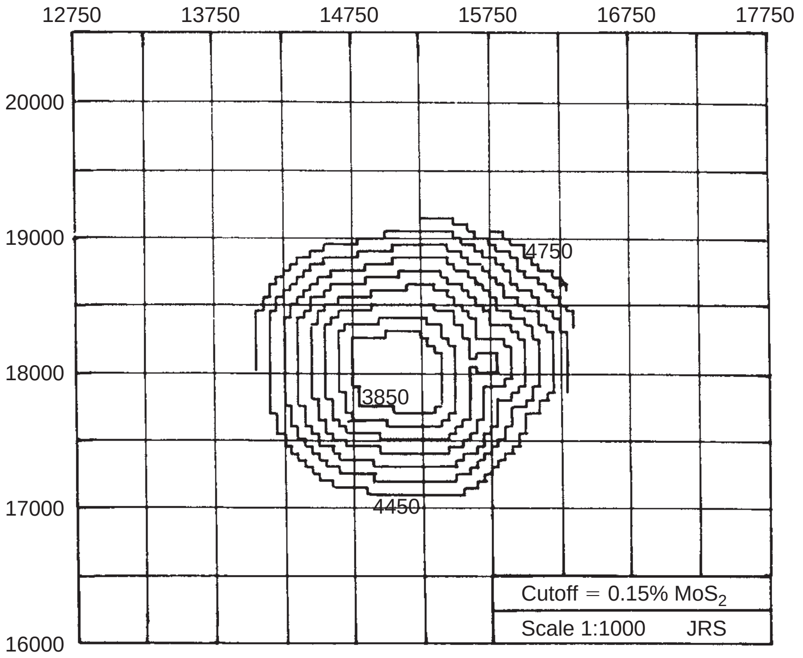


Figure 6.11. Pit outline for a 0.15 cutoff (Suriel, 1984).

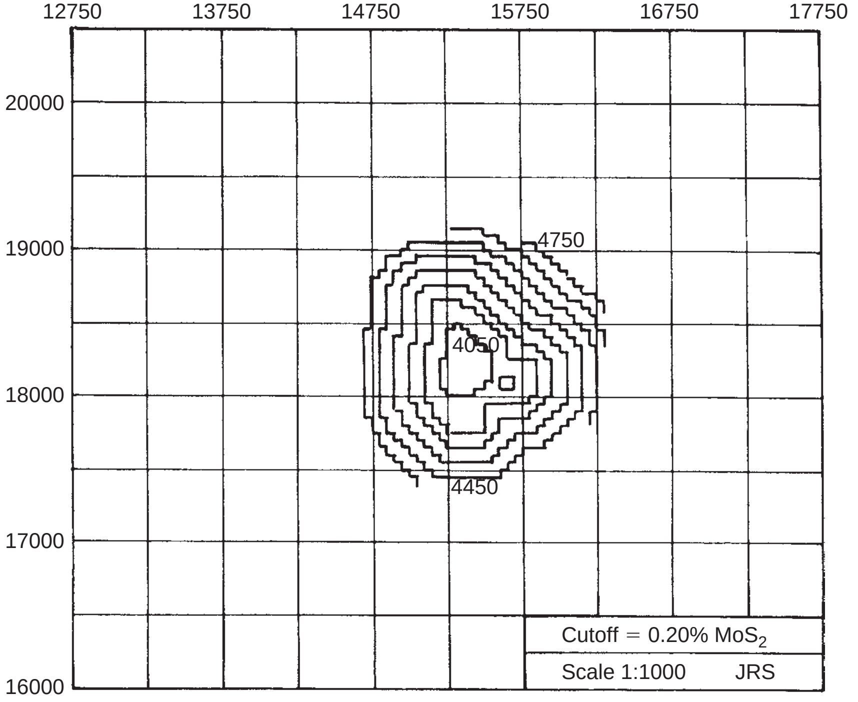


Figure 6.12. Pit outline for a 0.20 cutoff (Suriel, 1984).

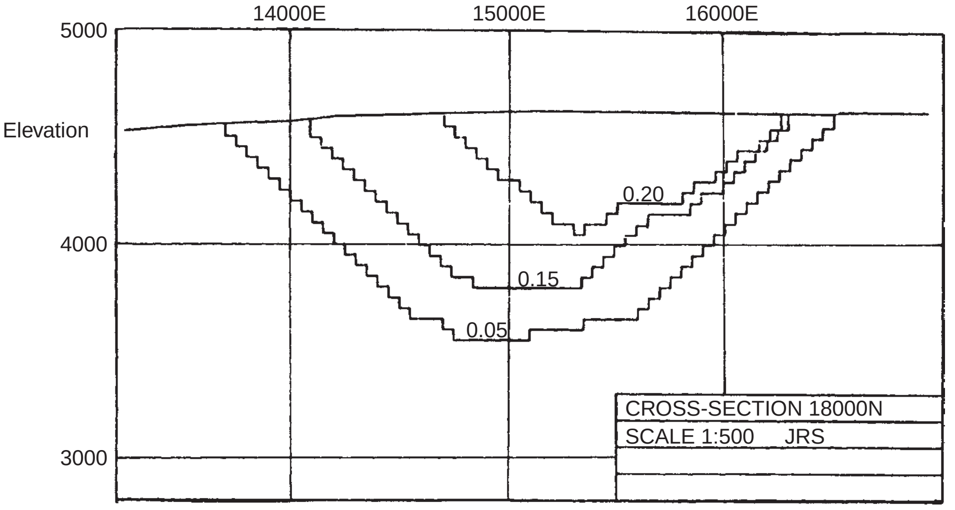


Figure 6.13. Cross-section 18000 N showing the three pit outlines (Suriel, 1984).

Table 6.6. Mineral inventory for the molybdenum pit.

Grade (% MoS_2)	Tons (10^3)
0 (overburden)	86,410
$0 < g < 0.05$	50,720
$0.05 \leq g < 0.10$	10,420
$0.10 \leq g < 0.15$	23,430
$0.15 \leq g < 0.20$	29,010
$0.20 \leq g < 0.25$	22,300
$g^* \geq 0.25$	17,810
	Total = 240,100

*The average grade is 0.30%.

Table 6.7. Mineral inventory by phase.

Grade (% MoS_2)	Tons (10^3)		
	Phase I	Phase II	Phase III
0 (overburden)	36,260	21,440	28,710
$0 < g < 0.05$	680	6170	43,870
$0.05 \leq g < 0.10$	180	1640	8600
$0.10 \leq g < 0.15$	1710	9700	12,020
$0.15 \leq g < 0.20$	4530	11,060	13,420
$0.20 \leq g < 0.25$	4260	5670	12,370
$g^* \geq 0.25$	5670	4860	7280

* The average grade is 0.30%.

it is seen that 57,700,000 tons of rock overburden (the Phase I and Phase II overburden) must be removed in a pre-production period in order to have 6 months ore supply available at the time of production. This leaves 182,400,000 tons to be mined (ore and waste) during the production period. Assuming that the mine also works 5 days/week, the daily mining rate is

$$\text{Mining rate} = 48{,}600\,\text{tpd}$$

The same equipment fleet will be used for the prestripping as the production mining. During the first year of stripping the average daily production rate will be assumed to be $\frac{1}{3}$ that maintained in later years due to equipment delivery, personnel training, and limited working place. Thus the time required for the prestripping is

$$t\ (\text{yrs}) = \frac{57{,}700{,}000}{(48{,}600(t-1) + 16{,}200 \times 1)250}$$
$$\cong 5.4\,\text{yrs}$$

To complete the stripping in 5 years the actual waste mining rate will be

$$\text{Year } 1 = 18{,}000\,\text{tpd}$$

$$\text{Years 2 to 5} = 53{,}200\,\text{tpd}$$

With this milling rate, there are approximately 2.4 years of reserves in Phase I and 4.8 years of reserves in Phase II. Phase III represents the reserves for years 7.2 through 15. The following data will be used:

Total material to be mined = 240,100,000 tons

Ore grade > 0.05% MoS_2 material = 102,970,000 tons

Average ore grade = 0.186% MoS_2

Mill recovery = 90%

Price = \$3.24/lb MoS_2

Mining rate = 48,600 tpd

Milling rate = 27,500 tpd

Operating time = 250 days per year

Total material to be pre-stripped = 57,700,000 tons

Pre-stripping period = 5 years

Stripping rate year 1 = 16,200 tpd

Stripping rate years 2–5 = 48,600 tpd

This example is continued in the cash flow calculations of Section 6.5.

6.5 CASH FLOW CALCULATIONS

In Section 6.4 the concept of nested pits was discussed with respect to a molybdenum orebody. In this section the example will be continued with consideration of the resulting cash

Table 6.8. Assumed schedule for the pre-production period (Suriel, 1984).

	Year						
	1	2	3	4	5	6	7
Exploration	✓	✓					
Property acquisition		✓					
Development			✓	✓	✓	✓	✓

flows. The basic ideas involved with cash flow calculations were introduced in Chapter 2. The specific application, however, has been left to this section. Each state, province, and country has specific rules and regulations regarding taxes, depreciation, depletion, royalties, etc. Even for a given state these rules change with time. Therefore, the authors have selected a somewhat simplified example originally presented by Suriel (1984) to illustrate the calculations involved. Hopefully the reader can adapt the procedures to fit the application at hand. The example applies roughly to an orebody located in the State of Colorado in the year 1983. Certain aspects of the laws in effect at the time such as minimum tax and investment tax credit have been left out since they add unnecessary complication to the example. It has been assumed that the mining company is a division of a profitable corporation. Costs involved in the development of this new mine will be expensed whenever possible.

In the mine life two periods will be considered:

– pre-production period, and
– production period.

Today there is generally a third period, the post-production or closure period in which final reclamation takes place. This is not covered here. The pre-production period which is assumed to require 7 years, can be broken down into 3 different and distinct expenditure categories:

– detailed exploration,
– property acquisition, and
– infrastructure and mine development.

A representative schedule of activities is illustrated in Table 6.8. The first two years are used for detailed exploration work. Property acquisition takes place in year 2. The following 5 years are required for infrastructure and mine development. Hence, the pre-production period requires a total of 7 years. Production is expected to take place over a period of 15 years.

The basic line items in the pre-production cash flow calculation are given in Table 6.9. A brief discussion of each line item will be presented below. The numerical values for this example have been inserted in Table 6.10.

Lines 1 and 2: *Project year and Calendar year*. The first calendar year in which major expenditures occur is 1984. For the cash flow calculations this is selected as project year 1. Discounting will be done back to the beginning of year 1984 (project year 1).

Line 3: *Capital expenditures*. The 'capital' expenditures include a variety of different types of items ranging from hardware (mine and mill equipment) to royalties and property taxes. They occur at varying times in the cash flow table.

Table 6.9. Typical pre-production cash flow categories (Suriel, 1984).

1. Project year	1	2	Last year of
2. Calendar year	1984	1985...	pre-production
3. Capital expenditures:			
4. Property acquisition			
5. Royalties			
6. Exploration			
7. Development			
8. Mine and mill buildings			
9. Mine and mill equipment			
10. Property tax			
11. Working capital			
12. Total capital expenditures			
13. Cash generated due to tax savings:			
14. Exploration			
15. Development			
16. Depreciation			
17. Property tax			
18. Total cash generated			
19. Net cash flow			

Line 4: *Property acquisition costs $2,000,000*. It takes place in project year 2. This expenditure is a primary component of the depletion account which controls unit depletion allowance.

Line 5: *Royalties*. In addition to receiving $2,000,000 for the property, the original owners will receive a royalty. Normally these royalties are a certain percentage of the net smelter return (NSR). In this case the royalty is 5% of the mill return. No royalty is, therefore, paid during the pre-production period.

Line 6: *Exploration*. As defined in the U.S. tax law, exploration costs are those incurred prior to any development of the deposit. Exploration refers to the activities performed in order to determine the location, size, extent, quality, and quantity of a mineral occurrence. The reader is encouraged to compare this interpretation with that appropriate at the specific location and time. The U.S. Internal Revenue Service (IRS) allows the firm to choose between two separate methods in accounting for exploration expenditures: capitalize exploration costs into the depletion account and allocate them over time as production occurs, or treat exploration costs as annual expenses. For this example, the exploration costs are broken down as follows:

$$\text{Year } 1 = \$1{,}500{,}000$$

$$\text{Year } 2 = \$1{,}000{,}000$$

To simplify the example, these have been fully expensed in the years in which they occurred.

Line 7: *Development*. In this example site development consists of three individual line items:

(a) Site preparation.
(b) Pre-production stripping.
(c) Plant site cleaning and mass excavation.

Table 6.10. Pre-production cash flow table ($1,000).

1. Project year	1	2	3	4	5	6	7	
2. Calendar year	1984	1985	1986	1987	1988	1989	1990	
3. Capital expenditures:								Total
4. Property acquisition	0	2000	0	0	0	0	0	2000
5. Royalties	0	0	0	0	0	0	0	
6. Exploration	1500	1000	0	0	0	0	0	2500
7. Development	0	0	3330	9842	9842	9842	9842	42,698
8. Mine/mill buildings	0	0	0	4836	13,057	1711	9948	29,552
9. Mine/mill equipment	0	0	10,000	0	9981	25,295	2999	48,275
10. Property tax	0	0	255	378	965	1654	1985	5237
11. Working capital	0	0	0	0	0	0	8353	8353
12. Total capital expenditures	1500	3000	13,585	15,005	33,744	38,349	33,127	138,310
13. Cash generated due to tax savings								
14. Exploration	690	460	0	0	0	0	0	1150
15. Development	0	0	1532	4527	4527	4527	4527	19,640
16. Depreciation	0	0	0	920	920	920	920	3680
17. Property tax	0	0	117	150	397	690	819	2173
18. Total cash generated	690	460	1649	5597	5844	6137	6266	26,643
19. Net cash flow	−810	−2540	−11,936	−9408	−27,900	−32,212	−26,861	−111,667

General site preparation (cost = $1,729,000) as well as the plant site cleaning (cost = $859,000) will occur in project year 3. The total amount of pre-production stripping is 57,700,000 tons. Since the expected cost per ton is $0.74, the total cost is $42,698,000. The stripping schedule is as follows:

Project year	Amount stripped (tons)	Cost ($)
3	4,500,000 (18,000 tpd)	3,330,000
4	13,300,000 (53,200 tpd)	9,842,000
5	13,300,000	9,842,000
6	13,300,000	9,842,000
7	13,300,000	9,842,000

These costs are expensed in the year incurred.

Line 8: *Mine and mill buildings*. There are 7 items which fall into this category. They are indicated below together with their cost and the project year in which the expenditure is made.

Item	Project year	Cost ($)
1. Concrete foundation and detailed excavation	4	$4,836,000
2. Open-pit maintenance facilities	7	$6,623,000
3. Concentrator building	5	$13,057,000
4. Concentrate storage and loading	7	$139,000
5. Capital cost of general plant service	6	$1,711,000
6. Tailings storage	7	$1,775,000
7. Water supply	7	$1,411,000

These items, considered as improvements to the property, will be depreciated (straight line) over a 20 year period beginning in production year 1. Capital gains and losses will be ignored.

Line 9: *Mine and mill equipment*. There are 6 items which fall into this category. They are indicated below together with their cost and the project year in which the expenditure is made.

Item	Project year	Cost ($)
1. Open-pit equipment	3	$10,000,000
2. Crushing plant, coarse ore storage and conveyors	5	$9,981,000
3. Grinding section and fine ore storage	6	$16,535,000
4. Flotation section	6	$6,985,000
5. Thickening and filtering	6	$1,775,000
6. Electric power supply and distribution	7	$2,999,000

The open-pit equipment is broken down as follows:

(a) Shovels (3-15 yd^3) = $3,000,000
(b) Trucks (12-150 ton) = $5,000,000
(c) Drills, graders, dozers = $2,000,000

The equipment in categories (b) and (c) will be replaced every 5 years. Since this fleet will be used for the stripping as well as the production mining, there will be a capital expense of $7,000,000 in project years 1, 6 and 11. The shovels are expected to last the life of the mine. Straight line depreciation over a period of 5 years is used. There is no salvage. The equipment will be put into use in the year purchased, hence, depreciation will begin then.

Items 2 through 6 will be depreciated over a 7 year period (straight line, no salvage) beginning in production year 1.

Line 10: *Property tax*. This tax, sometimes called ad valorem tax is one of the most common types of state tax. Property tax in Colorado is assessed on: (a) personal and real property and (b) ore sales.

Colorado taxes personal and real property at 30% of the actual value of the property. In this case personal property is that listed in line 9 and real property in line 8. The taxable value for real and personal property is determined by its base year value. The tax is then determined by multiplying the total assessed value by a rate called the 'pro mille' levy or in the U.S. simply the 'mill' levy. In Colorado the 'pro mille' levy varies from county to county. In this case study that used is 85 (corresponding to that applied by Jefferson County). This means that the appropriate factor is $^{85}/_{1000}$ or 0.085. The assessed value for the calculation of the property tax on ore sales is: (a) 25 percent of gross proceeds or (b) 100 percent of net proceeds, whichever is greater.

Gross proceeds are defined as the gross value of the ore produced minus treatment, reduction, transportation and sales cost at the mine mouth. Net proceeds is equal to the gross proceeds minus all costs associated with extraction of the ore. In Colorado, royalties are not deductible for either case. The pro mille levy is applied to the assessed value.

Line 11: *Working capital*. This represents the amount of money necessary to cover operating costs during a portion of the project's life. Working capital consists of cash, inventories (parts, supplies and concentrate) and accounts receivable. Working capital was estimated at four months of operating costs. It was allocated to the last year of the pre-production period before production start up (project year 7). The account is maintained during the production period and recovered at the end of project life. In this case, the amount of required working capital is $12,383,000.

Line 12: *Total capital expenditures*. This is the sum year by year of line items 4 through 11.

Line 13: *Cash generated due to tax savings*. As indicated earlier, the mining company is one division of a large profitable corporation. The corporate structure enables the company to expense most costs whenever possible and generate tax savings during the preproduction period. The federal tax rate is 46 percent.

To illustrate this assume that the corporation has an income subject to federal tax, prior to including this mining venture, of $10,000,000. At a tax rate of 46%, the corporation would pay $4,600,000 in federal tax. However, in project year 1, the mining division incurs an exploration expense of $1,500,000. If this is considered as an expense of the corporation, then the taxable income of the corporation would drop from $10,000,000 to $8,500,000. The tax on this amount is $3,910,000. Hence, there is a tax savings of $690,000 for the corporation. This has been included on the cash flow table for the mining company (Table 6.10) under 'Cash generated due to tax savings – Exploration'. Similar 'tax savings' occur with regard to development, depreciation and property tax when they are applied against other income.

Line 14: *Exploration*. The exploration expenses result in a tax savings of \$690,000 (0.46 × \$1,500,000) in project year 1 and \$460,000 in project year 2.

Line 15: *Development*. Stripping, site preparation, etc., costs are multiplied by the tax rate and included here.

Line 16: *Depreciation*. Straight line depreciation of the mining equipment begins in project year 3. The yearly amount is multiplied by the tax rate.

Line 17: *Property tax*. The property tax is the assessed value multiplied by the tax rate.

Line 18:*Total cash generated*. The items in lines 14 through 17 are summed and entered here.

Line 19: *Net cash flow*. The total capital expenditures (line 12) are subtracted from the total cash generated (line 18). This is the net cash flow.

The basic line items in the production period cash flow calculation are given in Table 6.11. A brief discussion of these will be presented below.

Line 1, 2, and 3: *Production, project and calendar year*. The first year of production is 1991 which is the eighth year of the project. In doing the discounted cash flows and NPV calculations, they will be brought back to the beginning of the project.

Line 4: *Revenue.* Revenue is calculated by multiplying concentrate tonnage by the price of molybdenite concentrate at the mill site. The molybdenite concentrate price is estimated

Table 6.11. Typical production period cash flow categories (Suriel, 1984).

1. Production year
2. Project year
3. Calendar year
4. Revenue
5. Royalty
6. Net revenue
7. Mining cost
8. Processing cost
9. General cost
10. Property tax
11. Severance tax
12. Depreciation
13. State income tax
14. Net income after costs
15. Depletion
16. Taxable income
17. Federal income tax
18. Profit
19. Depreciation
20. Depletion
21. Cash flow
22. Capital expenditures
23. Working capital
24. Net cash flow

at $6 per pound of contained molybdenum. Concentrate tonnage is equal to ore tonnage multiplied by average grade, the mill recovery, and by the concentrate percent of Mo. (MoS_2 contains about 60% Mo.)

Line 5: *Royalty*. The royalty is 5 percent of the revenue.

Line 6: *Net revenue*. This is the difference between revenue and the royalty payment.

Line 7: *Mining cost*. The mining cost depends somewhat on the production rate as can be seen in Figure 6.14 and in Table 6.12. Because 57,700,000 tons have been mined in the pre-production period, 182,400,000 tons remain. The length of the production period has been selected as 15 years, and the mine will work 5 days per week (250 days per year). Thus the mining rate is 48,640 tons per day. From Figure 6.14, the mining cost of $0.77/ton has been selected.

Line 8: *Processing cost*. The mill will also run 5 days per week, 250 days per year. The mill cutoff grade has been selected as 0.05% MoS_2 hence there are 102,970,000 tons to be processed. The milling rate is therefore 27,460 tons/day and yearly ore production is 6,865,000 tons. The milling cost selected from Figure 6.15 and Table 6.12 is $1.93/ton.

Line 9: *General and administrative cost*. This cost has been computed per ton of ore processed. Its value as can be seen in Figure 6.16 and Table 6.12 is $0.67/ton. In some operations there is a G&A cost attached to waste removal as well. This has not been done here.

Line 10: *Property tax*. The property tax is computed in the same way as was discussed for the pre-production period. The assessed value is 30% of the initial cost and the pro mille rate is 85 (0.085). The value at the end of the pre-production period is $77,827,000. Applying the 30% assessed valuation and the 85 pro mille levy, the annual property tax is $1,985,000.

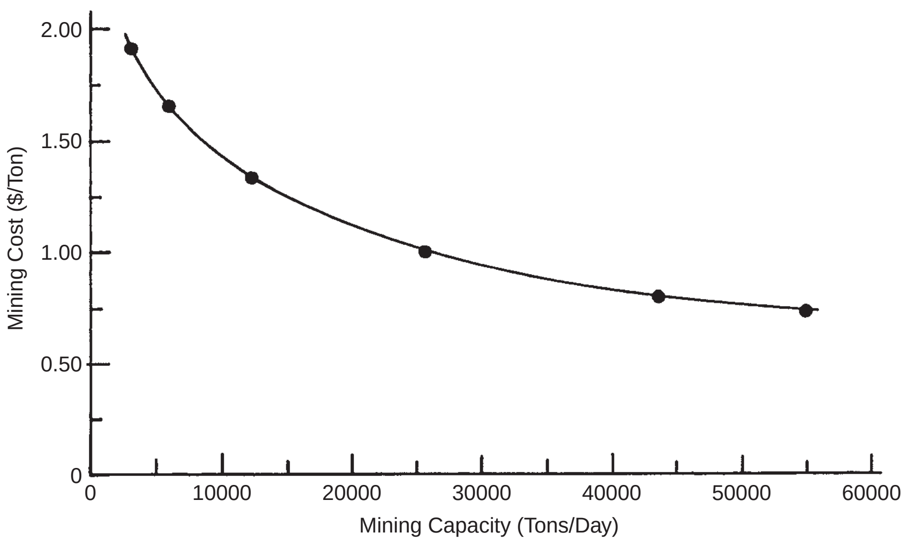


Figure 6.14. Mining cost versus mining capacity (Suriel, 1984).

This annual value will be carried throughout the production period. The book value has not been used and the effect of replacement capital has not been included.

The gross revenue from annual ore sales is $82,742,000. The net revenue after subtracting mining, processing, and general costs is $57,683,000. Note that royalties have not

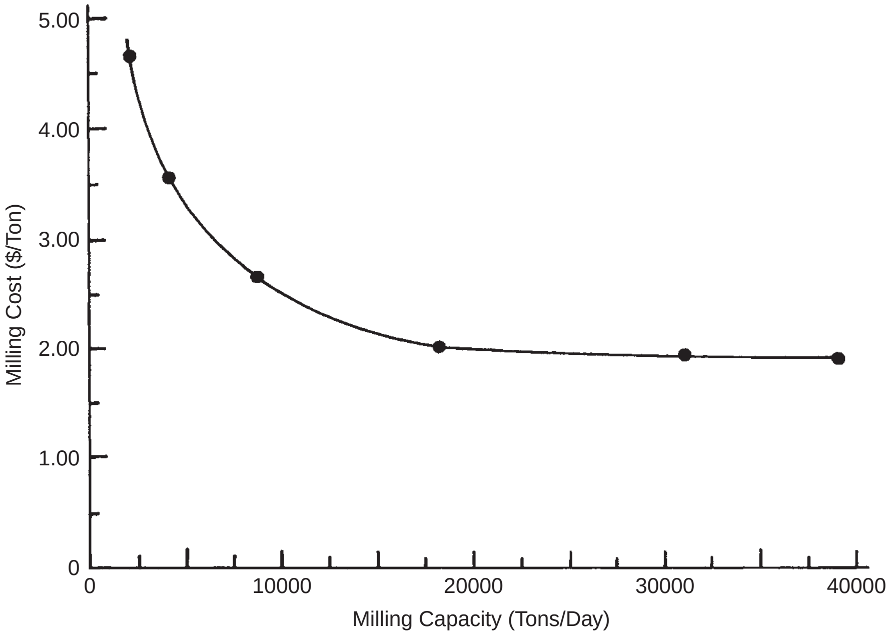


Figure 6.15. Milling cost versus milling capacity (Suriel, 1984).

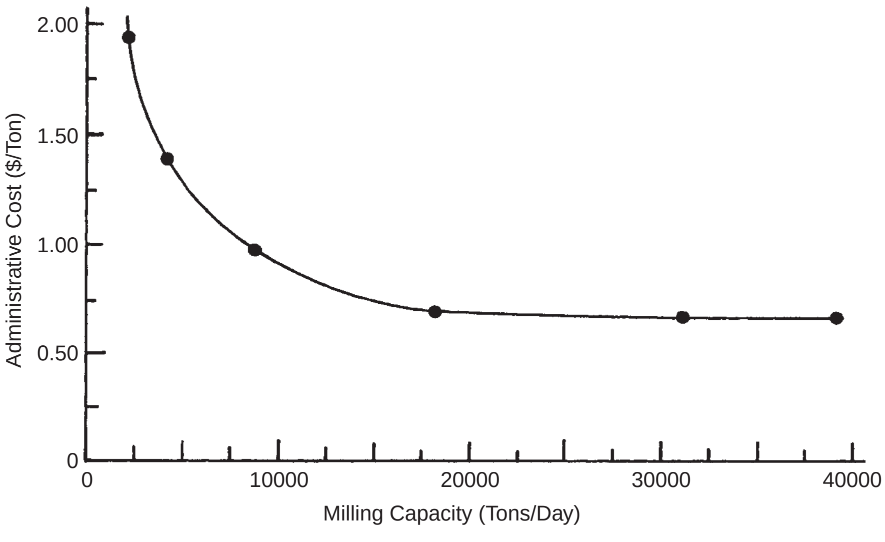


Figure 6.16. Administrative cost versus milling capacity (Suriel, 1984).

been subtracted. Applying the two rules (25 percent of gross proceeds or 100 percent of net proceeds) one finds that the greatest amount (which becomes the assessed valuation) is $57,683,000. Applying the 85 pro mille levy, this contribution to the property tax is $4,903,055. Thus the total annual property tax is $6,888,055.

Line 11: *Severance tax*. A severance tax is levied for removing or 'severing' the mineral from the earth and the state/country. Colorado imposes a such severance tax on molybdenum. It was imposed on production at a rate of $0.15 per ton of ore.

Line 12: *Depreciation*. The depreciation allowance is a deduction over a period of years for use, deterioration, wear and tear of depreciable assets used in generating income for the project. Depreciable assets for a mining project are grouped into either personal or real property. Mine and mill equipment are considered personal property. The total value is $48,275,000. The open pit mining equipment is depreciated over 5 years. The initial investment is $10,000,000 in project year 3 while replacement equipment (trucks, drills, etc.) at a cost of $7,000,000 is purchased in project years 8, 13, and 18. The remainder of the mine and mill equipment (initial cost of $38,275,000) is depreciated over a seven year period beginning at the start of production. Real property, including mine and mill buildings and improvements, amount to $29,552,000. A twenty year life is used for real property with depreciation beginning at the start of production. Thus the depreciation per year is $1,477,600. The depreciation by class and year is shown in Table 6.13.

Line 13: *State income tax*. The state tax is 5% of the net income. In computing the net income the following deductions are allowed:

- royalties,
- operating costs,
- depreciation,
- property tax,
- severance tax, and
- depreciation.

For production years 1 through 7 the state income tax is $1,864,000. For years 8 through 15 it is $2,138,000.

Line 14: *Net income after costs*. This is the net revenue (line 6) minus lines 7 through 13.

Line 15: *Depletion*. In order to recognize that minerals, oil, and gas are non-renewable assets which are depleted through production, the U.S. government permits a depletion allowance to be deducted prior to the calculation of federal taxable income. For mineral resources the depletion allowance is calculated and claimed by either of two methods, whichever gives the largest amount of pre-tax deduction. It is permissible to change methods from year to year. The two methods for calculating the depletion allowance are statutory depletion and cost depletion.

For statutory depletion, the amount which can be deducted is the smaller of: (a) 50 percent of net income or (b) a certain percentage of net revenue (revenue minus royalties). For the (a) calculation, net income is defined as revenues minus royalties, operating costs, state taxes, and depreciation. For (b) the percentage depletion for molybdenum is 22 percent. Note that the amount of depreciation varies throughout the life of the mine. For the first 7 years of production it is $8,346,000 per year. The annual net income during this period is $35,418,000 and hence, the allowable depletion on this basis is $17,709,000.

Table 6.12. Assumed operating cost data (Suriel, 1984).

Cutoff grade % MoS_2	Geologic reserves ($\times 10^6$ st)	90% geologic reserves ($\times 10^6$ st)	Average grade % M_0S_2	Tonnage milled (tpd)	Tonnage mined (tpd)	Total mining cost ($/ton)	Total milling cost ($/ton)	General & administrative cost ($/ton)	Total operating cost ($/ton)
0.30	12.680	11.412	0.40	2173	3043	1.91	4.64	1.93	8.48
0.25	24.390	21.950	0.34	4181	5854	1.65	3.55	1.39	6.35
0.20	51.300	46.170	0.28	8794	12,312	1.33	2.65	0.97	4.95
0.15	107.010	96.309	0.22	18,345	25,682	1.00	1.98	0.68	3.66
0.10	181.490	163.341	0.18	31,113	43,558	0.80	1.91	0.67	3.38
0.05	228.280	205.452	0.16	39,134	54,787	0.74	1.89	0.67	3.30

Table 6.13. Depreciation by class and project year.

Depreciation class (yrs)	Project year depreciation ($1000)																					
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
5	0	0	2000	2000	2000	2000	2000	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400	1400
7	0	0	0	0	0	0	0	5468	5468	5468	5468	5468	5468	5468	0	0	0	0	0	0	0	0
20	0	0	0	0	0	0	0	1478	1478	1478	1478	1478	1478	1478	1478	1478	1478	1478	1478	1478	1478	1478
Total	0	0	2000	2000	2000	2000	2000	8346	8346	8346	8346	8346	8346	8346	2878	2878	2878	2878	2878	2878	2878	2878

The annual net revenue is \$78,605,000. Taking 22% of this number yields \$17,293,000. The lesser of these two numbers is \$17,293,000.

Cost depletion is based on the cost of the property, non-expensed exploration costs, number of units of mineral sold during the year and reserves available in the deposit at the end of the year. Cost depletion is calculated by multiplying the adjusted basis, (equivalent to all property acquisition costs plus capitalized exploration costs) by the tons mined during the year divided by remaining reserves. The adjusted basis is reduced each year by the depletion allowance claimed.

$$\text{Cost depletion} = (\text{Adjusted basis}) \times \frac{\text{Mineral units removed during the year}}{\text{Mineral units recoverable at start of the year}}$$

$$\begin{aligned}\text{Adjusted basis} &= \text{Cost basis} \pm \text{Adjustments}\\ &\quad -\text{Cumulative depletion allowance claimed}\end{aligned}$$

In this case the cost of the property was \$2,000,000. For the first year, 6,865,000 tons of ore are mined. The initial ore tonnage is 102,970,000. Hence, for year 1 the allowable cost depletion would be

$$\$2{,}000{,}000 \times \frac{6{,}865{,}000}{102{,}970{,}000} \times \$133{,}340$$

The greater of the cost and percentage depletion values is chosen. Thus, one chooses \$17,293,000 to be deducted. However, according to the tax rules one must recapture the exploration expense of \$2,500,000 by reducing the amount of depletion earned. Hence, in year 1, the allowed depletion deduction is \$17,293,000 − \$2,500,000 = \$14,793,000.

In year 2 the statutory depletion allowance is \$17,293,000. For the cost depletion calculation the adjusted basis is

$$\text{Adjusted basis} = \text{Cost basis} - \text{Cumulative depletion}$$

The cost basis is \$2,000,000. The cumulative depletion already taken (through project year 8) is \$14,793,000. Hence the adjusted basis is

$$\text{Adjusted basis} = \$2{,}000{,}000 - \$14{,}793{,}000 = -\$12{,}793{,}000$$

The cost of the property was fully recovered through depletion in production year 1 and hence only statutory depletion is used for the remaining mine life. In production year 2, a full depletion of \$17,293,000 is taken since all previously expensed exploration costs have been recaptured.

Line 16: *Taxable income*. This is the net income after cost (line 14) minus depletion (line 15).

Line 17: *Federal income tax*. In this case the federal income tax is 46% of the taxable income.

Line 18: *Profit*. The profit is the taxable income minus the federal income tax.

Line 19 and 20: *Depreciation and depletion*. These two items which had been previously considered as expenses when computing taxes are now added to the profit to arrive at a cash flow.

Line 21: *Cash flow*. This is the sum of lines 18 through 20.

Line 22: *Capital expenditures*. In years 1, 6, and 11 new trucks, drills, dozers, etc., are purchased. The capital cost involved in each of these years is \$7,000,000.

Line 23: *Working capital*. The working capital was allocated to the last year of the pre-production period before production start-up. The account is maintained during the production period and recovered at the end of project lift (project year 22).

Line 24: *Net cash flow*. The net cash flow for any particular year is the sum of lines 21, 22, and 23.

The cash flows during the production period are summarized in Table 6.14.

6.6 MINE AND MILL PLANT SIZING

6.6.1 *Ore reserves supporting the plant size decision*

In the preceding sections, the final pit limits have been determined using the following steps:

1. The metal content of a block together with forecasted sales prices and estimated full scale plant metallurgical recoveries are used for determining the revenue.
2. Production costs through to sales and shipping costs are estimated.
3. A net value is calculated for each block. A block with a net value equal to zero is one where the revenues equal the production costs. The grade corresponds to a mining cutoff.
4. A computer technique (such as the floating cone) is used to identify all blocks capable of paying for the costs of uncovering them. These are included within the final pit.
5. A mineral inventory – ore reserve is constructed.
6. Milling and mining rates are assumed and the associated capital costs determined. For example a series of different milling cutoff grades might be applied to the total pool of material. Mill ore quantities would be calculated. These reserves would be used to size the mill plant.
7. A financial analysis would be conducted for each alternative to determine the NPV, internal ROR, etc. The best option would be selected.

As can be seen, the introduction of the capital cost takes place at a very late stage. The profit is an output rather than an input. After having gone through this entire process, it may be that the rate of return is too low. The process would then have to be repeated focussing on the higher grade ore, using a different (lower) production rate, reducing the investment in mine and mill plant, etc. In the process just described a variety of materials of different economic value are being mined at any one time. Those with the highest values contribute more to paying off the investment and profit than those of lower value. The summing or integration of these values occurs for a given production year. This overall value is used as an entry in the cash flow table. To illustrate this concept, assume that there are 60 ore blocks to be mined in a given period. The net value distribution (at step 3, prior to introducing the capital cost and profit) is as follows:

Grade	Net value/block	No. of blocks	Net value
g_1	\$0	10	\$0
g_2	\$10	10	\$100
g_3	\$20	10	\$200
g_4	\$30	10	\$300
g_5	\$40	10	\$400
g_6	\$50	10	\$500

Table 6.14. Cash flow during mine production life ($1000).

1. Production year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	
2. Project year	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
3. Calendar year	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
4. Revenue	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	82,742	1,241,130
5. Royalty	4137	4137	4137	4137	4137	4137	4137	4137	4137	4137	4137	4137	4137	4137	4137	62,055
6. Net revenue	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	78,605	1,179,075
7. Mining cost	9363	9363	9363	9363	9363	9363	9363	9363	9363	9363	9363	9363	9363	9363	9363	140,445
8. Processing cost	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	13,249	198,735
9. General cost	4600	4600	4600	4600	4600	4600	4600	4600	4600	4600	4600	4600	4600	4600	4600	69,000
10. Property tax	6888	6888	6888	6888	6888	6888	6888	6888	6888	6888	6888	6888	6888	6888	6888	103,320
11. Severance tax	1030	1030	1030	1030	1030	1030	1030	1030	1030	1030	1030	1030	1030	1030	1030	15,450
12. Depreciation	8346	8346	8346	8346	8346	8346	8346	2878	2878	2878	2878	2878	2878	2878	2878	81,446
13. State income tax	1756	1756	1756	1756	1756	1756	1756	2030	2030	2030	2030	2030	2030	2030	2030	28,532
14. Net income after costs	33,373	33,373	33,373	33,373	33,373	33,373	33,373	38,567	38,567	38,567	38,567	38,567	38,567	38,567	38,567	542,147
15. Depletion	14,793	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	256,895
16. Taxable income	18,580	16,080	16,080	16,080	16,080	16,080	16,080	21,274	21,274	21,274	21,274	21,274	21,274	21,274	21,274	285,252
17. Federal income tax	8547	7397	7397	7397	7397	7397	7397	9786	9786	9786	9786	9786	9786	9786	9786	131,217
18. Profit	10,033	8683	8683	8683	8683	8683	8683	11,488	11,488	11,488	11,488	11,488	11,488	11,488	11,488	154,035
19. Depreciation	8346	8346	8346	8346	8346	8346	8346	2878	2878	2878	2878	2878	2878	2878	2878	81,446
20. Depletion	14,793	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	17,293	256,895
21. Cash flow	33,172	34,322	34,322	34,322	34,322	34,322	34,322	31,659	31,659	31,659	31,659	31,659	31,659	31,659	31,659	492,376
22. Capital expenditures	7000	0	0	0	0	7000	0	0	0	0	7000	0	0	0	0	21,000
23. Working capital	0	0	0	0	0	0	0	0	0	0	0	0	0	0	8353	8353
24. Net cash flow	26,172	34,322	34,322	34,322	34,322	27,322	34,322	31,659	31,659	31,659	24,659	31,659	31,659	31,659	23,306	463,023

There are 50 associated waste blocks each carrying a value of −\$20. Thus the apparent total net value for this period would be \$500. At first glance this might appear good. However when a cash flow calculation (now including the capital cost and profit) is made, it is found that the desired rate of return is too low. The required net value at step 3 is \$1,100 rather than \$500. Since there are 60 ore blocks involved, this would mean an average 'assessment' of \$10/block. If this is done then the modified net value distribution becomes

Grade	Net value/block	No. of blocks	Net value
g_1	−\$10	10	−\$100
g_2	\$0	10	\$0
g_3	\$10	10	\$100
g_4	\$20	10	\$200
g_5	\$30	10	\$300
g_6	\$40	10	\$400

Clearly those blocks with grade g_1 are not contributing their fair share of the capital investment and profit. Thus the cutoff grade should be g_2 rather than g_1. The number of ore blocks would drop from 60 to 50. If the plant size is not changed, this translates into a shorter property life which changes the cash flow which in turn would affect the \$10 average block assessment. If the mine life is maintained constant, then the size of the mill plant should be reduced. This also affects the cash flow. For this situation, perhaps the average assessment per block would drop to \$5/block. The number of blocks would increase again and the process would be repeated.

One way of handling this problem early on is to include a cost item for capital and profit amongst those used to generate the economic block model. These costs would then be covered up front.

The alternative sequence of steps is listed below (Halls et al., 1969):

1. The metal content of a block, together with forecasted sales prices and estimated full-scale plant metallurgical recoveries is used for determining the revenue.
2. Production costs through to sales and shipping costs are estimated.
3. The amount of depreciation to be added to the operating costs is calculated on a straight line basis by dividing the estimated capital expenditure by the estimated ore reserve tonnage.
4. A minimum profit after tax, related to minimum acceptable return on investment is assigned to each block.
5. The net value for each block is calculated. The 'cutoff grade' used to separate ore from waste is that grade of material which produces a revenue equal to the production costs through to sales together with depreciation and minimum profit.
6. Using the above data the computer will include all blocks of material in the ore reserve which are capable of producing more than the minimum pre-determined profit (after taxes) after paying for the stripping costs necessary to uncover them.

There are no real difficulties in assessing the revenues to be produced from a block or the production costs involved in producing its metal for sale, including such considerations as cost variations due to varying haulage distances and rock types. It is, however, extremely difficult to estimate unit depreciation and required minimum profit per ton accurately before the ore reserve tonnage is known because both

– depreciation, and
– required minimum profit

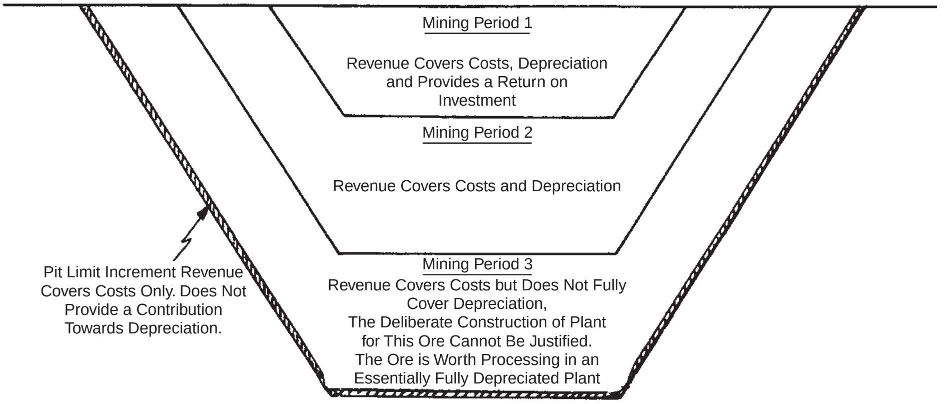


Figure 6.17. Economic pit limits as a function of included costs (Halls et al., 1970).

will vary with plant size and total ore tonnage. In order to estimate depreciation and profit

- an ore reserve tonnage,
- a production rate, and
- an effective tax rate

must be assumed. An iteration process is used until convergence between the assumed and final values is achieved.

The resulting pit is shown as the ‘Mining Period 1’ pit in Figure 6.17. A second pit outline can be produced by eliminating the required profit (profcost) assessed each block. The depreciation cost would remain, however. By eliminating the profit cost element and keeping the other costs and revenues the same, the positive block values would increase. Each positive block would be able to carry more stripping and the pit would increase in size. This is shown by the ‘Mining Period 2’ pit. The ‘no-profit’ condition only applies when determining the pit limit. The average ore grade of the material between the Mining Period 1 and 2 limits would be higher than the pit limit cutoff. Hence a profit is realized. To increase the size of the pit further, the depreciation cost would have to be dropped from the economic block model calculation. This further lowers the grade required to produce positive valued blocks. The pit expands to the limit shown by the heavy shaded line (Mining Period 3 limit). As indicated by Halls (1970), plant capacity cannot be deliberately constructed for this ore. The ore however is worth processing in an essentially fully depreciated plant. Even though a specific profit cost has not been attached to each block in Mining Period 3, there will still be an overall profit since the average ore grade in the period is higher than the cutoff. The Mining Period 2 pit encloses those reserves which should be considered when sizing the plant. The final pit outline would be the same as produced by the first procedure described in this section. Here however the reserves contained within specific mineable pits are used rather than a certain portion of the overall reserves. There are a number of objections (Halls, 1970) which have been raised concerning the requirement that each ore block irrespective of grade contribute an equal portion to profits and capital payback:

(a) The only practical way of determining unit depreciation for use in individual block evaluation is the straight-line method. In the financial evaluation of the ore reserve, the depreciation method employed will be the one which reduces the impact of taxation to a minimum. Also it is virtually impossible to incorporate in the cutoff calculations the unit depreciation which would apply to replacement capital needed throughout the life of the mine.

(b) The use of a minimum after-tax profit for each block does not ensure that the summation of the profits from each block will meet the corporate investment goal (except possibly in the case of an ideal mine, where the annual cash flows remain the same throughout its life).

(c) In the case of properties which can commence mining in a high-grade portion of the pit, or where taxation benefits are allowed in the initial stages of production, the accurate establishment of a profit factor to ensure that such increment of investment yields at least a minimum corporate return is an impossibility.

An alternative procedure based upon incremental financial analysis will now be discussed.

6.6.2 *Incremental financial analysis principles*

The incremental financial analysis approach described in this section is based upon material originally presented by Halls et al. (1969). The rate of return on the total investment is important for assessing the potential of a property as a whole. It does not however indicate the profitability of each capital increment. Only by considering a series of potential pit expansions and evaluating the yields and returns from each is it possible to ascertain that every increment of capital can pay its way. This evaluation process is termed incremental financial analysis. For a new property, the financial returns of alternative ore inventories and associated plant sizes for a given mine life (for example) are compared. One begins from the smallest tonnage ore inventory and plant size. Each progressively larger tonnage ore inventory is considered as a possible expansion. The additional cash flow developed from each expansion is determined as a rate of return on the additional (incremental) capital required for such an expansion. The largest tonnage and hence the lowest grade ore inventory in which every increment of invested capital yields at least the minimum desired corporate return is the optimum ore reserve for the assumed life. The process is repeated using other realistic lives for the property. These studies provide one basis for the selection of a depreciation life (depreciation) for the property. As discussed earlier, the actual mine life may be longer than this since once the invested capital has been fully depreciated, the cutoff grade drops (and the reserves increase). This method of optimizing ore reserves and plant size for a new property involves three steps:

Step 1. Calculate ore inventories using arbitrary cutoff grades.

Step 2. Prepare a financial analysis for each ore inventory.

Step 3. Select that ore inventory which is the optimum ore reserve by incremental financial analysis.

Each of these steps is outlined below.

In step 1, a series of arbitrarily chosen, but reasonable, cutoff grades are chosen. For an open pit copper mine cutoff grades in the range 0.3 to 0.7 percent equivalent copper might be chosen. Using the expected production costs and recoveries, the price needed for breakeven with the selected cutoff (0.7, for example) would be calculated. Note that no capital costs and profit is included in these calculations. The price together with the costs are then used to obtain an economic block model. A technique to provide a pit, such as the floating cone,

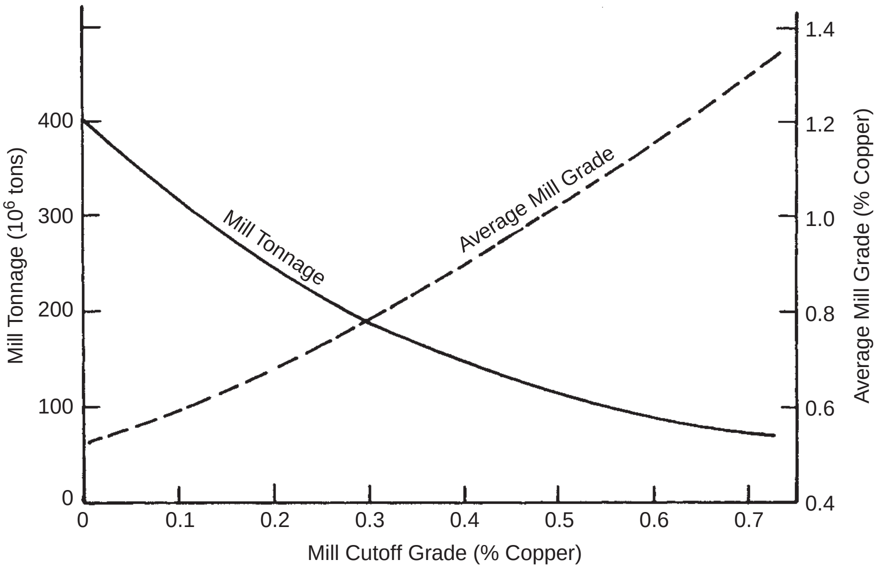


Figure 6.18. Pit material inventory for a 0.7 cutoff grade pit.

Table 6.15. Mill tonnage and average grade as a function of mill cutoff grade.

Mill cutoff grade (% Cu)	Mill tons ($\times 10^6$ st)	Avg. grade (% Cu)
0.70	72	1.29
0.60	88	1.17
0.50	116	1.02
0.40	148	0.90
0.30	188	0.78
0	400	0.52

is applied. This would be the 0.7% cutoff grade pit. The inventory of material contained would be created. One such example is shown in Figure 6.18.

The total tonnage contained is

$$T = 400 \times 10^6 \text{ tons}$$

and the average grade is

$$g = 0.52\% \text{ Cu}$$

If the mill cutoff were 0% Cu then this entire amount would be processed. For various mill cutoff grades, the total tons and grades are as given in Table 6.15.

All material has to be removed from the pit and the costs of depositing it on surface, whether waste or ore are virtually the same. Therefore some material which is below the mining cutoff grade, but within the confines of the pit may be processed at a profit.

Another mining cutoff grade would now be selected, for example 0.6%. The breakeven price would be calculated and the floating cone run on the deposit again using this price.

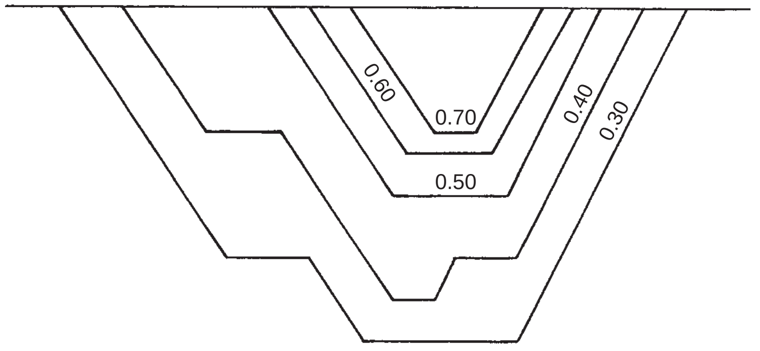


Figure 6.19. Pit outlines corresponding to different mining cutoff grades.

A table similar to Table 6.15 would be constructed for this condition. An idealized cross section showing the series of pits which might be produced is shown in Figure 6.19.

In step 2 a financial evaluation of each ore inventory is prepared. To do this the tonnage and grade to be mined from the inventory each year must be decided. Lane (1964) has discussed one way of doing this. His approach will be the topic of Section 6.7. Here the same life, 25 years, will be used to evaluate all inventories. In practice several realistic lives might be tried. Furthermore the production rate will be held constant. Obviously other variations can be applied. The following data are required to complete the financial evaluations:

(1) the estimated capital costs, broken down into the years in which the money will be spent (including replacement capital),

(2) the estimated operating costs for each production rate,

(3) the metal recoveries, based on metallurgical laboratory testing after discounting for commercial operating conditions,

(4) the long- and short-term estimates of metal sales prices,

(5) the legally applicable depletion and depreciation allowances which will minimize taxation, and

(6) the current and forecasted taxation rates.

The cash flows for each ore inventory are calculated on a year-by-year basis. Any one of a number of financial techniques which consider the time value of money can be utilized in determining the rates of return for both total investment and incremental investment. The discounted cash flow (DCF) method will be used here.

In step 3, the different pit expansions are compared. To illustrate the process, consider the pit generated with the 0.7% Cu cutoff. Two mill cutoffs are considered: 0.7% and 0.6%. The capital investments for mine and concentrator are:

Mill cutoff	Capital cost
0.7	$43,500,000
0.6	$46,832,000

The incremental capital cost is $3,332,000. The annual cash flow for each after taxes is:

Mill cutoff	Annual cash flow
0.7	$7,550,000
0.6	$8,323,000

The incremental annual cash flow is \$773,000. These cash flows occur over a period of 25 years. The net present value of these incremental cash flows is given by

$$\mathrm{NPV_{CF}} = \$773{,}000 \left[\frac{(1+i)^{25} - 1}{i(1+i)^{25}} \right]$$

where i is the interest rate.

These are achieved through the incremental capital investment of \$3,332,000.

$$\mathrm{NPV_{CI}} = -\$3{,}332{,}000$$

The interest rate (i) which makes the sum of these equal to zero is called the rate of return. In this case

$$-3{,}332{,}000 + 773{,}000 \left[\frac{(1+i)^{25} - 1}{i(1+i)^{25}} \right] = 0$$

$$i = 0.231$$

Thus there is a 23.1% effective rate of return (ROR) on the incremental investment. Since this exceeds the company's desired rate of 12%, the mill cutoff grade of 0.6% would become that used in future comparisons (0.5%, 0.4%, etc.) for the given mining cutoff (0.7%). This process would continue until ROR values are less than the base.

6.6.3 *Plant sizing example*

The incremental financial analysis process described in the previous section will now be demonstrated step-by-step using an example adapted from Halls et al. (1969).

1. Prices and costs are estimated. These are entered into the computer for a floating cone analysis or are used to generate a stripping ratio – grade curve for a hand analysis.

This is a breakeven analysis such that

Revenue = Costs

The revenue is calculated by

$$\text{Revenue (\$/ton)} = 2000\,\text{lbs/st} \times \frac{\text{Recovery (\%)}}{100} \times \frac{\text{Grade (\%)}}{100} \times \text{Price(\$/lb)} \tag{6.3}$$

The costs are given by

$$\text{Costs (\$/ton)} = \text{Mining (\$/st)} + \text{Milling (\$/st)} + \text{G\&A (\$/st)} + 2000\,\text{lbs/st} \times \frac{\text{Recovery (\%)}}{100} \times \frac{\text{Grade (\%)}}{100} \times \text{SRS (\$/lb)} \tag{6.4}$$

where SRS is refining, smelting, selling cost (\$/lb).

Setting Equation (6.3) equal to (6.4) yields

$$\text{Mining (\$/ton)} + \text{Milling (\$/ton)} + \text{G\&A (\$/ton)} = (\text{Price} - \text{SRS}) \times \frac{2000\,\text{lbs}}{\text{st}} \times \frac{\text{Recovery (\%)}}{100} \times \frac{\text{Grade (\%)}}{100} \tag{6.5}$$

Solving for the cutoff grade one finds that

$$\text{Grade (\%)} = \frac{[\text{Mining (\$/st)} + \text{Milling (\$/st)} + \text{G\&A (\$/st)}]10^4}{[\text{Price (\$/lb)} - \text{SRS (\$/lb)}]2000 \times \text{Recovery (\%)}} \tag{6.6}$$

If the expected price is used then the maximum (ultimate) pit will result.

2. The grades and tonnage in the pit are determined. This becomes the reserve of material to be mined.

Assume the following values:

Mining cost = \$0.45/st
Milling + G&A = \$1.25/st
Price = \$0.40/lb
SRS = \$0.059/lb
Recovery = 80%

(Note that for simplicity the G&A has been applied only to the mill (i.e. the ore).)

Then the mining cutoff grade becomes

Grade (%) = 0.31%

One would generate a pit with this as the cutoff.

Table 6.16 shows that for a pit based upon a 0.3% Cu pit limit cutoff, the tonnage would be 1,410,000,000. The average grade would be 0.31% Cu (all material). If the mill cutoff grade were 0% then the entire amount would be sent to the mill. If only those blocks having a grade of +0.3% were sent to the mill, then the mill tonnage would be 390,000,000 having an average grade of 0.73% Cu. This would be the largest pit.

3. One could consider smaller pits as well since the biggest pit would not necessarily lead to the desired financial result. This result could be measured in NPV, total profit, return on investment, etc. Other pits can be generated corresponding to different pit limit cutoff grades. As can be seen from Equation (6.6), this is a simple matter of just changing the price. For a lower price, the cutoff grade must go up. In Table 6.16 the following results were obtained.

Pit limit cutoff grade (% Cu)	Total tons in pit ($\times 10^6$ st)	Avg. grade (% Cu)
0.3	1410	0.31
0.4	1206	0.33
0.5	875	0.36
0.6	650	0.38
0.7	400	0.40

For each of these ultimate pits all the material would be mined. One must decide whether it should go to the dump or to the mill.

4. Each of the potential pits is examined with respect to mill and dump material. Consider the pit developed assuming a final pit cutoff grade of 0.7%. The mill cutoff could be set at

Table 6.16. Summary of ore inventory for each mining plan (Halls et al., 1969).

Milling cutoff grade		Pit limit cutoff grade				
		0.7% Cu	0.6% Cu	0.5% Cu	0.4% Cu	0.3% Cu
0.7% Cu	Accum. tons	80,000,000				
	Average grade	1.20% Cu				
	Stripping ratio	4.00 : 1				
0.6% Cu	Accum. tons	92,000,000	135,000,000			
	Average grade	1.13% Cu	1.12% Cu			
	Stripping ratio	3.35 : 1	3.81 : 1			
0.5% Cu	Accum. tons	105,000,000	160,000,000	195,000,000		
	Average grade	1.06% Cu	1.03% Cu	0.99% Cu		
	Stripping ratio	2.81 : 1	3.06 : 1	3.49 : 1		
0.4% Cu	Accum. tons	115,000,000	185,000,000	235,000,000	290,000,000	
	Average grade	1.01% Cu	0.95% Cu	0.90% Cu	0.85% Cu	
	Stripping ratio	2.48 : 1	2.51 : 1	2.73 : 1	3.06 : 1	
0.3% Cu	Accum. tons	130,000,000	205,000,000	265,000,000	330,000,000	390,000,000
	Average grade	0.93% Cu	0.89% Cu	0.84% Cu	0.79% Cu	0.73% Cu
	Stripping ratio	2.08 : 1	2.17 : 1	2.30 : 1	2.51 : 1	2.62 : 1
0.0% Cu	Accum. tons	400,000,000	650,000,000	875,000,000	1,206,000,000	1,410,000,000
	Average grade	0.40% Cu	0.38% Cu	0.36% Cu	0.33% Cu	0.31% Cu
	Stripping ratio	0.0 : 1	0.0 : 1	0.0 : 1	0.0 : 1	0.0 : 1

0.3%, 0.4%, 0.5%, 0.6% or 0.7%. The amount of material to be milled and the average mill grade for each of these scenarios is given below.

Scenario	Milled tons (10^6)	Avg. grade (% Cu)	SR
7-7	80	1.20	4:1
7-6	92	1.13	3.35:1
7-5	105	1.06	2.81:1
7-4	115	1.01	2.48:1
7-3	130	0.93	2.08:1

The total amount of material to be moved is 400×10^6 tons and hence an overall stripping ratio SR can be calculated.

5. A mine life can be assumed. In this case 25 years is chosen. The mine/mill will work 300 days/year. Therefore the daily mining and milling rates can be calculated. For the scenario given above (scenario 7-7) one finds that:

$$\text{Milling rate } = \frac{80 \times 10^6}{25 \times 300} = 10{,}667\,\text{tpd}$$

$$\text{Ore mining rate} = \frac{80 \times 10^6}{25 \times 300} = 10{,}667\,\text{tpd}$$

$$\text{Stripping rate} = \frac{320 \times 10^6}{25 \times 300} = 42{,}667\,\text{tpd}$$

$$\text{Overall mining rate } = 53{,}334\,\text{tpd}$$

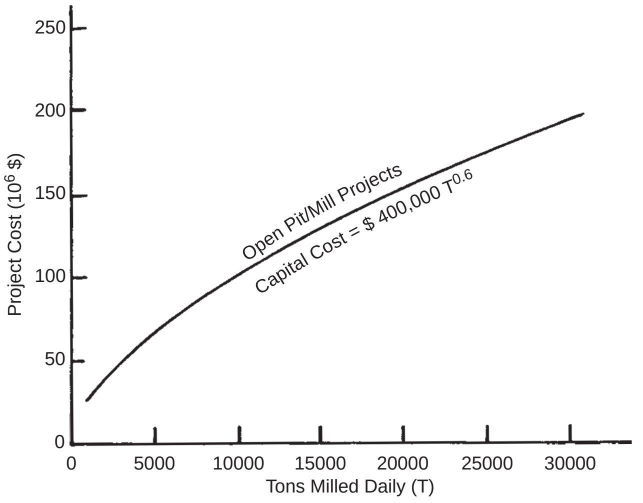


Figure 6.20. Open pit/mill project costs as a function of milling rate (O'Hara, 1980).

The results for pit 7 are given in the table below.

Scenario	Milling rate (tpd)	Mining rate (tpd)
7-7	10,667	53,334
7-6	12,267	53,334
7-5	14,000	53,334
7-4	15,333	53,334
7-3	17,333	53,334

6. Each of the different mining and milling rates has a certain associated capital cost. O'Hara (1980) has presented a curve, Figure 6.20, which could be used for estimating. In the present case (Halls et al., 1969), the costs which were used are:

Scenario	Capital cost (10^3\$)
7-7	\$43,500
7-6	46,832
7-5	50,166
7-4	52,832
7-3	56,832

7. The mining, milling, and G&A operating costs can also be estimated using the curves developed by O'Hara (1980). In this case the following apply:

Ore mining cost = \$0.45/st
Stripping cost = 0.45/st
Milling + G&A = \$1.25/st
Mill recovery = 80%

The smelting, refining and sales cost are estimated at 5.9¢/lb.

8. One can calculate the gross revenues for the various scenario's. A price of 40¢/lb is used.

$$\text{Gross revenue (\$/yr)} = \text{Recovered copper} \times \text{Price}$$
$$= \text{Milling rate (t/day)} \times \text{days/yr} \times \frac{\text{Grade(\%)}}{100} \times 2000 \times \text{Price (\$/lb)}$$

For scenario 7-4:

$$\text{Gross revenue} = 15,333 \times 300 \times \frac{1.01}{100} \times 2000 \times 0.40 = \$29{,}734{,}000$$
$$\text{Recovered copper} = 74{,}334{,}384\,\text{lbs} = 37{,}167\,\text{tons}$$

9. The operating costs can be similarly computed. Scenario 7-4 will be continued.

$$\text{Mining cost (ore + waste)} = 0.45 \times 53{,}334 \times 300 = \$7{,}200{,}000$$
$$\text{Milling + G\&A} = 1.25 \times 15{,}333 \times 300 = \$5{,}750{,}000$$
$$\text{SRS} = 0.059 \times 74{,}334{,}384 = \$4{,}386{,}000$$

10. Straight line depreciation will be used.

$$\text{Depreciation (\$/yr) for scenario 7-4} = \frac{\$52{,}832{,}000}{25\,\text{yrs}}$$
$$\text{Depreciation (\$/yr)} = \$2{,}113{,}280$$

11. Percentage depletion is computed for copper as 15% of the gross revenues. It cannot exceed 50% of the taxable income.

12. Taxes are computed at a rate of 52%.

13. A cash flow analysis can now be made of the various scenarios. This will be illustrated with scenario 7-4.

Cash flow analysis for scenario 7-4

1. Gross revenues		= \$29,734,000
2. Operating expenses		= \$17,336,000
Mining	= 7,200,000	
Milling	= 5,750,000	
SRS	= 4,386,000	
	= 17,336,000	
3. Net revenue		= \$12,398,000
4. Depreciation		= \$2,113,380
5. Taxable income before depletion		= \$10,284,620
6. Depletion		= \$4,460,100
15% of gross revenues	= \$4,460,100	
50% of taxable income	= \$5,142,310	
(choose 15% of gross revenue since this is smaller)		

7. Taxable income after depletion = $5,824,520
8. Taxes @52% = $3,028,750
9. Annual net income after taxes (line 5 minus line 8) = $7,255,870
10. Cash flow after taxes (line 9 plus line 4) = $9,369,250

(This cash flow analysis is similar to that done by Halls in Table 6.17.) This cash flow analysis has been done for all of the scenarios.

14. Calculate the internal ROR. It is assumed that there are 25 uniform positive cash flows (in years 1 through 25) which will be compared with the negative cash flow in year zero (capital expense). The interest rate which makes the sum (NPV) zero is to be determined. This is the internal ROR. For scenario 7-4 the situation is

$$-\$52{,}832{,}000 + \$9{,}353{,}000\left[\frac{(1+i)^{25}-1}{i(1+i)^{25}}\right] = 0$$

In this case the value of Halls ($9,353,000) rather than $9,369,250 has been used.

Solving for i, one finds that

$$i = 0.174$$

Therefore the rate of return on the invested capital is 17.4%.

The results of this calculation for all of the pit 7 scenarios are summarized below:

Scenario	Capital investment ($\$10^3$)	Annual cash flow after taxes ($\$10^3$)	Internal ROR (%)
7-7	43,500	7550	17.0
7-6	46,832	8323	17.5
7-5	50,166	8951	17.6
7-4	52,832	9353	17.4
7-3	56,832	9727	16.8

If the company's minimum desired ROR is 12% then all of these meet the criterion. The one chosen is scenario 7-3 due to the fact that the total profits would be highest.

15. An incremental financial analysis is now performed on these data. As can be seen the increase in the annual cash flows in going from scenario 7-7 to 7-3 is due to an increase in the capital investment. The question is 'what is the rate of return on the increment of capital needed to go from one scenario to the next?' These increments in capital and annual cash flows are summarized in Table 6.18. One now goes through the calculation of determining the rate of return on the invested incremental capital.

$$\Delta\text{Capital} = \Delta\text{CF}\left[\frac{(1+i)^{25}-1}{i(1+i)^{25}}\right]$$

Examining the change between scenarios 7-7 and 7-6 one finds that

$$3{,}332{,}000 = 773{,}000\left[\frac{(1+i)^{25}-1}{i(1+i)^{25}}\right]$$

$$i = 23.1\%$$

Table 6.17. Summary of evaluations for ABC mine (Halls et al., 1969).

	Scenario designation								
	7-7	7-6	7-5	7-4	7-3	6-6	6-5	6-4	6-3
Ore inventories									
Tons (10^6)	80	92	105	115	130	135	180	185	205
Grade (% Cu)	1.20	1.13	1.06	1.0	0.83	1.12	1.03	0.95	0.89
Stripping ratio	4.00:1	3.35:1	2.81:1	2.48:1	2.08:1	3.81:1	3.06:1	2.51:1	2.17:1
Concentrator capacity for 25-year life (tons/day)	10,700	12,300	14,000	15,300	17,300	18,000	21,300	24,700	27,300
Capital investment for mine and concentrator (10^3\$)	\$43,600	\$48,832	\$50,166	\$52,832	\$56,832	\$62,500	\$69,166	\$75,834	\$81,166
Copper production									
Annual tons (refined)	30,720	33,268	35,616	37,168	38,688	48,384	52,736	56,240	58,384
Total tons (refined)	768,000	831,700	890,400	929,200	967,200	1,209,600	1,318,400	1,408,000	1,544,400
Annual net income after taxes	\$5810	\$6462	\$8945	\$7239	\$7455	\$9397	\$10,320	\$10,960	\$11,320
Annual cash flow after taxes	\$7530	\$8323	\$8951	\$9363	\$9727	\$11,897	\$13,086	\$13,993	\$14,567
Financial analysis of investment									
Return on total investment-DCF	17.0%	17.5%	17.6%	17.4%	16.8%	18.8%	18.8%	18.2%	17.7%
Return on incremental investment of this									
Scenario compared to	–	7-7	7-6	7-6	7-4	7-4	6-6	6-5	6-4
Incremental investment required (10^3\$)	–	\$3332	\$3334	\$2666	\$4000	\$9666	\$8888	\$6668	\$5232
Incremental cash flow (10^3\$)	–	\$773	\$628	\$4092	\$374	\$2,544	\$1,189	\$907	\$574
DCF return on incremental investment	–	23.1%	18.6%	14.6%	8.9%	26.4%	17.6%	13.0%	9.4%
Extrapolated cutoff grade to provide 12% return on incremental investment	Economic cutoff for 0.7% Cu pit = 0.36% Cu					Economic cutoff for 0.6% Cu pit = 0.37% Cu			

Table 6.17. (Continued).

	Scenario designation					
	5-5	5-4	5-3	4-4	4-3	3-3
Ore reserves						
Tons (10^6)	195	235	265	290	330	390
Grade (% Cu)	0.99	0.9	0.84	0.84	0.79	0.73
Stripping ratio	3.49:1	2.73:1	2.30:1	3.16:1	2.65:1	2.62:1
Concentrator capacity for 25-year life (tons/day)	26,000	31,000	35,300	38,700	44,000	52,000
Capital investment for mine and concentrator (10^3\$)	\$80,000	\$90,667	\$98,667	\$103,000	\$112,666	\$120,000
Copper production						
Annual tons (refined)	61,776	67,680	71,200	78,880	83,424	91,104
Total tons (refined)	1,544,400	1,692,000	1,780,000	1,972,000	2,085,600	2,277,600
Annual next income after taxes (10^3\$)	\$11,752	\$12,763	\$13,779	\$13,728	\$14,431	\$14,812
Annual cash flow after taxes (10^3\$)	\$14,752	\$16,390	\$17,327	\$17,808	\$18,938	\$19,612
Financial analysis of investment						
Return on total investment-DCF	17.9%	17.8%	17.3%	17.0%	16.4%	15.9%
Return on incremental investment of this						
Scenario compared to	6-4	5-5	5-4	5-4	5-4	5-4
Incremental investment required (10^3\$)	\$4166	\$10,667	\$8,000	\$12,333	\$21,999	\$29,333
Incremental cash flow (10^3\$)	\$759	\$1638	\$937	\$1418	\$2548	\$3222
DCF return on incremental investment	17.9%	15.3%	10.2%	10.6%	10.7%	10.0%
Extrapolated cutoff grade to provide 12% return on incremental investment	Economic cutoff for 0.5% Cu pit = 0.34% Cu					

The scenario designations refer to the cutoff grades for the pit limits and the concentrator (i.e. scenario 5-4 means an ultimate pit cutoff grade of 0.5% Cu and concentrator cutoff grade of 0.4% Cu). All dollar values are recorded in thousands.

Table 6.18. Incremental capital investments and annual cash flows.

Scenario	ΔCapital investment ($$10^3$)	ΔCash flow ($$10^3$)
7-7		
	3332	773
7-6		
	3334	628
7-5		
	2666	402
7-4		
	4000	374
7-3		

Repeating this process for the others one finds

Scenario	*i*
7-7 $\rightarrow$ 7-6	23.1%
7-6 $\rightarrow$ 7-5	18.6%
7-5 $\rightarrow$ 7-4	14.6%
7-4 $\rightarrow$ 7-3	8.9%

Now it can be seen that the optimum scenario is 7-4 rather than 7-3. The milling rate would be 15,300 tpd.

16. Examine all of the other scenarios with respect to the best current one. For pit 7, the best scenario is 7-4. For pit 6, scenario 6-6 has a higher total ore tonnage than 7-4 with an acceptable rate of return on the additional capital expenditure. It thus becomes the best current alternative. Plan 6-5 is better than 6-6 and 6-4 is better than 6-5. Plan 6-3 is not acceptable. Pit 5 alternatives are compared against 6-4. As can be seen from the table, the overall best plan is between Plan 5-4 and Plan 5-3. The economic cutoff (mill cutoff) for the pit is 0.34% Cu.

This leads to the following operation

Ore reserve = 253,000,000 tons
Mill rate = 33,580 tpd
Mining rate = 116,600 tpd
Average ore grade = 0.864% Cu
Mine life = 25 years
Capital investment = $95,500,000

6.7 LANE'S ALGORITHM

6.7.1 *Introduction*

In 1964, K.F. Lane (Lane, 1964) presented what has become a classic paper entitled 'Choosing the Optimum Cut-off Grade'. This section will describe his approach and illustrate it with an example. As has been discussed earlier cutoff grade is the criterion normally used in mining to discriminate between ore and waste in the body of a deposit. Waste may either be left in place or sent to waste dumps. Ore is sent to the treatment plant for further processing and eventual sale.

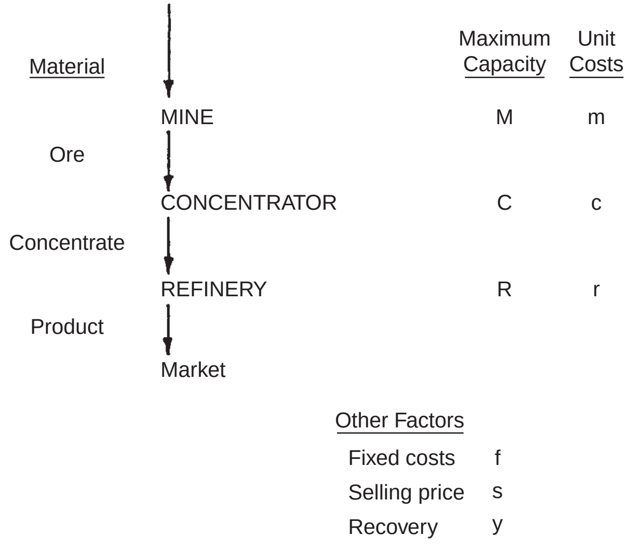


Figure 6.21. The model described by the Lane algorithm (Lane, 1964).

The choice of cutoff grade can directly affect profits. This chapter will examine the principles which determine the best choice of a cutoff grade under different circumstances.

A mining operation is considered to consist of three stages:

– mining,
– concentrating, and
– refining.

Each stage has its own associated costs and a limiting capacity. The operation as a whole will incur continuing fixed costs. The three most important economic criteria which can be applied are:

Case I: Maximum present value.
Case II: Maximum total profits.
Case III: Maximum immediate profit.

The maximum present value gives the economic optimum and is that generally applied lacking special circumstances. It is the one which will be used in this book. As has been shown by Lane (1964) the second and third correspond to the application of special discount rates in the first. Case II, maximum total profits, corresponds to a discount rate of zero percent whereas Case III is for a high value.

In this chapter, attention is focussed on choosing a cutoff grade to maximize the present value of the cash flow from the operation.

6.7.2 *Model definition*

Figure 6.21 is a diagrammatic representation of the elements and symbols used in the model.

Definitions of the maximum capacities, unit costs and quantities involved in the evaluation are presented below.

1. Maximum capacity: M is the maximum amount of material (ore and waste) that the mine can produce in a given time period (for example 1000 tons/year). It is therefore a restriction on the maximum rate of progress through the orebody.

C is the maximum amount of ore which can be put through the concentrator in a given time period (for example 500 tons/year), assuming unrestricted availability of input ore from the mine. A concentrate of fixed grade is produced.

R is the maximum amount of final product produced in the time period (for example 500 lbs/year), assuming unrestricted availability of concentrate from the concentrator. The maximum can be due to a restriction on refinery throughput or a market limitation.

2. Costs: m are the mining costs expressed in \$/ton of material moved. These are assumed to be the same irrespective as to whether the material is classified as ore or waste. The unit mining costs include drilling, blasting, loading, hauling, etc.

c are the concentrating costs expressed in \$/ton of material milled. The unit cost c includes crushing, grinding, floating, leaching, etc. It also includes some haulage if ore is trucked farther than waste (if not, this can become a credit item in calculating c).

r includes all costs incurred at the product and selling stages such as smelting, refining, packaging, freight, insurance, etc. These are expressed in terms of \$ per unit of product. For copper it would be \$/lb.

f, the fixed cost, includes all costs such as rent, administration, maintenance of roads and buildings, etc. which are independent of production levels (within normal limits of variation) but which would cease were the mine to be closed. It is expressed in terms of a fixed cost over the production period considered (for example 1 year). Other costs such as head office charges, depreciation, etc. are not included.

s, the selling price, is expressed in terms of selling price per unit of product. It is a gross figure provided all selling charges are included in r. If not they must be subtracted from s.

y, the recovery, is an overall figure for the concentrator and the refinery. It is that proportion of the mineral contained in the original ore feed retained in the final product.

3. Quantities: T is the length of the production period being considered (for example 1 year); Q_m is the quantity of material to be mined, Q_c is the quantity of ore sent to the concentrator and Q_r is the amount of product actually produced over this production period.

6.7.3 *The basic equations*

Using the definitions given in the preceding section, the basic equations can be developed. The total costs T_c are

$$T_c = mQ_m + cQ_c + rQ_r + fT \tag{6.7}$$

Since the revenue R is

$$R = sQ_r \tag{6.8}$$

the profit P is given by

$$P = R - T_c = sQ_r - (mQ_m + cQ_c + rQ_r + fT) \tag{6.9}$$

Combining terms, yields

$$P = (s - r)Q_r - cQ_c - mQ_m - fT \tag{6.10}$$

This is the basic profit expression. It can be used to calculate the profit from the next Q_m of material mined.

Table 6.19. Initial mineral inventory for the Lane example.

Grade (lbs/ton)	Quantity (tons)
0.0–0.1	100
0.1–0.2	100
0.2–0.3	100
0.3–0.4	100
0.4–0.5	100
0.5–0.6	100
0.6–0.7	100
0.7–0.8	100
0.8–0.9	100
0.9–1.0	100
	Q_m = 1000

6.7.4 *An illustrative example*

To introduce the reader in a soft way to the problem being explored in detail in this section consider the following example. A final pit has been superimposed on a mineral inventory. Within the pit outline are contained 1000 tons of material. The grade distribution is shown in Table 6.19. The associated costs, price, capacities, quantities and recovery are:

Costs

m = mining = \$1/ton
c = concentrating = \$2/ton
r = refining = \$5/lb
f = fixed cost = \$300/year

Price

s = \$25/lb

Capacities

M = 100 tons/year
C = 50 tons/year
R = 40 lbs/year

Quantities

Q_m = amount to be mined (tons)
Q_c = amount sent to the concentrator (tons)
Q_r = amount of concentrator product sent for refining (lbs)

Recovery(Yield)

y = 1.0(100%recovery is assumed).

There are a great number of possible mine, concentrator and refinery operating combinations. Which is the optimum? In this section the basic equations will be developed in addition to demonstrating the process. However, prior to beginning the theoretical treatment, it is considered useful to briefly consider just one of these operating combinations. The total amount of material to be mined Q_m is 1000 tons. If the mine is operated at capacity (100 tons/year) then the pit would be mined out over a time period of 10 years. Assuming that the grades (Table 6.19) are equally distributed throughout the pit and a concentrator cutoff grade of 0.50 lbs/ton is used, then 50 tons of material having an average grade of 0.75 lbs/ton would be sent to the concentrator every year. The other 50 tons would be sent to the waste dump. Since the concentrator capacity C is 50 tons/year, this is an acceptable situation. The concentrator product becomes the refinery feed. In this case it would be 37.5 lbs/year (0.75 lbs/ton × 50 tons/year). Since the refinery can handle 40 lbs/year, it would be operating at below its rated capacity R. This combination of mining rate and cutoff grade would yield a yearly profit P_y of

$$P_y = (25 - 5)37.5 - 2 \times 50 - 1 \times 100 - 300 = \$250$$

These profits would continue for 10 years and hence the total profit would be \$2500. The NPV assuming an interest rate of 15% would be

$$\text{NPV} = \frac{250[(1.15)^{10} - 1]}{0.15(1.15)^{10}} = \$1,254.69$$

The first question to be asked is whether some other combination of mine production rate and concentrator cutoff grade would yield a better profit from this deposit? The larger question is whether the various plant capacities (with their associated costs) are optimum? The procedure described in this section is a way of determining the combination yielding the maximum profit for a given set of operating constraints. The constraints may then be changed (mine, concentrator and refining capacities, for example) and the profit corresponding to this new combination determined as well as how the various capacities should be utilized over the life of the pit.

6.7.5 *Cutoff grade for maximum profit*

Step 1. Determination of the economic cutoff grade – one operation constraining the total capacity

As indicated, the basic profit expression (6.10) is

$$P = (s - r)Q_r - cQ_c - mQ_m - fT$$

Calculate cutoff grade assuming that the mining rate is the governing constraint. If the mining capacity M is the applicable constraint, then the time needed to mine material Q_m is

$$T_m = \frac{Q_m}{M} \tag{6.11}$$

Equation (6.10) becomes

$$P = (s - r)Q_r - cQ_c - \left(m + \frac{f}{M}\right)Q_m \tag{6.12}$$

To find the grade which maximizes the profit under this constraint one first takes the derivative of (6.12) with respect to g.

$$\frac{dP}{dg} = (s - r)\frac{dQ_r}{dg} - c\frac{dQ_c}{dg} - \left(m + \frac{f}{M}\right)\frac{dQ_m}{dg} \tag{6.13}$$

However the quantity to be mined is independent of the grade, hence

$$\frac{\mathrm{d}Q_m}{\mathrm{d}g} = 0 \tag{6.14}$$

Equation (6.13) becomes

$$\frac{\mathrm{d}P}{\mathrm{d}g} = (s - r)\frac{\mathrm{d}Q_r}{\mathrm{d}g} - c\frac{\mathrm{d}Q_c}{\mathrm{d}g} \tag{6.15}$$

The quantity refined Q_r is related to that sent by the mine for concentration Q_c by

$$Q_r = \bar{g}yQ_c \tag{6.16}$$

where $\bar{g}$ is the average grade sent for concentration, and y is the recovery.

Taking the derivative of Q_r with respect to grade one finds that

$$\frac{\mathrm{d}Q_r}{\mathrm{d}g} = \bar{g}y\frac{\mathrm{d}Q_c}{\mathrm{d}g} \tag{6.17}$$

Substituting Equation (6.17) into (6.15) yields

$$\frac{\mathrm{d}P}{\mathrm{d}g} = \left[(s - r)\bar{g}y - c\right]\frac{\mathrm{d}Q_c}{\mathrm{d}g} \tag{6.18}$$

The lowest acceptable value of $\bar{g}$ is that which makes

$$\frac{\mathrm{d}P}{\mathrm{d}g} = 0$$

Thus the cutoff grade g_m based upon mining constraints is the value of $\bar{g}$ which makes

$$(s - r)\bar{g}y - c = 0$$

Thus

$$g_m = \bar{g} = \frac{c}{y(s - r)} \tag{6.19}$$

Substituting the values from the example yields

$$g_m = \frac{\$2}{1.0(\$25 - \$5)} = 0.10\,\text{lbs/ton}$$

Calculate cutoff grade assuming that the concentrating rate is the governing constraint. If the concentrator capacity C is the controlling factor in the system, then the time required to mine and process a Q_c block of material (considering that mining continues simultaneously with processing) is

$$T_c = \frac{Q_c}{C} \tag{6.20}$$

Substituting Equation (6.20) into (6.10) gives

$$P = (s - r)Q_r - cQ_c - mQ_m - f\frac{Q_c}{C} \tag{6.21}$$

Rearranging terms one finds that

$$P = (s - r)Q_r - \left(c + \frac{f}{C}\right)Q_c - mQ_m$$

Differentiating with respect to g and setting the result equal to zero yields

$$\frac{\mathrm{d}P}{\mathrm{d}g} = (s-r)\frac{\mathrm{d}Q_r}{\mathrm{d}g} - \left(c + \frac{f}{C}\right)\frac{\mathrm{d}Q_c}{\mathrm{d}g} - m\frac{\mathrm{d}Q_m}{\mathrm{d}g} = 0$$

As before

$$\frac{\mathrm{d}Q_m}{\mathrm{d}g} = 0$$

$$\frac{\mathrm{d}Q_r}{\mathrm{d}Q_c} = gy$$

Thus

$$(s-r)\frac{\mathrm{d}Q_r}{\mathrm{d}g} = \left(c + \frac{f}{C}\right)\frac{\mathrm{d}Q_c}{\mathrm{d}g}$$

$$\frac{\mathrm{d}Q_r}{\mathrm{d}Q_c} = \frac{c + \frac{f}{C}}{s-r} = gy$$

The cutoff grade when the concentrator is the constraint is

$$g_c = \frac{c + \frac{f}{C}}{y(s-r)} \tag{6.22}$$

For the example, this becomes

$$g_c = \frac{\$2 + \frac{\$300}{50}}{1.0(\$25 - \$5)} = 0.40$$

Calculate cutoff grade assuming that the refining rate is the governing constraint. If the capacity of the refinery (or the ability to sell the product) is the controlling factor then the time is given by

$$T_r = \frac{Q_r}{R} \tag{6.23}$$

Substituting (6.23) into Equation (6.10) yields

$$P = (s-r)Q_r - \frac{fQ_r}{R} - cQ_c - mQ_m$$

or

$$P = \left(s - r - \frac{f}{R}\right)Q_r - cQ_c - mQ_m \tag{6.24}$$

Differentiating with respect to g and setting the result equal to zero gives

$$\frac{\mathrm{d}P}{\mathrm{d}g} = \left(s - r - \frac{f}{R}\right)\frac{\mathrm{d}Q_r}{\mathrm{d}g} - c\frac{\mathrm{d}Q_c}{\mathrm{d}g} - m\frac{\mathrm{d}Q_m}{\mathrm{d}g} = 0$$

Simplifying and rearranging gives

$$g_r = \frac{c}{\left(s - r - \frac{f}{R}\right)y} \tag{6.25}$$

For this example

$$g_r = \frac{\$2}{\left(\$25 - \$5 - \frac{\$300}{40}\right)1.0} = 0.16 \tag{6.26}$$

One can now calculate the amount of material which would be concentrated and refined under the various constraints as well as the time required. When the mining rate of 100 tons/year is the constraint

$$T_m = \frac{1000\,\text{tons}}{100\,\text{tons/year}} = 10\,\text{years}$$

Since the cutoff grade g_m is 0.10 lbs/ton, a quantity Q_c of 900 tons having an average grade of 0.55 lbs/ton would be sent to the concentrator. The total amount of product refined and sold Q_r is

$$Q_r = 900 \times 0.55 = 495\,\text{lbs}$$

Substituting these values into the profit equation gives

$$P_m = (\$25 - \$5)495 - \$2 \times 900 - \$1 \times 1000 - \$300 \times 10 = \$4100$$

The same procedure can be followed with the other two limiting situations. The results are given below:

Concentrator limit:

$$g_c = 0.40\,\text{lbs/ton}$$

$$Q_c = 0.60 \times 1000 = 600\,\text{tons}$$

$$C = 50\,\text{tons/year}$$

$$T_c = \frac{600}{50} = 12\text{years}$$

$$Q_m = \frac{1000}{12} = 83.3\,\text{tons/year}$$

$$\bar{g} = 0.7\,\text{lbs/tons}$$

$$Q_r = 600 \times 0.7 \times 1.0 = 420\,\text{lbs}$$

$$\begin{aligned} P_c &= (25 - 5)420 - 2 \times 600 - 1 \times 1000 - 300 \times 12 \\ &= \$2600 \end{aligned}$$

Refinery limit:

$$g_r = 0.16\,\text{lbs/ton}$$

$$\bar{g} = 0.58\,\text{lbs/tons}$$

$$Q_c = 0.84 \times 1000 = 840\,\text{tons}$$

$$Q_r = 840 \times 0.58 \times 1.0 = 487.2\,\text{lbs}$$

$$T_r = \frac{487.2}{40} = 12.18\text{years}$$

$$M = \frac{1000}{12.18} = 82.1\,\text{tons/year}$$

$$C = \frac{840}{12.18} = 69\,\text{tons/year}$$

$$\begin{aligned} P_r &= (25 - 5)487.2 - 2 \times 840 - 1 \times 1000 - 300 \times 12.18 \\ &= \$3410 \end{aligned}$$

Table 6.20. Total profits as a function of concentrator cutoff with mine operating at capacity.

g	Profits ($)		
	P_m	P_c	P_r
0.0	4000	1000	3250
0.1	4100	1700	3386
0.16	4064	2024	3410
0.2	4000	2200	3400
0.3	3700	2500	3287.50
0.4	3200	2600	3050
0.5	2500	2500	2687.50
0.6	1600	2200	2200
0.7	500	1700	1587.50
0.8	−800	1000	850
0.9	−2300	100	−12.50
0.95	−3125	−425	−511.25

In summary, for each operation taken as a single constraint, the optimum cutoff grades are:

$$g_m = 0.10$$
$$g_c = 0.40$$
$$g_r = 0.16$$

The total profits assuming the single constraint of mining, concentrating or refining are given as a function of cutoff grade in Table 6.20. The values have been plotted in Figure 6.22.

Step 2. Determination of the economic cutoff grade by balancing the operations

In the first step, it was assumed that only one of the operations was the limiting factor to production capacity.

A second type of cutoff is based simply on material balance. To be able to calculate this one needs to know the distribution of grades of the mined material. The average grade of the treated material can be found as a function of the chosen cutoff. The average grade, and the number of units involved are given as a function of cutoff grade in Table 6.21.

For both the mine and mill to be at their respective capacities, then

$$Q_m = 100 \text{ tons}$$
$$Q_c = 50 \text{ tons}$$

As can be seen from Table 6.21, the cutoff grade should be 0.5 lbs/ton. This balancing cutoff between mine and concentrator is expressed as g_{mc}. For the concentrator and the refinery to be at full capacity

$$Q_c = 50 \text{ tons}$$
$$Q_r = 40 \text{ tons}$$

The relationship between Q_c and Q_r is shown in Table 6.22.

In examining Table 6.22, it can be seen that the required balancing cutoff grade g_{cr} is

$$g_{cr} = 0.60$$

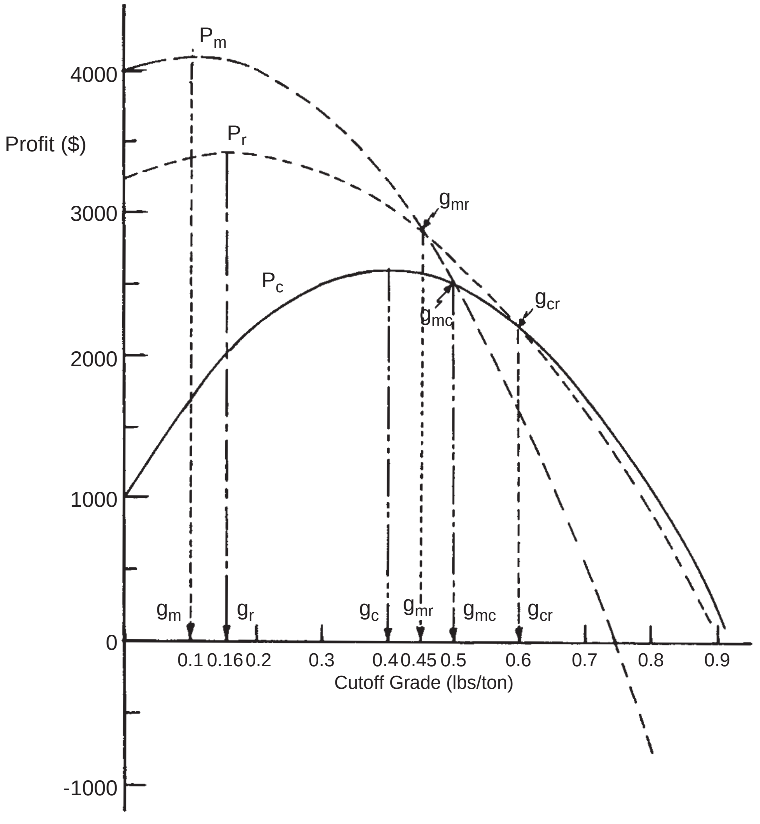


Figure 6.22. Total profit as a function of cutoff grade under different constraints.

Table 6.21. Concentrator feed as a function of concentrator cutoff with mine operating at capacity.

Mined amount (Q_m) (tons)	Concentrator cutoff grade (g_c) (lbs/ton)	Feed going to the concentrator (Q_c)(tons)
100	0	100
100	0.1	90
100	0.2	80
100	0.3	70
100	0.4	60
100	0.5	50
100	0.6	40
100	0.7	30
100	0.8	20
100	0.9	10

The final balancing cutoff is between the mine and the refinery. As seen in Table 6.23 (assuming 100% concentrator recovery) a cutoff grade of 0.4 yields 42 lbs of product whereas 0.5 yields 37.5 lbs. The desired level of 40 lbs lies between these two. Interpolating one finds that

$$g_{mr} \cong 0.456\,\text{lbs/ton}$$

Table 6.22. Refinery product as a function of concentrator cutoff with concentrator operating at capacity.

Amount to be concentrated (Q_c) (tons)	Concentrator cutoff grade (g_c) (lbs/ton)	Avg. conc. feed grade ($\bar{g}_c$) (lb/ton)	Refinery product (Q_r) (lbs)
50	0	0.5	25
50	0.1	0.55	27.5
50	0.2	0.5	30
50	0.3	0.65	32.5
50	0.4	0.7	35
50	0.5	0.75	37.5
50	0.6	0.8	40
50	0.7	0.85	42.5
50	0.8	0.9	45
50	0.9	0.95	47.5

Table 6.23. Refinery feed as a function of mine cutoff with the mine operating at capacity (assuming 100% concentratory recovery).

Mined amount (Q_m) (tons)	Mine cutoff grade (g_m) (lbs/ton)	Refinery product (Q_r) (lbs)
100	0	50
100	0.1	49.5
100	0.2	48
100	0.3	45.5
100	0.4	42
100	0.5	37.5
100	0.6	32
100	0.7	25.5
100	0.8	18
100	0.9	9.5

In summary, when the operations are taken in combination, the optimum cutoff grades are:

$$g_{mc} = 0.50$$

$$g_{cr} = 0.60$$

$$g_{mr} = 0.456$$

Step 3. Determining the overall optimum of the six cutoff grades

There are six possible cutoff grades. Three (g_{mc}, g_{cr}, and g_{mr}) are based simply upon the grade distribution of the mined material and capacities. The other three (g_m, g_c, and g_r) are based upon capacities, costs and the price. The objective is to find the cutoff grade which produces the overall maximum profit in light of the mining, concentrating and refining constraints. The local optimums for each pair of operations are first considered. The corresponding optimum grades for each pair (G_{mc}, G_{rc}, and G_{mr}) are selected using the following rules:

$$G_{mc} = \begin{cases} g_m & \text{if}\, g_{mc} \leq g_m \\ g_c & \text{if}\, g_{mc} \geq g_c \\ g_{mc} & \text{otherwise} \end{cases} \tag{6.26a}$$

$$G_{rc} = \begin{cases} g_r & \text{if} g_{rc} \le g_r \\ g_c & \text{if} g_{rc} \ge g_c \\ g_{rc} & \text{otherwise} \end{cases} \tag{6.26b}$$

$$G_{mr} = \begin{cases} g_m & \text{if} g_{mr} \le g_m \\ g_r & \text{if} g_{mr} \ge g_r \\ g_{mr} & \text{otherwise} \end{cases} \tag{6.26c}$$

The overall optimum cutoff grade G is just the middle value of G_{mc}, G_{mr}, and G_{rc}. In our example the six possible cutoff grades are:

$$\begin{aligned} g_m &= 0.10 \\ g_c &= 0.40 \\ g_r &= 0.16 \\ g_{mc} &= 0.50 \\ g_{mr} &= 0.456 \\ g_{cr} &= 0.60 \end{aligned}$$

Consider them in groups of three:

Group1	$g_m = 0.10$ $g_c = 0.40$ $g_{mc} = 0.50$	Choose the middle value $G_{mc} = 0.40$
Group2	$g_m = 0.10$ $g_r = 0.16$ $g_{mr} = 0.456$	Choose the middle value $G_{mr} = 0.16$
Group3	$g_r = 0.10$ $g_c = 0.40$ $g_{cr} = 0.60$	Choose the middle value $G_{cr} = 0.40$

Considering the three middle values

$$\begin{aligned} G_{mc} &= 0.40 \\ G_{mr} &= 0.16 \\ G_{cr} &= 0.40 \end{aligned}$$

one chooses one numerically in the middle

$$G = 0.40\,\text{lbs/ton}$$

From Table 6.24, the average grade $\bar{g}_c$ of the material sent to the concentrator for a cutoff of 0.40 lbs/ton would be

$$\bar{g}_c = 0.70\,\text{lbs/ton}$$

For 100% recovery the quantities are

$$\begin{aligned} Q_m &= 1000\,\text{tons} \\ Q_c &= 600\,\text{tons} \\ Q_r &= 420\,\text{lbs} \end{aligned}$$

Table 6.24. Grade distribution for the first 100 ton parcel mined.

Grade (lb/ton)	Quantity (tons)
0.0–0.1	10
0.1–0.2	10
0.2–0.3	10
0.3–0.4	10
0.4–0.5	10
0.5–0.6	10
0.6–0.7	10
0.7–0.8	10
0.8–0.9	10
0.9–1.0	10
	$Q_m = 100$

Applying the respective capacities to these quantities one finds that

$$T_c = \frac{600}{50} = 12\,\text{years}$$

$$T_r = \frac{420}{40} = 10.5\text{ years}$$

$$T_m = \frac{1000}{100} = 10\text{ years}$$

Since the concentrator requires the longest time, it controls the production capacity. The total profit is

$$P = \$2600$$

and the profit per year P_y is

$$P_y = \frac{\$2600}{12\text{years}} = \$216.70/\text{year}$$

The net present value of these yearly profits assuming an interest rate of 15% is

$$\text{NPV} = P_y \frac{(1+i)^{12} - 1}{i(1+i)^{12}} = \$216.70 \frac{1.15^{12} - 1}{0.15(1.15)^{12}} = \$1174.60$$

In summary: the concentrator is the controlling production limiter; concentrator feed = 50 tons/year; optimum mining cutoff grade = 0.40 lbs/ton (constant); total concentrator feed = 600 tons; average concentrator feed grade = 0.70 lbs/ton; years of production = 12 years; copper production/year = 35 lbs; total copper produced = 420 lbs; total profits = \$2600.40; net present value = \$1174.60.

6.7.6 *Net present value maximization*

The previous section considered the selection of the cutoff grade with the objective being to maximize profits. In most mining operations today, the objective is to maximize the net present value NPV. In this section the Lane approach to selecting cutoff grades maximizing NPV subject to mining, concentrating and refining constraints will be discussed.

For the example of the previous section a fixed mining cutoff grade of 0.40 lbs/ton was used. One found that

$$
\begin{aligned}
Q_m &= 83.3\ \text{tons/year}\\
Q_c &= 50\ \text{tons/year}\\
Q_r &= 35\ \text{lbs/year}\\
f &= \$300/\text{year}\\
\text{Lifetime} &= 12\ \text{years}
\end{aligned}
$$

The yearly profit would be

$$
\begin{aligned}
P_j &= 35.0 \times \$20 - 50 \times \$2 - 83.3 \times \$1 - \$300\\
&= \$216.70
\end{aligned}
$$

when finding the maximum total profit, the profits realized in the various years are simply summed.

The total profit is, therefore

$$P_{\mathrm{T}} = \sum_{j=1}^{12} P_j = 12 \times 216.7 = \$2600.40$$

The net present value for this uniform series of profits (Chapter 2) is calculated using

$$\text{NPV} = P_j \frac{(1+i)^n - 1}{i(1+i)^n}$$

Assuming an interest (discount) rate of 15% one finds that

$$\text{NPV} = 216.7 \frac{(1.15)^{12} - 1}{0.15 \times (1.15)^{12}} = \$1174.6$$

The question to be raised is 'Could the NPV be increased using a cutoff grade which, instead of being fixed, varies throughout the life of the mine?' If so, then 'What should be the cutoff grades as a function of mine life?' These questions have been addressed by Lane and are the subject of this section.

Assume that just prior to mining increment Q_m (shown to commence at time $t = 0$ in Figure 6.23 for simplicity), the present value of all remaining profits is V. This is composed of two parts. The first, PV_p, is from the profit P realized at time T by mining the quantity Q_m. The second, PV_w is obtained from profits realized by mining the material remaining after time T. These profits are indicated as P_1 occurring at time T_1, P_2 occurring at time T_2, etc., in Figure 6.23. The value of all these remaining profits for mining conducted after $t = T$ as expressed at time T is W. The present values of W and P, respectively, discounted to time $t = 0$ are given by

$$\text{PV}_w(t=0) = \frac{W}{(1+d)^T} \tag{6.27}$$

$$\text{PV}_p(t=0) = \frac{P}{(1+d)^T} \tag{6.28}$$

where d is the discount rate.

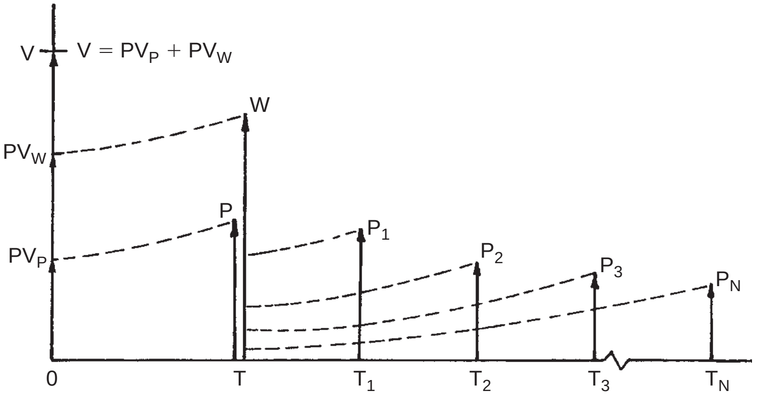


Figure 6.23. A diagrammatic representation of the NPV calculation.

The present value at time $t = 0$ is therefore

$$V = \frac{W}{(1+d)^T} + \frac{P}{(1+d)^T} \tag{6.29}$$

Since the present value at time $t = T$ of the remaining reserves is W, the difference v between the present values of the remaining reserves at times $t = 0$ and $t = T$ is

$$v = V - W \tag{6.30}$$

Equation (6.29) can be rewritten as

$$W + P = V(1+d)^T \tag{6.31}$$

Applying the binomial expansion to the term $(1+d)^T$ one finds that

$$(1+d)^T = 1 + Td + T\frac{(T-1)d^2}{2!} + \frac{T(T-1)(T-2)d^3}{3!} + \cdots \tag{6.32}$$

For d small, $(1+d)^T$ can be approximated by

$$(1+d)^T \approx 1 + Td \tag{6.33}$$

Combining Equations (6.31) and (6.33) results in

$$W + P = V(1 + Td) = V + VTd$$

or

$$V - W = P - VTd \tag{6.34}$$

Comparing Equations (6.30) and (6.34) one finds that

$$v = P - VTd \tag{6.35}$$

The profit P obtained through the mining of Q_m in time T is given as before by

$$P = (s - r)Q_r - cQ_c - mQ_m - fT \tag{6.36}$$

Combining Equations (6.35) and (6.36) yields

$$v = (s - r)Q_r - cQ_c - mQ_m - T(f + Vd) \tag{6.37}$$

One would now like to schedule the mining in such a way that the decline in remaining present value takes place as rapidly as possible. This is because later profits get discounted more than those captured earlier. In examining Equation (6.37), this means that v should be maximized. As in the previous section one first takes the derivative of v with respect to grade. Setting the derivative equal to zero, one can solve for the appropriate cutoff grades subject to mining, concentrating and refining constraints.

Step 1. Determination of the economic cutoff grades – one operation constraining the total capacity

(a) *Calculate cutoff grade assuming that the mining rate is the governing constraint.*
The time T_m is given by

$$T_m = \frac{Q_m}{M} \tag{6.38}$$

Substituting this into Equation (6.37) yields

$$v_m = (s-r)Q_r - cQ_c - \left[m + \frac{f+Vd}{M}\right] Q_m \tag{6.39}$$

Differentiating (6.39) with respect to grade g gives

$$\frac{\mathrm{d}v_m}{\mathrm{d}g} = (s-r)\frac{\mathrm{d}Q_r}{\mathrm{d}g} - c\frac{\mathrm{d}Q_c}{\mathrm{d}g} - \left[m + \frac{f+Vd}{M}\right] \frac{\mathrm{d}Q_m}{\mathrm{d}g} \tag{6.40}$$

However the quantity mined Q_m does not depend upon the grade:

$$\frac{\mathrm{d}Q_m}{\mathrm{d}g} = 0 \tag{6.41}$$

Hence

$$\frac{\mathrm{d}v_m}{\mathrm{d}g} = (s-r)\frac{\mathrm{d}Q_r}{\mathrm{d}g} - c\frac{\mathrm{d}Q_c}{\mathrm{d}g} \tag{6.42}$$

The relationship between the quantities refined Q_r and those sent for concentration Q_c is

$$Q_r = Q_c \bar{g}_c y \tag{6.43}$$

where $\bar{g}_c$ is the average grade of ore sent for concentration and y is the recovery in concentration.

Thus

$$\frac{\mathrm{d}Q_r}{\mathrm{d}g} = \bar{g}_c y \frac{\mathrm{d}Q_c}{\mathrm{d}g} \tag{6.44}$$

Substituting Equation (6.44) into (6.42) yields

$$\frac{\mathrm{d}v_m}{\mathrm{d}g} = \left[(s-r)\bar{g}_c y - c\right] \frac{\mathrm{d}Q_c}{\mathrm{d}g} \tag{6.45}$$

The average grade $\bar{g}_c$ is defined as the mining cutoff (breakeven) grade g_m when

$$\frac{\mathrm{d}v_m}{\mathrm{d}g} = 0 \tag{6.46}$$

Setting Equation (6.45) equal to zero and solving for $\bar{g}_c = g_m$ one finds that

$$g_m = \frac{c}{(s-r)y} \tag{6.47}$$

Substituting the values from the example yields

$$g_m = \frac{\$2}{(\$25 - \$5)1.0} = 0.10\,\text{lbs/ton}$$

(b) *Calculate cutoff grade assuming that the concentrating rate is the governing constraint.* If the concentrator throughput rate is the limiting factor then the time T is controlled by the concentrator.

$$T = \frac{Q_c}{C} \tag{6.48}$$

where Q_c is the total number of tons which will be sent to the concentrator, and C is the tons/year capacity.

Equation (6.37) becomes

$$v_c = (s-r)Q_r - cQ_c - mQ_m - (f + dV)\frac{Q_c}{C} \tag{6.49}$$

$$v_c = (s-r)Q_r - \frac{c+f+dV}{C}Q_c - mQ_m - m \tag{6.50}$$

Since the total amount of material Q_m is fixed,

$$mQ_m = \text{const}$$

Thus the cutoff grade affects only Q_r and Q_c.

Substituting as before

$$Q_r = Q_c\bar{g}_c y$$

one finds that

$$v_c = \left[(s-r)\bar{g}_c y - \left(c + \frac{f+dV}{C}\right)\right]Q_c - mQ_m \tag{6.51}$$

To make v_c as large as possible the term

$$(s-r)\bar{g}_c y - \left(c + \frac{f+dV}{C}\right)$$

should be as large as possible. At breakeven (the cutoff grade), the term is zero. Thus

$$g_c = \frac{c + \frac{f+dV}{C}}{y(s-r)} \tag{6.52}$$

(c) *Calculate cutoff grade assuming the refining rate is the governing constraint.* If the refinery output is the limiting factor then the time T is controlled by the refinery,

$$T = \frac{Q_r}{R} \tag{6.53}$$

where Q_r is output of the refinery and R is the refining/sales capacity per year.

Substituting into Equation (6.37) yields

$$v_r = (s-r)Q_r - cQ_c - mQ_m - (f + dV)\frac{Q_r}{R} \tag{6.54}$$

$$v_r = \left(s - r - \frac{f+dV}{R}\right)Q_r - cQ_c - mQ_m \tag{6.55}$$

As before

$$Q_r = g_r y Q_c$$

Thus

$$v_r = \left(s - r - \frac{f + dV}{R}\right) g_r y Q_c - cQ_c - mQ_m$$

The total amount of material in the pit is fixed, therefore

$$mQ_m = \text{const}$$

Maximizing the expression for V_r one finds that

$$\left(s - r - \frac{f + dV}{R}\right) g_r y Q_c = cQ_c$$

Solving yields

$$g_r = \frac{c}{\left(s - r - \frac{f+dV}{R}\right) y} \tag{6.56}$$

In summary, this first type of cutoff grade determination is based upon finding the grade for which the net increase in overall present value is zero. The expressions are as in formulas (6.47), (6.52) and (6.56):

$$g_m = \frac{c}{(s - r)y}$$

$$g_c = \frac{c + \frac{f+dV}{C}}{y(s - r)}$$

$$g_r = \frac{c}{\left(s - r - \frac{f+dV}{R}\right) y}$$

As can be seen, the expressions for g_c and g_r contain the unknown value of V.

Step 2. Determination of the economic cutoff grade by balancing the operations

This step is exactly the same as that discussed in the previous section. Hence only the results will be presented here.

$$g_{mc} = 0.50$$
$$g_{cr} = 0.60$$
$$g_{mr} = 0.456$$

Step 3. Determining the optimum of the six cutoff grades

There are six possible cutoff grades. Three are based simply upon the grade distribution of the mined material and capacities.

$$g_{mc} = 0.50$$
$$g_{cr} = 0.60$$
$$g_{mr} = 0.456$$

The other three are based upon capacities and cost/price. Substituting in the known values one finds that

$$g_m = 0.10$$

$$g_c = \frac{c + \frac{f+dV}{c}}{y(s-r)} = \frac{2 + \frac{300+0.15V}{50}}{1.0(25-5)}$$

$$= \frac{8 + 0.003V}{20}$$

$$g_r = \frac{c}{\left(s - r - \frac{f+dV}{R}\right)y}$$

$$= \frac{2}{\left(25 - 5 - \frac{300+0.15V}{40}\right)1.0} = \frac{2}{12.5 - 0.00375V}$$

Of these, two of the limiting economic cut-off grades are not known initially since they depend upon knowing the overall present value. This in turn depends upon the cutoff grade. Since the unknown V appears in the equations an iterative process must be used.

An optimum grade will be determined for each of the three pairs of operations. This will be followed by finding the optimum of the three final candidates. For the mine and the concentrator considered as a pair, there are three possible candidates for the optimum cutoff grade G_{mc}. These are g_m, g_c, and g_{mc}. The following rules are used to select G_{mc}.

$$G_{mc} = \begin{cases} g_m & \text{if} g_{mc} \le g_m \\ g_c & \text{if} g_{mc} \ge g_c \\ g_{mc} & \text{otherwise} \end{cases}$$

This simple sorting algorithm yields the middle value. Treating the concentrator and refinery as a pair, the optimum G_{rc} is found from

$$G_{rc} = \begin{cases} g_r & \text{if} g_{rc} \le g_r \\ g_c & \text{if} g_{rc} \ge g_c \\ g_{rc} & \text{otherwise} \end{cases}$$

Finally the optimum cutoff grade G_{mr} when the mine and refinery are treated as a pair is

$$G_{mr} = \begin{cases} g_m & \text{if} g_{mr} \le g_m \\ g_r & \text{if} g_{mr} \ge g_r \\ g_{mr} & \text{otherwise} \end{cases}$$

As the first step in the iteration process, it will be assumed that $V = 0$.

Applying these rules to the example values

$$g_m = 0.10$$
$$g_c = 0.40$$
$$g_r = 0.16$$
$$g_{mc} = 0.50$$
$$g_{rc} = 0.60$$
$$g_{mr} = 0.456$$

Table 6.25. Concentrator product as a function of concentrator cutoff grade with mine operating at capacity.

Concentrator cutoff (lbs/ton)	Concentrator feed (tons)	Average feed grade (lbs/ton)	Concentrator/refinery product (lbs)
0.0	100	0.50	50
0.1	90	0.55	49.5
0.2	80	0.60	48
0.3	70	0.65	45.5
0.4	60	0.70	42
0.5	50	0.75	37.5
0.6	40	0.80	32
0.7	30	0.85	25.5
0.8	20	0.90	18
0.9	10	0.95	9.5
1.0	0	1.00	0

one finds that

$$G_{mc} = 0.40$$
$$G_{rc} = 0.40$$
$$G_{mr} = 0.16$$

The overall optimum cutoff grade G is the middle value of G_{mc}, G_{mr}, and G_{rc}.

$$G = \text{middle value } (G_{mc}, G_{mr}, G_{rc})$$

In this case

$$G = 0.40$$

Step 4. Calculation of quantities

The next step in the procedure is to determine the maximum quantities Q_m, Q_c and Q_r which could be produced and not violate the capacities. Assume that 100 tons are mined ($Q_m = 100$). The grade distribution of this material is as shown in Table 6.24.

From Table 6.25, a cutoff grade of 0.40 would yield an average feed grade of 0.70. Each of the capacities must, however, be considered. A mining capacity ($Q_m = M$) of 100 tons would mean 60 tons to the concentrator and 42 product units. Both the concentrator and refinery capacities are exceeded. In meeting the concentrator capacity ($Q_c = 50$), the required mining and refinery capacities are:

$$Q_m = \frac{5}{6} \times 100 = 83.3$$
$$Q_r = 50 \times 0.70 = 35$$

These are both less than the maximum values. In meeting the refinery capacity of $Q_r = 40$, the required concentrating and mining capacities are:

$$Q_c = \frac{40}{0.7} = 57.1$$
$$Q_m = \frac{57.1}{60} 100 = 95.2$$

Thus the concentrating capacity is violated. The result is that the concentrator is the bottleneck.

In the further calculations

$$Q_m = 83.3$$
$$Q_c = 50$$
$$Q_r = 35$$

The profit from time period T is expressed as

$$P = (s - r)Q_r - cQ_c - mQ_m - fT$$

For $T = 1$ year one finds that

$$P = (25 - 5) \times 35 - 2 \times 50 - 1 \times 83.3 - 300 \times 1 = \$216.7$$

Since the total amount of material to be mined from the pit is $Q = 1000$ units, the number n of years required is

$$n = \frac{1000}{83.3} = 12 \text{ years}$$

The present value V corresponding to 12 equally spaced payments of $P = \$216.7$ using an interest rate of 15 percent is

$$V = \frac{216.7[1.15^{12} - 1]}{0.15(1.15)^{12}} = \$1174.6$$

This value of V becomes the second approximation of V (the first was $V = 0$) for use in the formulas to calculate g_c and g_r.

$$g_c = \frac{c + \frac{f+dV}{C}}{y(s-r)} = \frac{2 + \frac{300+0.15\times1174.6}{50}}{25-5} = 0.576$$

$$g_r = \frac{c}{s - r - \frac{f+dV}{R}y} = \frac{2}{20 - \frac{300+0.15\times1174.6}{40}} = 0.247$$

The new six choices become

$$g_m = 0.10$$
$$g_c = 0.576$$
$$g_r = 0.247$$

$$g_{mc} = 0.50$$
$$g_{rc} = 0.60$$
$$g_{mr} = 0.456$$

Applying the rules to select the overall pair optimum yields

$$G_{mc} = 0.50$$
$$G_{rc} = 0.576$$
$$G_{mr} = 0.247$$

The overall optimum is the middle value

$$G = 0.50$$

Returning to the grade distribution Table 6.24 one finds that the average grade is 0.75. If the mining rate $Q_m = 100$, then $Q_c = 50$ and $Q_r = 37.5$. Both the mine and the concentrator are at their rated capacities.

The profit in a given year is

$$\begin{aligned} P &= (s - r)Q_r - cQ_c - mQ_m - fT \\ &= 20 \times 37.5 - 2 \times 50 - 1 \times 100 - 300 \\ &= \$250 \end{aligned}$$

The number of years is

$$n = \frac{Q}{Q_m} = \frac{1000}{100} = 10 \text{ years}$$

The present value becomes

$$V = 250 \frac{1.15^{10} - 1}{0.15 \times 1.15^{10}} = \$1254.7$$

This becomes the third estimate for V to be used in calculating g_c and g_r.

$$g_c = \frac{2 + \frac{300+0.15\times1254.7}{50}}{20} = 0.588$$

$$g_r = \frac{2}{20 - \frac{300+0.15\times1254.7}{40}} = 0.257$$

The six possible values are:

$$\begin{aligned} g_m &= 0.10 \\ g_c &= 0.588 \\ g_r &= 0.257 \end{aligned}$$

$$\begin{aligned} g_{mc} &= 0.50 \\ g_{rc} &= 0.60 \\ g_{mr} &= 0.456 \end{aligned}$$

The optimum pairs are:

$$\begin{aligned} G_{mc} &= 0.50 \\ G_{rc} &= 0.588 \\ G_{mr} &= 0.257 \end{aligned}$$

and the overall optimum ($G = 0.50$) is the same as found with the previous estimate. Hence in year 1

Optimum cutoff grade = 0.50 lbs/ton
Quantity mined = 100 tons
Quantity concentrated = 50 tons
Quantity refined = 37.5 lbs
Profit = \$250

Table 6.26. Reserve distribution at the end of year 1.

Grade (lbs/ton)	Quantity (tons)
0.0–0.1	90
0.1–0.2	90
0.2–0.3	90
0.3–0.4	90
0.4–0.5	90
0.5–0.6	90
0.6–0.7	90
0.7–0.8	90
0.8–0.9	90
0.9–1.0	90
	Total = 900

Table 6.27. Reserve distribution at the start of year 8.

Grade (lbs/ton)	Quantity (tons)
0.0–0.1	30
0.1–0.2	30
0.2–0.3	30
0.3–0.4	30
0.4–0.5	30
0.5–0.6	30
0.6–0.7	30
0.7–0.8	30
0.8–0.9	30
0.9–1.0	30
	Total = 300

The reserves must now be adjusted to those given in Table 6.25 and the process is repeated assuming $V = 0$, calculating g_c and g_r, etc.

Through year 7, it will be found that the optimum cutoff grade remains at 0.50 with the quantities mined, concentrated and refined being 100, 50 and 37.5, respectively. The annual profit is \$250. The reserves going into year 8 are those given in Table 6.27. The balancing grades remain at

$$g_{mc} = 0.50$$
$$g_{rc} = 0.60$$
$$g_{mr} = 0.456$$

The first approximation for the economic cutoff grades ($V = 0$) is

$$g_m = 0.10$$
$$g_c = 0.40$$
$$g_r = 0.16$$

The optimum values of the pairs are

$$G_{mc} = G_{rc} = 0.40$$
$$G_{mr} = 0.16$$

The overall optimum is 0.40, and the quantities are

$$Q_m = 83.3$$
$$Q_c = 50$$
$$Q_r = 35$$

The profit is \$216.7 as before. The number of years becomes

$$n = \frac{300}{83.3} = 3.6 \text{ years}$$

The present value V becomes

$$V = 216.7\frac{(1.15)^{3.6} - 1}{0.15(1.15)^{3.6}} = 571.2$$

Substituting this into the formulas for g_c and g_r yields

$$g_c = 0.486$$
$$g_r = 0.193$$

Combining them with the others

$$g_m = 0.10$$
$$g_{mc} = 0.50$$
$$g_{rc} = 0.60$$
$$g_{mr} = 0.456$$

yields

$$G_{mc} = 0.486$$
$$G_{rc} = 0.486$$
$$G_{mr} = 0.193$$

The overall optimum cutoff is $G = 0.486$ and the average grade above cutoff drops to 0.743:

Tons	Grade	Tons × Grade
4.2	0.493	2.07
30	0.55	16.50
30	0.65	19.50
30	0.75	22.50
30	0.85	25.50
30	0.95	28.50
Total = 154.2	Avg = 0.743	Sum = 114.57

There are 154.2 ore tons out of the 300 tons remaining to be mined. Since the concentrator capacity is 50 tons/year, the mine life would be

$$n = \frac{154.2}{50} = 3.08 \text{ years}$$

Table 6.28. Reserve distribution at the start of year 9.

Grade (lbs/ton)	Quantity (tons)
0.0–0.1	20.27
0.1–0.2	20.27
0.2–0.3	20.27
0.3–0.4	20.27
0.4–0.5	20.27
0.5–0.6	20.27
0.6–0.7	20.27
0.7–0.8	20.27
0.8–0.9	20.27
0.9–1.0	20.27
	Total = 202.27

The yearly mine production becomes

$$Q_m = \frac{300}{3.08} = 97.3$$

and $Q_r = 37.15$. Calculating the profit one finds that

$$P = 20 \times 37.15 - 2 \times 50 - 1 \times 97.3 - 300 = 245.7$$

The corresponding present value is

$$V = 245.7\frac{(1.15)^{3.08} - 1}{0.15 \times (1.15)^{3.08}} = \$572.96$$

Repeating the process with this new estimate of V yields

$$g_c = 0.486$$
$$g_r = 0.193$$

These are the same as before. Hence the values for year 8 are

$$\begin{aligned} G &= 0.486 \\ Q_m &= 97 \\ Q_c &= 50 \\ Q_r &= 37.1 \\ \text{Profit} &= \$245.7 \end{aligned}$$

The reserves are those given in Table 6.28. In year 9, the initial values for a cutoff of 0.4 are

$$\begin{aligned} Q_m &= 83.3 \\ Q_c &= 50 \\ Q_r &= 35 \end{aligned}$$

The profit would be \$216.7.

Based upon this mining rate, the reserves would last

$$n = \frac{202.7}{83.3} = 2.43 \text{ years}$$

and the present value is

$$V = 216.7\frac{(1.15)^{2.43} - 1}{0.15 \times (1.15)^{2.43}} = \$416.0$$

Recomputing g_r and g_c we obtain

$$g_c = 0.462$$
$$g_r = 0.183$$

The other possible values are:

$$g_m = 0.10$$
$$g_{mc} = 0.50$$
$$g_{rc} = 0.60$$
$$g_{mr} = 0.456$$

The optimum pair values are:

$$G_{mc} = 0.462$$
$$G_{rc} = 0.462$$
$$G_{mr} = 0.183$$

The middle value of these is

$$G = 0.462$$

Examining the reserve distribution suggests that there are 109.05 tons out of the total 202.7 tons which are above cutoff. The average grade of this remaining ore is 0.731:

Tons	Grade	Tons × Grade
7.70	0.48	3.70
20.27	0.55	11.15
20.27	0.65	13.18
20.27	0.75	15.20
20.27	0.85	17.23
20.27	0.95	19.26
Total = 109.05	Avg = 0.731	Sum = 79.72

Since the maximum concentrating rate is 50 tons/year the life is

$$\frac{109.05}{50} = 2.18 \text{ years}$$

The amount of product is

$$Q_r = 50 \times 0.731 = 36.55$$
$$Q_m = \frac{202.7}{2.18} = 93$$

The profit becomes

$$P = 20 \times 36.55 - 2 \times 50 - 1 \times 93 - 300 = \$238$$

Table 6.29. Reserve distribution at the start of year 10.

Grade (lbs/ton)	Quantity (tons)
0.0–0.1	11
0.1–0.2	11
0.2–0.3	11
0.3–0.4	11
0.4–0.5	11
0.5–0.6	11
0.6–0.7	11
0.7–0.8	11
0.8–0.9	11
0.9–1.0	11
	Total = 110

The present value is

$$V = 238\frac{(1.15)^{2.18} - 1}{0.15 \times 1.15^{2.18}} \times \$417$$

Iterating again does not change the values. The new distribution is shown in Table 6.29.

In year 10, the initial values for a cutoff of 0.4 yields:

$$\begin{aligned} Q_m &= 83.3 \\ Q_c &= 50 \\ Q_r &= 35 \\ \text{Profit} &= \$216.7 \end{aligned}$$

Based upon this mining rate, the reserves would last

$$n = \frac{110}{83.3} = 1.32 \text{ years}$$

The present value is

$$V = 216.7\frac{(1.15)^{1.32} - 1}{0.15 \times (1.15)^{1.32}} = \$243.4$$

Calculating g_c and g_r and using this approximation for V yields

$$\begin{aligned} g_c &= 0.437 \\ g_r &= 0.172 \end{aligned}$$

The other possible values are:

$$\begin{aligned} g_m &= 0.10 \\ g_{mc} &= 0.50 \\ g_{rc} &= 0.60 \\ g_{mr} &= 0.456 \end{aligned}$$

The optimum pair values are:

$$G_{mc} = 0.437$$
$$G_{rc} = 0.437$$
$$G_{mr} = 0.172$$

The optimum value is $G = 0.437$. Examining the reserve distribution suggests that there are 62 tons of the 110 tons remaining which are above this cutoff.

Tons	Grade	Tons × Grade
7	0.469	3.28
11	0.55	6.05
11	0.65	7.15
11	0.75	8.25
11	0.85	9.35
11	0.95	10.45
Total = 62	Avg = 0.718	Sum = 44.53

The average grade is 0.718. The number of years would be

$$n = \frac{62}{50} = 1.24 \text{ years}$$

The rate of mining and refining would be

$$Q_m = \frac{110}{1.24} = 89$$
$$Q_r = 35.9$$

and the profit would become

$$P = 20 \times 35.9 - 2 \times 50 - 1 \times 89 - 300$$
$$= \$229$$

The present value is

$$V = \$229 \frac{(1.15)^{1.24} - 1}{0.15 \times (1.15)^{1.24}} = \$243$$

Further iteration yields no change. In year 11, the grade distribution is shown in Table 6.30.

The initial values ($V = 0$), yield a cutoff of 0.4 and

$$Q_m = 21$$
$$Q_c = 12.6$$
$$Q_r = 8.8$$

The time would be the largest of

$$T_m = \frac{21}{100} = 0.21$$

$$T_c = \frac{12.6}{50} = 0.25$$

$$T_t = \frac{8.8}{40} = 0.22$$

Table 6.30. Reserve distribution at the start of year 11.

Grade (lbs/ton)	Quantity (tons)
0.0–0.1	2.1
0.1–0.2	2.1
0.2–0.3	2.1
0.3–0.4	2.1
0.4–0.5	2.1
0.5–0.6	2.1
0.6–0.7	2.1
0.7–0.8	2.1
0.8–0.9	2.1
0.9–1.0	2.1
	Total = 21.0

which is again controlled by the concentrator. The profit is

$$\begin{aligned} P &= 20 \times 8.8 - 12.6 \times 2 - 21 \times 1 - \frac{300}{4} \\ &= \$54.8 \end{aligned}$$

The present value is

$$V = \$54.8 \frac{(1.15)^{0.25} - 1}{0.15 \times (1.15)^{0.25}} = \$12.5$$

Solving for g_c and g_r yields

$$\begin{aligned} g_c &= 0.402 \\ g_r &= 0.161 \end{aligned}$$

Combining with the others

$$\begin{aligned} g_m &= 0.16 \\ g_{mc} &= 0.50 \\ g_{rc} &= 0.60 \\ g_{mr} &= 0.456 \end{aligned}$$

one finds that

$$\begin{aligned} G_{mc} &= 0.402 \\ G_{rc} &= 0.402 \\ G_{mr} &= 0.161 \end{aligned}$$

The cutoff grade is

$$G = 0.402$$

The distribution is only slightly changed and further iteration is not warranted.

Table 6.31. The production schedule determined by the first pass.

Year	Optimum cutoff grade (lbs/ton)	Quantity mined (tons)	Quantity concentrated (tons)	Quantity refined (lbs)	Profit ($)	Net present value
1	0.5	100	50	37.5	250	1255
2	0.5	100	50	37.5	250	1193
3	0.5	100	50	37.5	250	1122
4	0.5	100	50	37.5	250	1040
5	0.5	100	50	37.5	250	946
6	0.5	100	50	37.5	250	838
7	0.5	100	50	37.5	250	714
8	0.486	97	50	37.1	245.7	574
9	0.462	93	50	36.55	238	417
10	0.437	89	50	35.9	229	243
11	0.40	21	12.6	8.8	55	53

Table 6.32. The final schedule for the manual example.

Year	Optimum cutoff grade (lbs/ton)	Quantity mined (tons)	Quantity concentrated (tons)	Quantity refined (lbs)	Profit ($)	Net present value
1	0.50	100.0	50.0	37.5	250.0	1257.8
2	0.50	100.0	50.0	37.5	250.0	1196.5
3	0.50	100.0	50.0	37.5	250.0	1126.0
4	0.50	100.0	50.0	37.5	250.0	1044.9
5	0.50	100.0	50.0	37.5	250.0	951.7
6	0.50	100.0	50.0	37.5	250.0	844.5
7	0.50	100.0	50.0	37.5	250.0	721.1
8	0.49	97.2	50.0	37.1	245.6	579.3
9	0.46	93.0	50.0	36.6	238.2	420.6
10	0.44	88.7	50.0	35.9	229.5	245.5
11	0.41	21.0	12.5	8.8	54.7	52.8

The net present value is calculated using the yearly profits.

$$\begin{aligned}
\mathrm{NPV} &= 250\frac{(1.15)^7 - 1}{0.15 \times (1.15)^7} + \frac{245.7}{(1.15)^8} + \frac{238}{(1.15)^9} + \frac{229}{(1.15)^{10}} + \frac{55}{(1.15)^{10.25}} \\
&= 1040 + 80.32 + 67.70 + 56.61 + 13.12 \\
&\cong \$1258
\end{aligned}$$

Step 5. Repetition of the iteration process

In Table 6.31, the present value column reflects the current approximation to V as each years cutoff grade was calculated. The present value of $1258 obtained using the yearly profits should be the same as that shown in the table for year 1. Since the values are not the same ($1258 versus $1255), the process is repeated from the beginning using $V = \$1258$ as the initial estimate for V. Using a computer this iterative procedure is completed in fractions of a second. The final results are shown in Table 6.32. The NPV is slightly higher than the $1255 which would have been obtained by maintaining a constant cutoff grade of 0.5.

In summary:

– Initially the mine and the concentrator are in balance, both operating at capacity. In the last few years, the concentrator is the limiter.
– The cutoff grade begins at 0.50 lbs/ton and drops to 0.41 lbs/ton at the end of mine life.
– Mine life is slightly more than 10 years.
– Total copper produced = 380.9 lbs.
– Total profits = $2518.
– Net present value = $1257.80.

This net present value should be compared to that of $1174.60 obtained with the fixed cutoff grade.

6.8 MATERIAL DESTINATION CONSIDERATIONS

6.8.1 *Introduction*

The term ‘cutoff grade’ is a rather poorly defined term in the mining literature. A major reason for this is that there are many different cutoff grades. Furthermore the values change with time, mining progress, etc. A cutoff grade is simply a grade used to assign a destination label to a parcel of material.

The destination can change. During the evaluation of final pit limits, the destinations to be assigned are:

– to the surface, and
– left in the ground.

Once the destination ‘to the surface’ has been assigned, then the destination label ‘where on the surface’ must be assigned as well. In the distant past there were really only two surface destinations:

– to the mill, and
– to the waste dump.

A grade was used to assign the location. The distinction between destinations was called the mill cutoff grade. In more recent times, the potential future value of material carrying values has been recognized. Hence the lean (low grade) ore dump has become a destination. Thus the 3 destinations require 2 distinguishing grades:

Destination	Assignment
– to the mill	
	mill cutoff grade
– to the lean ore dump	
	waste cutoff grade
– to the waste dump	

Today there are many more possible destinations as our ability to handle and treat materials have improved. Leach dumps/leach pads are a common destination. An active stockpile is a less common destination.

This section will deal with alternate destinations to the mill and waste dump. These will be discussed with respect to cutoff grade. However the reader should remember that these simply are a way of assigning material destinations.

6.8.2 *The leach dump alternative*

For copper minerals and some others as well, the leach dump (or pad) is an alternative destination to the mill or waste dump. Although this section will focus on copper, the approach used is typical. Two cutoff grades must be determined:

1. Waste dump – leach cutoff.
2. Leach dump – mill cutoff.

In one copper leaching process, sulfuric acid is percolated through the dump. The copper is taken into solution as cupric sulfate or cuprous sulfate. The collected solution is then run through a pachuca tank containing shredded iron where the iron goes into solution and the copper is precipitated. The reactions are given below:

Dump

$$\text{Leach material} + H_2SO_4 \rightarrow CuSO_4 \text{ or } Cu_2SO_4$$

Pachuca tank

$$CuSO_4 + Fe \rightarrow Cu + FeSO_4 \cdot H_2O$$

The precipitated copper is then sent to the smelter. A full discussion of copper recovery from dump material is far beyond the scope of this book. The rate of recovery and the overall percent recovery depend upon both the ore and waste minerals present, the type of sprinkling and collection system, the size distribution of the particles involved, the way the material was placed, etc. Here the results presented by Davey (1979c) will be used for illustrative purposes.

For copper mineralization type 1, no recovery is expected when the copper content is below 0.2%. Using mineral dressing terminology, this would be the 'Fixed Tails' limit. Thus, when considering material destinations, the waste dump – leach dump cutoff criteria would be:

$$\text{Waste dump destination: copper content} < 0.2\%$$

$$\text{Leach dump destination: copper content} \geq 0.2\%$$

There will be another cutoff grade above which all material should be sent to the mill. To determine this value one must look both at the total copper recovery through leaching and the recovery as a function of time. For copper mineralization type 1, a fairly uniform distribution of the various grades may be assumed for the low grade material having a head grade of 0.4% copper or less. In this case

$$\text{Head grade} \leq 0.4\%$$

the percent recovery is expressed by

$$\text{Recovery} = \frac{\text{Head grade} - \text{Fixed tails limit}}{\text{Head grade}} \times 100\% \tag{6.57a}$$

If the head grade is better than 0.4%,

$$\text{Head grade} > 0.4\%$$

then the copper recovery by leaching is 50%

$$\text{Recovery} = 50\% \tag{6.57b}$$

Four examples will be used to demonstrate the use of the recovery formula.

Table 6.33. Recovery factor as a function of year for the leach example (Davey, 1979c).

Year	Recovery factor
1	0.40
2	0.30
3	0.14
4	0.10
5	0.06

1. Head grade = 0.45% Cu

$$\text{Recovery} = 50\%$$

$$\text{Recovered copper} = \frac{0.45}{100} \times 0.5 \times 2000 = 4.5\,\text{lbs/ton}$$

2. Head grade = 0.40% Cu

$$\text{Recovery} = \frac{0.40 - 0.20}{0.40} \times 100 = 50\%$$

$$\text{Recovered copper} = \frac{0.40}{100} \times 0.5 \times 2000 = 4.0\,\text{lbs/ton}$$

3. Head grade = 0.30% Cu

$$\text{Recovery} = \frac{0.30 - 0.20}{0.30} \times 100 = 33.3\%$$

$$\text{Recovered copper} = \frac{0.30}{100} \times 0.333 \times 2000 = 2\,\text{lbs/ton}$$

4. Head grade = 0.15% Cu

$$\text{Recovery} = 0\%$$

$$\text{Recovered copper} = 0\,\text{lbs/ton}$$

The copper is recovered from the ton of material over some period of time, normally several years. The amount recovered per year from the ton normally decreases with time as shown below.

$$\text{Yearly recovery (lbs)} = \text{Recovery factor} \times \text{Total lbs to be recovered} \tag{6.58b}$$

The factors are given in Table 6.33.

For a copper ore running 0.55% (head grade), the amount of copper in situ is 11 lbs/ton. Of this 5.5 lbs is expected to be recovered over the 5-year period. For year 1, one would expect to recover

$$\text{Year 1 copper recovered} = 0.40 \times 5.5\,\text{lbs} = 2.20\,\text{lbs}$$

For year 2 the result would be

$$\text{Year 2 copper recovered} = 0.30 \times 5.5\,\text{lbs} = 1.65\,\text{lbs}$$

Table 6.34 summarizes the overall recovery.

Table 6.34. Estimated recovery as a function of year (Davey, 1979c).

Year	Estimated recovery (lbs/ton)
1	2.20
2	1.65
3	0.77
4	0.55
5	0.33
	Total = 5.50

To demonstrate the application, consider the following example for a leach ore running 0.55% Cu. The costs of mining and transporting the mineral are assumed to be the same for both the milling and leaching alternatives. Since the material will be removed from the pit in any case, these are considered as sunk costs. An interesting exercise for the reader is to calculate the value of potential leach material when deciding the ultimate pit limits. As can be seen in Table 6.35, the net value per ton of material when considered for dump leaching is $2.07.

The same process is repeated for the milling alternative. This is done in Table 6.36.

As can be seen, the net value is now $2.75/ton of material treated. Obviously, the destination of the material should be the mill.

The process illustrated in Table 6.35 and 6.36 is repeated until a breakeven grade between leaching and milling is found (see Figure 6.24). In this particular case it is 0.45% Cu. Material carrying grades above this level should be sent to the mill. Material grading between 0.2 and 0.45% copper should go to the leach dump.

In this simplified calculation, two important considerations have been left out. The first is the cost of capital associated with the mill. Inclusion of these costs will increase the breakeven grade. However, possible by-product credits from gold, molybdenum, etc., which would be realized by milling and not leaching have not been included. Obviously, a more detailed analysis would take into account both of these factors.

It may also be that the mill capacity is taken up with higher grade material. In such a case another destination, a stockpile, might be considered. Cutoff grades are involved in the decision as well.

If these cutoff grades are applied to the tonnage-grade curve given in Figure 6.25, one finds the following.

Class of material	Grade %	Tons	Average grade* (%)
Mill ore	≥ 0.45	7.4×10^6	0.96
Leach ore	$0.20 < g < 0.45$	3.4×10^6	0.33
Waste	≤ 0.20	20.0×10^6	–

*See Figure 6.26.

In this particular case, the actual production results were

Class of material	Tons	Average grade % Cu
Mill ore	7.5×10^6	0.93
Leach	6.4×10^6	0.35
Waste	16.9×10^6	–

Table 6.35. Economic analysis regarding the leaching of material containing 0.55% copper (Davey, 1979c).

	$/ton of material
A. Production cost breakdown	
1. Precipitation cost	
($0.20/lb of recovered copper)	
– Recovery = 5.5 lbs/ton	
– Precipitation cost = 5.5 × $0.20 = $1.10	$1.10
– The precipitate runs 85% copper	
2. Treatment cost	$1.08
Smelting	
($50/ton of precipitate)	
– Ratio of concentration $= \frac{2000 \times 0.85}{5.5} = 309.09$	
– Smelting cost/ton of leach material $= \frac{\$50}{309.09} = \$0.16$	
– Recovery = 97.5%	
– Freight (smelter → refinery)	
($50/ton of blister copper)	
– Freight cost $= \frac{5.5 \times 0.98}{2000} \times \$50 = \$0.1351$	
Refining	
($130/ton of blister copper)	
– Refining cost $= \frac{5.404}{2000} \times \$130 = \$0.351$	
– Recovery = 99.7%	
Selling and delivery	
($0.01/lb Cu)	
cost = 0.01 × 5.39 = $0.054	
General plant	
($0.07/lb Cu)	
cost = 0.07 × 5.39 = $0.38	
3. Total production cost	$2.18
B. Revenue ($/ton of leach material)	
1. Total value of in situ saleable copper	
(price $1.00/lb)	
5.39 lbs × $1/lb = $5.39	
(discount rate $i = 0\%$)	
2. Discounted value	
(discount rate = 12.5%)	

Year	Recovered copper (lbs)	Discounted value ($)
1	2.15	1.92
2	1.62	1.28
3	0.75	0.53
4	0.54	0.34
5	0.32	0.18
		$4.25

3. In general no by-products are recovered.
4. It has been assumed that costs and prices have inflated at the same rate.

C. Net value ($/ton) of leach material

Net value = Revenue – Costs
= $4.25 – $2.18 = $2.07

Table 6.36. Milling evaluation of 0.55% copper feed (Davey, 1979c).

	$/ton of material
A. Production cost breakdown	
1. Milling cost	
– Operating cost	= $2.80
– G&A cost (15% of operating)	= 0.42
Mill recovery = 80%	
Mill concentrate = 20% Cu	
2. Shipment of mill concentrate to smelter ($1.40/ton of concentrate)	
Rate of concentration	
$r = \dfrac{2000 \times 0.20}{\frac{0.55}{100} \times 2000 \times 0.80} = 45.45$	
Freight cost $= \dfrac{\$1.40}{45.45} = \$0.03$	= 0.03
3. Treatment cost	
Smelting ($50/ton of concentrate)	
– Cost $= \dfrac{\$50}{45.45} = \$1.10$	= 1.10
– Recovery = 97.5%	
Freight (smelter → refinery) ($50/ton of blister copper)	
Blister transport $= \$50 \times \dfrac{0.975 \times 8.8}{2000} = \$0.21$	= $0.21
Refining cost ($130.00/ton of blister copper)	
– Recovery = 99.8%	
– Cost $= \$130 \times \dfrac{8.58}{2000} = \$0.56$	= $0.56
Selling and delivery ($0.01/lb of copper)	
Cost $= 8.56 \times 0.01 = \$0.09$	= $0.09
General plant ($0.07/lb of copper)	
Cost $= 8.56 \times 0.07 = \$0.60$	= $0.60
4. Total production cost	= $5.81
B. Revenues ($/ton)	
There are a total of 8.56 lbs of copper recovered. A price of $1/lb will be used. Hence	
Revenue $= 8.56 \times \$1/\text{lb} = \$8.56$	
C. Net value ($/ton)	
The net value is given as the revenue minus the costs. It has been assumed here that there is no appreciable time required in the processing chain so that discounting is not used.	
Net value $= \$8.56 - \$5.81 = \$2.75$	

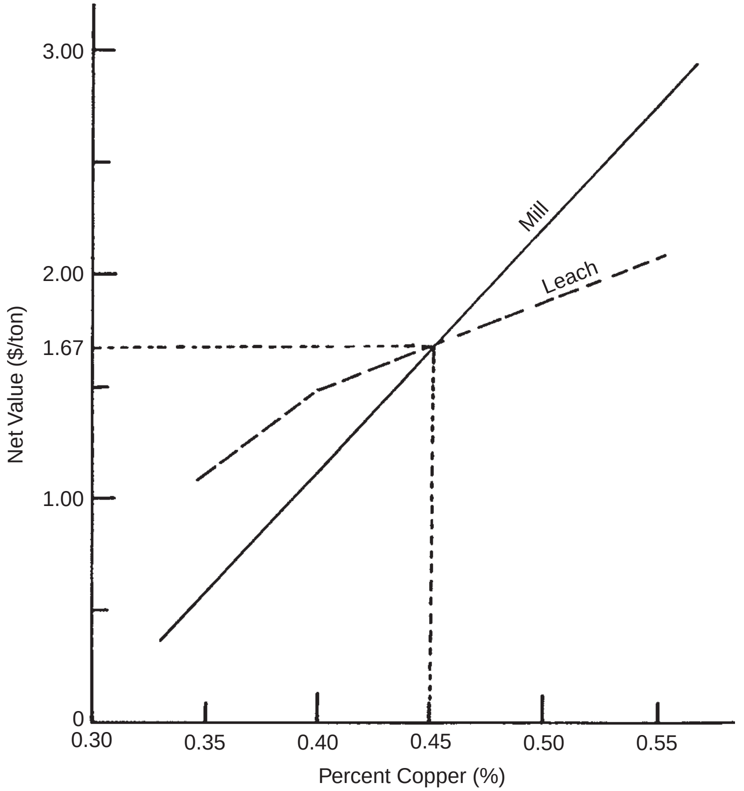


Figure 6.24. Breakeven between milling and leaching.

6.8.3 *The stockpile alternative*

As has been discussed there are a series of cutoff grades which are applied during the life of a mine-mill complex. Early in the life, there is a desire to recover the capital investment as early as possible. Since the mill capacity, in terms of tons per day, is normally fixed, it is better to use this capacity with higher rather than lower grade material. Hence the initial mill cutoff grade may be fairly high. As the mine-mill complex matures, the mill cutoff will normally decrease. Assume that for a copper operation the set mill cutoff is 0.7% Cu. The mining cutoff (based upon covering the production costs, smelting, refining, sales, etc.) could be 0.3%. If the material has to be mined to reach deeper higher grade ore, then the breakeven cutoff for the mill destination could be even lower than 0.3%. As indicated, because of mill capacity restrictions and the requirement for rapid capital payback, the mill cutoff is 0.7%. Therefore a large amount of material which, if produced later in the life of the mine would go to the mill, is now below cutoff. One possibility is to stockpile the material lying between 0.3% and 0.7% for later treatment by the mill. An alternative would be to send the material to leach dumps for a quicker return of the costs expended in the mining. No rehandling (with its added cost) would be required. Also there would not be a cost associated with the storage areas themselves.

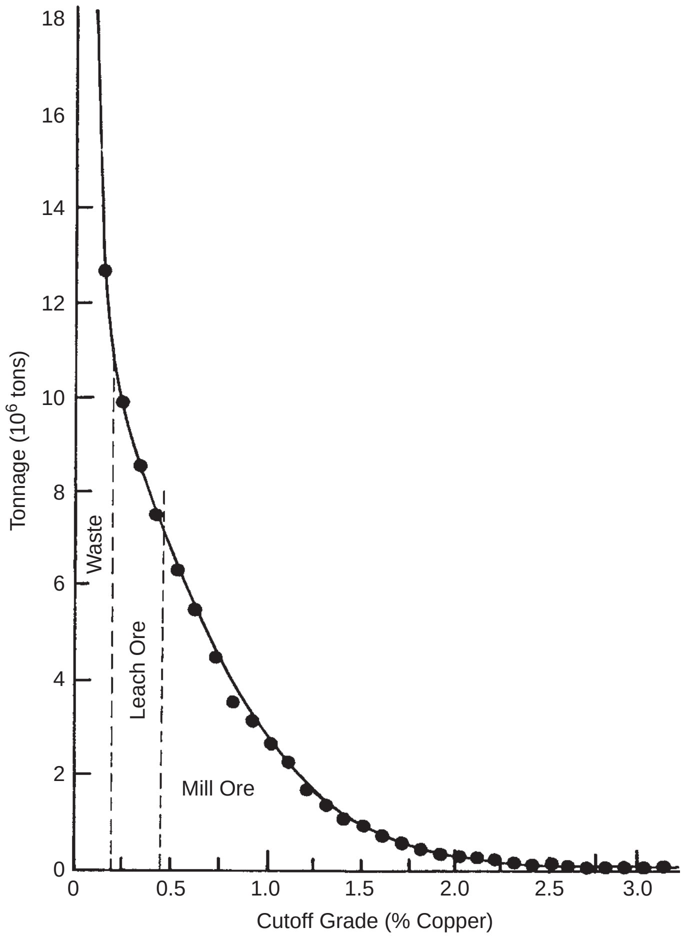


Figure 6.25. Superposition of the mill-leach and leach-waste cutoff grades on the Silver Bell oxide pit cumulative tonnage curve.

There are, in summary, a number of disadvantages with ore stockpiles (Schellman, 1989):

1. A rehandling step is necessary. The costs involved may be significant.

2. Space is required to accommodate the various qualities. Often such space is scarce near the operations.

3. Over time some materials become more difficult to treat.

4. Additional expense is incurred in tracking the various qualities of material (sampling, production control).

For these and other reasons, stockpiles have not been popular with most mining operations. There are some exceptions however.

Taylor (1985) justified the use of the stockpile system through the use of some actual examples from mines. At the Craigmont copper mine, a substantial sub-grade stockpile that averaged about 0.6% Cu was accumulated from the initial high grade open pit. The operation then moved underground, where sublevel caving yielded a daily output of about

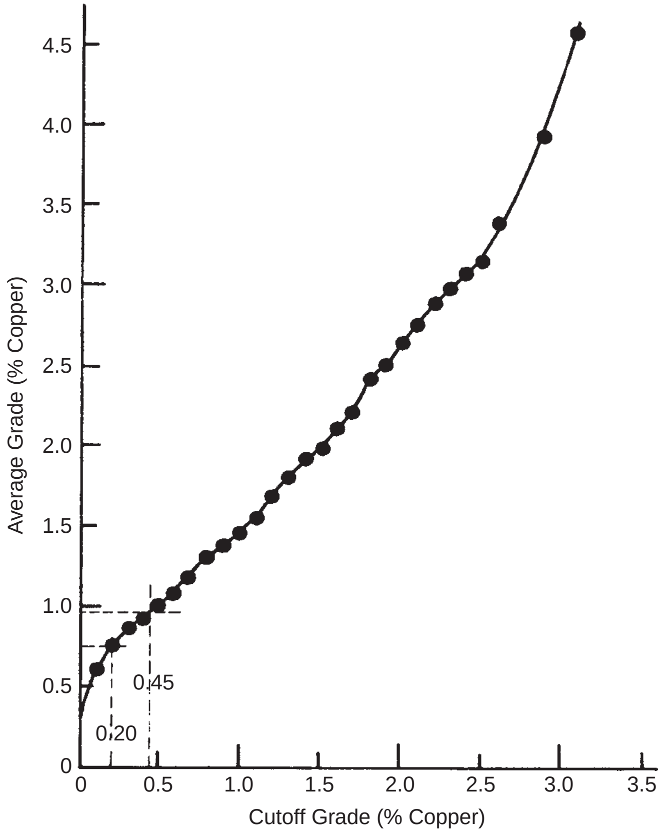


Figure 6.26. Superposition of the mill-leach and leach-waste cutoff grades on the Silver Bell oxide pit average grade curve.

4500 tons at 1.8% Cu. For many years the stockpile provided the remaining 1500 tons per day that were needed to supply the mill. Another practical example is the Gibraltar copper mine. A cutoff grade of 0.3% Cu was used in the early years and more than 100,000,000 short tons, grading down to 0.25% Cu, were stockpiled. In 1982–83 this surface ore enjoyed a cost advantage of at least $2.50 Canadian/ton over ore that had yet to be exposed in the pit. At the then low copper price, the 0.27% grade had a higher net value than the new 0.37% Cu ore.

A leach dump is a form of stockpile since cutoff grades have been used to select the material for placement. Generally no rehandle is involved. Today a number of such 'waste' dumps are being retreated with improved technology and there is reason to believe that this process will occur even for the 'waste' dumps being created today.

There are a number of advantages to having stockpiles:

1. They can be used for blending to ensure a constant head grade to the mill. Normally recoveries are higher when such fluctuations are low.

2. They can be used as a buffer to make up for production short falls due to various reasons.

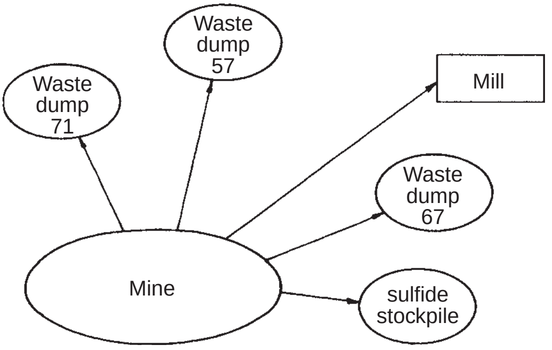


Figure 6.27. Schematic representation of various material destination (Schellman, 1989).

3. The metal recovery is generally far higher in the mill than in the leach dump. Thus resource utilization is better. Since the overall recovery is higher the overall costs get spread over a bigger base.

4. The presence of a buffer offers more flexibility in examining design options. For example one haulage road might be used instead of two due to the production security provided by the buffer. Steeper pit slopes might be considered.

In the future it is expected that the use of stockpiles will play a much more important part in the life of mines.

Computerized truck dispatching has simplified the changing of destinations. As can be seen in Figure 6.27, the dumping points are the concentrator plant, the three waste dumps (71, 57, and 67) and the sulfide stockpile. The addition of new dumps would not introduce major changes in the loading-haulage operation. Strategies regarding the number and types of stockpiles as well as how to best incorporate the stockpile into the overall operating plan (when and how much to draw as compared to primary production) must be developed.

Lane (1988) has provided a rather complete discussion regarding use of the stockpile alternative.

Schellman (1989) recently modified Lane's algorithm to include the stockpiling option with the grades below current mill cutoff being segregated. The optimum time to draw upon the stockpile as well as the tons and grades moved have been determined for three options. His approach will be illustrated using the data given in Table 6.37.

There are a total of 600×10^6 tons to be mined (100×10^6 tons in each phase). The capacities and costs used in applying the standard Lane approach described in Section 6.7 are given in Table 6.38. The results without stockpiling are given in Table 6.39. As can be seen the total profits (P_T) are \$$484.2 \times 10^6$ and the net present value (NPV) is \$$150.33 \times 10^6$. Of the total tonnage mined 319×10^6 goes to the mill and the remainder 281×10^6 tons to the waste dump. The cutoff grade in the final year is 0.23% Cu.

In the proposed system, a series of stockpiles are generated with material which is below the cutoff grade for each period but above the lowest cutoff grade for the whole project. Considering only increment 1 in the preceding example (Table 6.39), all material below 0.5% and above 0.23% will be sent to the stockpile. For the second increment (year 6), material below 0.53% and above 0.23% will be sent to the stockpile.

Three different alternatives for handling the stockpiled material will be considered.

Table 6.37. Input data for the demonstration example (Lane, 1964) (The material amounts given are in 10^6 tons).

Grade categories (%Cu)	Pit increments in mining sequence					
	1	2	3	4	5	6
0.00–0.15	14.4	15.9	17.9	20.3	23.4	27.7
0.15–0.20	4.6	5.1	5.5	6.3	7.2	8.3
0.20–0.25	4.4	4.9	5.4	6.0	6.7	7.7
0.25–0.30	4.3	4.7	5.3	5.6	6.4	7.3
0.30–0.35	4.2	4.5	4.9	5.5	6.2	6.7
0.35–0.40	4.1	4.4	4.7	5.3	5.6	6.3
0.40–0.45	3.9	4.3	4.6	4.9	5.4	5.7
0.45–0.50	3.8	4.1	4.5	4.8	5.1	5.3
0.50–0.55	3.7	3.9	4.2	4.5	4.6	4.7
0.55–0.60	3.6	3.8	3.9	4.2	4.4	4.3
0.60–0.65	3.4	3.6	3.8	3.9	4.0	3.7
0.65–0.70	3.3	3.5	3.7	3.7	3.6	3.3
+0.70	42.3	37.5	31.6	25.0	17.4	9.0
Average % Cu	1.13	1.07	1.00	0.93	0.87	0.80

Table 6.38. Capacities and costs used in the demonstration example (Lane, 1964).

Capacities:		
	Mine	20 million tons material/year
	Mill	10 million tons ore/year
	Refinery	90,000 tons product/year
Costs:		
	Mining	\$0.50/ton of material from pit
	Mill	\$0.60/ton of ore to mill
	Refining	\$50.0/ton of product
	Fixed charges	\$4 million/year
	Selling price	\$550/ton of product
	Recovery	90%

Alternative A. Mill feed coming simultaneously from the mine and stockpile

In this alternative, the analysis of whether or not to send material from the stockpile to the mill is done taking into account the profit generated by the material in the stockpile and the profit generated by the material in the push back (pit increment). If the profitability of the ore in the stockpile is greater than that of the ore in the mine, ore from both the stockpile and the mine will feed the mill. The comparison is established between the same grade categories for both alternatives. If, for example, at the mine the mineral grades between 0.4% and 0.7% and at the stockpile it grades between 0.4% and 0.55%, only the mineral between 0.4% and 0.55% will be examined. If the profitability of the stockpiled ore is better, then ore between 0.4% and 0.55% will be sent from the stockpile. Ore with grades between 0.55% and 0.7% will be sent from the mine. Mined material between 0.23% and 0.55% would be sent to the stockpile.

This stockpile system is dynamic, in the sense that, for some determined year, the stock pile is receiving ore from the mine, and sending ore to the mill. When the mining of the

Table 6.39. Final production schedule for the demonstration example (Schellman, 1989).

Year	Pit increment	Cutoff grade (% Cu)	Quantity mined (10^6 tons)	Quantity concentrated (10^6 tons)	Quantity refined (10^6 tons)	Profit (10^6 \$)	Total profit (10^6 \$)	Net present value (10^6 \$)
1	1	0.50	17.9	10.0	90.0	26.1	484.2	150.33
2	1	0.50	17.9	10.0	90.0	26.1	458.1	146.80
3	1	0.50	17.9	10.0	90.0	26.1	432.0	142.75
4	1	0.50	17.9	10.0	90.0	26.1	405.9	138.08
5	1	0.50	17.9	10.0	90.0	26.1	379.8	132.72
6	1	0.50	10.7	6.0	54.2	15.7	353.7	126.55
6	2	0.53	8.0	4.0	34.2	9.1	338.0	126.55
7	2	0.5	20.0	10.0	85.9	23.0	328.9	120.69
8	2	0.53	20.0	10.0	85.9	23.0	305.9	115.83
9	2	0.53	20.0	10.0	85.9	23.0	282.9	110.23
10	2	0.53	20.0	10.0	85.9	23.0	259.9	103.80
11	2	0.53	20.0	10.0	85.9	23.0	259.9	103.80
11	3	0.47	7.8	3.9	29.5	7.0	222.8	96.40
12	3	0.47	20.0	10.0	76.1	18.0	215.8	89.80
13	3	0.47	20.0	10.0	76.1	18.0	197.8	85.22
14	3	0.47	20.0	10.0	76.1	18.0	178.8	79.95
15	3	0.47	20.0	10.0	76.4	18.0	161.8	73.90
16	3	0.45	12.3	6.4	47.7	11.3	143.8	66.94
16	4	0.41	7.2	3.6	24.0	4.8	132.5	66.94
17	4	0.41	20.0	10.0	66.5	13.2	127.7	60.87
18	4	0.41	20.0	10.0	66.5	13.2	114.5	56.77
19	4	0.40	19.4	10.0	65.6	13.1	101.3	52.06
20	4	0.38	18.8	10.0	64.5	12.9	88.2	46.78
21	4	0.33	14.6	8.1	51.2	10.2	75.3	40.91
21	5	0.35	3.8	1.9	11.0	1.7	65.3	40.91
22	5	0.34	19.5	10.0	56.5	8.5	63.5	35.18
23	5	0.33	19.0	10.0	55.8	8.4	55.0	31.92
24	5	0.32	18.4	10.0	55.1	8.3	46.5	28.27
25	5	0.30	17.9	10.0	54.3	8.2	38.1	24.17
26	5	0.29	17.3	10.0	53.3	8.0	29.9	19.6
27	5	0.27	4.2	2.5	13.1	2.0	21.8	14.50
27	6	0.27	14.1	7.5	34.7	2.8	20.3	14.50
28	6	0.26	18.3	10.0	45.7	3.7	17.3	11.91
29	6	0.26	18.0	10.0	45.4	3.7	13.6	10.00
30	6	0.25	17.7	10.0	44.9	3.6	9.9	7.83
31	6	0.24	17.3	10.0	44.4	3.6	6.3	5.39
32	6	0.23	14.6	8.7	37.9	3.0	2.7	2.63

particular push back is completed, then the process continues. The stockpile would be considered as a new push back.

Alternative B

In this alternative if the profit from the ore in the stockpile is greater than that for ore in the push back, the mill is fed with material exclusively coming from the stockpile.

The material to feed the mill is sent either from the stockpile or from the push back, but not from both places simultaneously.

Alternative C

In this alternative, material is not sent to the mill from the stockpile during the mining of the pit. Material is stockpiled until all the material in the pit is exhausted. Only at that moment does the stockpile start to work as if it were a new push back.

The three stockpile alternatives will now be applied to the earlier example. The cost involved with the stockpile is assumed to be $0.225/ton. This is 45% of the original mining cost. This incremental cost is $0.20/ton to cover material re-handling and $0.025/ton for increased pit supervision, sampling, etc. The results from alternative A are given in Table 6.40. As can be seen the total profit is significantly higher than the no stockpile alternative. The NPV is only slightly higher because the major contribution occurs late in the life of the property. Table 6.41 is a year-by-year breakdown of the stockpile content. This is theoretical since one clearly would not keep so many separate grade categories.

The results from alternative B are shown in Table 6.42. As can be seen the NPV and the undiscounted profits are better than without stockpiling, but not as good as for alternative A. This is because the stockpile only starts to send ore to the mill in year 28. In the previous alternative, the first time that the stockpile sends ore to the mill is in year 15. Moreover, using material from the stockpile in years 28 through 35 means that almost all of the material in the stockpile is used. At the end of year 40, mining the remaining material in the stockpile is not economically profitable even though there is material in the stockpile between the grades of 0.23% and 0.26%.

Alternative C (Table 6.43) has a better net present value and a better undiscounted profit than without stockpiling. This alternative also has a slightly higher net present value than alternative B, but smaller than alternative A. From an operational point of view, this alternative has several advantages. It is not necessary to blend material as in the production schedule generated by alternative A or to stop the mine production for 8 years and then start to produce again, as in alternative B.

The stockpile option should always be considered in open pit mine planning. The potential contribution to the NPV depends upon each particular case. It is particularly dependent upon the spread between the highest and the lowest grades in the project cutoff grade strategy. If that spread is significant, the stockpile alternative should be considered for the project.

6.9 PRODUCTION SCHEDULING

6.9.1 *Introduction*

Production scheduling is a very important part of the mining process. This will be demonstrated through a simple example. Consider the property shown in Figure 6.28 in which 10 ore blocks are overlain by 10 waste blocks.

A production rate of 5 blocks per year (irrespective of whether the blocks are ore or waste) will be assumed. The net value for an ore block is $2 and the cost of removing the waste is $1/block. The total cost involved in waste removal would be $10 and the ore value is $20.

If both ore and waste could be mined instantaneously, the net present value would be $10. However due to practical constraints, they cannot. Therefore a number of scheduling scenarios must be considered.

Table 6.40. Final schedule for the alternative A stockpile (Schellman, 1989).

Year	Pit increment	Cutoff grade (% Cu)	Quantity mined (10^6 tons)	Quantity concentrated (10^6 tons)	Quantity refined (10^6 tons)	Profit (10^6 \$)	Total profit (10^6 \$)	Net present value (10^6 \$)
1	1	0.50	17.8	10.0	89.8	26.0	528.4	150.56
2	1	0.50	17.8	10.0	89.8	26.0	502.4	147.13
3	1	0.50	17.8	10.0	89.8	26.0	476.4	143.19
4	1	0.50	17.8	10.0	89.8	26.0	450.4	138.66
5	1	0.50	17.8	10.0	89.8	26.0	424.4	133.45
6	1	0.50	11.2	6.3	56.6	16.4	398.4	127.46
6	2	0.53	7.4	3.7	31.8	8.5	382.0	127.46
7	2	0.53	20.0	10.0	85.9	23.0	373.5	121.69
8	2	0.53	20.0	10.0	85.9	23.0	350.5	116.97
9	2	0.53	20.0	10.0	85.9	23.0	327.5	111.55
10	2	0.53	20.0	10.0	85.9	23.0	304.5	105.31
11	2	0.53	12.8	6.4	55.0	14.7	281.5	98.14
11	3	0.47	7.2	3.6	27.4	6.5	266.8	98.14
12	3	0.47	20.0	10.0	76.1	18.0	260.3	91.66
13	3	0.47	20.0	10.0	76.1	18.0	242.3	87.36
14	3	0.47	20.0	10.0	76.1	18.0	224.3	82.42
15	***	0.47	1.0	10.0	4.6	1.2	206.3	76.73
15	3	0.47	18.7	9.0	71.5	17.2	205.1	76.73
16	3	0.47	13.1	6.4	48.8	11.4	187.9	69.82
16	4	0.41	6.9	3.5	23.1	4.6	176.5	69.82
17	4	0.41	20.0	10.0	66.5	13.2	171.9	64.28
18	4	0.41	20.0	10.0	66.5	13.2	158.7	60.70
19	4	0.41	20.0	10.0	66.5	13.2	145.5	56.57
20	***	0.41	2.2	2.2	9.2	2.2	132.3	51.83
20	4	0.41	17.3	7.8	57.0	11.7	130.1	51.83
21	4	0.41	13.6	6.6	43.8	8.4	118.4	45.64
21	5	0.35	6.4	3.2	18.3	2.8	110.0	45.64
22	5	0.35	20.0	10.0	57.2	8.6	107.2	41.35
23	5	0.35	20.0	10.0	57.2	8.6	98.6	38.94
24	5	0.35	20.0	10.0	57.2	8.6	90.0	36.17
25	***	0.35	3.7	3.7	15.4	3.7	81.4	32.98
25	5	0.35	15.6	6.3	42.5	6.6	77.7	32.98
26	5	0.35	14.7	7.2	40.5	5.6	71.1	27.60
26	6	0.29	5.3	2.6	12.6	1.0	65.5	27.60
27	6	0.29	20.0	10.0	47.6	3.8	64.5	25.08
28	6	0.29	20.0	10.0	47.6	3.8	60.7	25.03
29	6	0.29	20.0	10.0	47.6	3.8	56.9	24.98
30	***	0.29	5.5	5.5	18.8	3.6	53.1	24.91
30	6	0.29	13.6	4.5	27.6	1.6	49.5	24.91
31	6	0.29	16.0	7.8	36.7	2.5	47.9	23.50
31	***	0.31	2.2	2.2	6.8	0.8	45.6	23.50
32	***	0.31	10.0	10.0	36.5	6.0	44.6	23.79
33	***	0.30	10.0	10.0	36.5	6.0	38.6	21.36
34	***	0.29	10.0	10.0	35.1	5.3	32.6	18.57
35	***	0.28	10.0	10.0	34.5	5.0	27.3	16.05
36	***	0.27	10.0	10.0	33.7	4.6	22.3	13.47
37	***	0.27	10.0	10.0	32.9	4.2	17.7	10.90
38	***	0.26	10.0	10.0	30.8	3.1	13.5	8.35
39	***	0.25	10.0	10.0	29.0	2.3	10.4	6.45
40	***	0.24	10.0	10.0	27.2	1.4	8.1	5.17
41	***	0.23	10.0	10.0	27.2	1.4	6.7	4.58
42	***	0.23	10.0	10.0	27.2	1.4	5.3	3.91
43	***	0.23	10.0	10.0	28.2	1.9	3.9	3.13
44	***	0.23	8.2	8.2	24.2	2.0	2.0	1.7

*** Mineral from stockpile.

Table 6.41. Stockpile grade and tonnage (expressed in 10^6 tons) distribution by year for alternative A (Schellman, 1989).

STOCKPILE GRADE DISTRIBUTION
YEAR 1

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	.5
.250	.300	.275	.8
.300	.350	.325	.7
.350	.400	.375	.7
.400	.450	.425	.7
.450	.500	.475	.7
		TOTAL:	4.1

STOCKPILE GRADE DISTRIBUTION
YEAR 2

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	.9
.250	.300	.275	1.5
.300	.350	.325	1.5
.350	.400	.375	1.5
.400	.450	.425	1.4
.450	.500	.475	1.3
		TOTAL:	8.1

STOCKPILE GRADE DISTRIBUTION
YEAR 3

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	1.4
.250	.300	.275	2.3
.300	.350	.325	2.2
.350	.400	.375	2.2
.400	.450	.425	2.1
.450	.500	.475	2.0
		TOTAL:	12.2

STOCKPILE GRADE DISTRIBUTION
YEAR 4

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	1.9
.250	.300	.275	3.1
.300	.350	.325	3.0
.350	.400	.375	2.9
.400	.450	.425	2.8
.450	.500	.475	2.7
		TOTAL:	16.3

STOCKPILE GRADE DISTRIBUTION
YEAR 5

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	2.3
.250	.300	.275	3.8
.300	.350	.325	3.7
.350	.400	.375	3.6
.400	.450	.425	3.5
.450	.500	.475	3.4
		TOTAL:	20.4

STOCKPILE GRADE DISTRIBUTION
YEAR 6

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	2.6
.250	.300	.275	4.3
.300	.350	.325	4.2
.350	.400	.375	4.1
.400	.450	.425	3.9
.450	.500	.475	3.8
		TOTAL:	22.9

STOCKPILE GRADE DISTRIBUTION
YEAR 7

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	3.2
.250	.300	.275	5.2
.300	.350	.325	5.1
.350	.400	.375	5.0
.400	.450	.425	4.8
.450	.500	.475	4.6
.500	.528	.514	.4
		TOTAL:	28.4

STOCKPILE GRADE DISTRIBUTION
YEAR 8

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	3.8
.250	.300	.275	6.2
.300	.350	.325	6.0
.350	.400	.375	5.9
.400	.450	.425	5.6
.450	.500	.475	5.4
.500	.528	.514	.9
		TOTAL:	33.8

STOCKPILE GRADE DISTRIBUTION
YEAR 9

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	4.4
.250	.300	.275	7.1
.300	.350	.325	6.9
.350	.400	.375	6.7
.400	.450	.425	6.5
.450	.500	.475	6.3
.500	.528	.514	1.3
		TOTAL:	39.2

STOCKPILE GRADE DISTRIBUTION
YEAR 10

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	5.0
.250	.300	.275	8.1
.300	.350	.325	7.8
.350	.400	.375	7.6
.400	.450	.425	7.3
.450	.500	.475	7.1
.500	.528	.514	1.8
		TOTAL:	44.6

STOCKPILE GRADE DISTRIBUTION
YEAR 11

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	5.4
.250	.300	.275	8.7
.300	.350	.325	8.4
.350	.400	.375	8.2
.400	.450	.425	7.9
.450	.469	.459	2.9
.469	.500	.484	4.7
.500	.528	.514	2.0
		TOTAL:	48.1

STOCKPILE GRADE DISTRIBUTION
YEAR 12

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	6.0
.250	.300	.275	9.7
.300	.350	.325	9.3
.350	.400	.375	9.1
.400	.450	.425	8.8
.450	.469	.459	3.2
.469	.500	.484	4.7
.500	.528	.514	2.0
		TOTAL:	53.0

Table 6.41. (Continued).

STOCKPILE GRADE DISTRIBUTION YEAR 13

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	6.7
.250	.300	.275	10.8
.300	.350	.325	10.3
.350	.400	.375	10.1
.400	.450	.425	9.7
.450	.469	.459	3.6
.469	.500	.484	4.7
.500	.528	.514	2.0
		TOTAL:	57.9

STOCKPILE GRADE DISTRIBUTION YEAR 14

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	7.3
.250	.300	.275	11.8
.300	.350	.325	11.3
.350	.400	.375	11.0
.400	.450	.425	10.6
.450	.469	.459	3.9
.469	.500	.484	4.7
.500	.528	.514	2.0
		TOTAL:	62.7

STOCKPILE GRADE DISTRIBUTION YEAR 15

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	7.9
.250	.300	.275	12.9
.300	.350	.325	12.3
.350	.400	.375	11.9
.400	.450	.425	11.5
.450	.469	.459	4.2
.469	.500	.484	4.6
.500	.528	.514	2.2
		TOTAL:	67.5

STOCKPILE GRADE DISTRIBUTION YEAR 16

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	8.4
.250	.300	.275	13.6
.300	.350	.325	12.9
.350	.400	.375	12.5
.400	.410	.405	2.5
.410	.450	.430	9.7
.450	.469	.459	4.4
.469	.500	.484	4.6
.500	.528	.514	2.2
		TOTAL:	70.7

STOCKPILE GRADE DISTRIBUTION YEAR 17

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	9.1
.250	.300	.275	14.7
.300	.350	.325	14.0
.350	.400	.375	13.6
.400	.410	.405	2.7
.410	.450	.430	9.7
.450	.469	.459	4.4
.469	.500	.484	4.6
.500	.528	.514	2.2
		TOTAL:	74.9

STOCKPILE GRADE DISTRIBUTION YEAR 18

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	9.8
.250	.300	.275	15.8
.300	.350	.325	15.1
.350	.400	.375	14.7
.400	.410	.405	2.9
.410	.450	.430	9.7
.450	.469	.459	4.4
.469	.500	.484	4.6
.500	.528	.514	2.2
		TOTAL:	79.1

STOCKPILE GRADE DISTRIBUTION YEAR 19

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	10.5
.250	.300	.275	16.9
.300	.350	.325	16.2
.350	.400	.375	15.7
.400	.410	.405	3.1
.410	.450	.430	9.7
.450	.469	.459	4.4
.469	.500	.484	4.6
.500	.528	.514	2.2
		TOTAL:	83.3

STOCKPILE GRADE DISTRIBUTION YEAR 20

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	11.2
.250	.300	.275	18.0
.300	.350	.325	17.3
.350	.400	.375	16.7
.400	.410	.405	3.3
.410	.450	.430	9.4
.450	.469	.459	4.3
.469	.500	.484	5.1
.500	.528	.514	2.0
		TOTAL:	87.2

STOCKPILE GRADE DISTRIBUTION YEAR 21

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	11.7
.250	.300	.275	18.8
.300	.350	.325	18.0
.350	.400	.375	17.5
.400	.410	.405	3.4
.410	.450	.430	9.4
.450	.469	.459	4.3
.469	.500	.484	5.1
.500	.528	.514	2.0
		TOTAL:	90.2

STOCKPILE GRADE DISTRIBUTION YEAR 22

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	12.5
.250	.300	.275	20.1
.300	.350	.325	19.3
.350	.400	.375	17.5
.400	.410	.405	3.4
.410	.450	.430	9.4
.450	.469	.459	4.3
.469	.500	.484	5.1
.500	.528	.514	2.0
		TOTAL:	93.5

STOCKPILE GRADE DISTRIBUTION YEAR 23

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	13.3
.250	.300	.275	21.3
.300	.350	.325	20.5
.350	.400	.375	17.5
.400	.410	.405	3.4
.410	.450	.430	9.4
.450	.469	.459	4.3
.469	.500	.484	5.1
.500	.528	.514	2.0
		TOTAL:	96.8

STOCKPILE GRADE DISTRIBUTION YEAR 24

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	14.1
.250	.300	.275	22.6
.300	.350	.325	21.7
.350	.400	.375	17.5
.400	.410	.405	3.4
.410	.450	.430	9.4
.450	.469	.459	4.3
.469	.500	.484	5.1
.500	.528	.514	2.0
		TOTAL:	100.1

Table 6.41. (Continued).

STOCKPILE GRADE DISTRIBUTION
YEAR 25

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	14.9
.250	.300	.275	23.8
.300	.350	.325	22.9
.350	.400	.375	18.5
.400	.410	.405	3.6
.410	.450	.430	8.5
.450	.469	.459	4.4
.469	.500	.484	4.8
.500	.528	.514	1.6
		TOTAL:	103.0

STOCKPILE GRADE DISTRIBUTION
YEAR 26

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	15.5
.250	.293	.272	21.4
.293	.300	.297	3.4
.300	.350	.325	23.8
.350	.400	.375	18.5
.400	.410	.405	3.6
.410	.450	.430	8.5
.450	.469	.459	4.4
.469	.500	.484	4.8
.500	.528	.514	1.6
		TOTAL:	105.5

STOCKPILE GRADE DISTRIBUTION
YEAR 27

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	16.4
.250	.293	.272	22.6
.293	.300	.297	3.4
.300	.350	.325	23.8
.350	.400	.375	18.5
.400	.410	.405	3.6
.410	.450	.430	8.5
.450	.469	.459	4.4
.469	.500	.484	4.8
.500	.528	.514	1.6
		TOTAL:	107.7

STOCKPILE GRADE DISTRIBUTION
YEAR 28

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	17.3
.250	.293	.272	23.9
.293	.300	.297	3.4
.300	.350	.325	23.8
.350	.400	.375	18.5
.400	.410	.405	3.6
.410	.450	.430	8.5
.450	.469	.459	4.4
.469	.500	.484	4.8
.500	.528	.514	1.6
		TOTAL:	109.8

STOCKPILE GRADE DISTRIBUTION
YEAR 29

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	18.3
.250	.293	.272	25.2
.293	.300	.297	3.4
.300	.350	.325	23.8
.350	.400	.375	18.5
.400	.410	.405	3.6
.410	.450	.430	8.5
.450	.469	.459	4.4
.469	.500	.484	4.8
.500	.528	.514	1.6
		TOTAL:	112.0

STOCKPILE GRADE DISTRIBUTION
YEAR 30

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.1
.250	.293	.272	26.3
.293	.300	.297	3.3
.300	.350	.325	23.1
.350	.400	.375	18.2
.400	.410	.405	3.5
.410	.450	.430	8.6
.450	.469	.459	4.9
.469	.500	.484	5.0
.500	.528	.514	1.5
		TOTAL:	113.6

STOCKPILE GRADE DISTRIBUTION
YEAR 31

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.9
.250	.293	.272	22.6
.293	.300	.297	3.3
.300	.350	.325	23.1
.350	.400	.375	18.2
.400	.410	.405	3.5
.410	.450	.430	8.6
.450	.469	.459	4.9
.469	.500	.484	5.0
.500	.528	.514	1.5
.500	.528	.514	4.4
.528	.550	.539	4.8
		TOTAL:	124.5

STOCKPILE GRADE DISTRIBUTION
YEAR 32

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.5
.250	.284	.267	21.3
.284	.293	.289	5.5
.293	.300	.297	3.2
.300	.350	.325	22.7
.350	.400	.375	17.9
.400	.410	.405	3.5
.410	.450	.430	8.5
.450	.469	.459	4.8
.469	.500	.484	4.9
.500	.528	.514	1.4
.500	.528	.514	4.3
.528	.550	.539	4.7
.550	.600	.575	1.6
		TOTAL:	123.9

Table 6.41. (Continued).

STOCKPILE GRADE DISTRIBUTION
YEAR 33

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.5
.250	.278	.264	17.6
.278	.284	.281	3.8
.284	.293	.289	5.5
.293	.300	.297	3.2
.300	.350	.325	19.7
.350	.400	.375	15.5
.400	.410	.405	3.0
.410	.450	.430	7.3
.450	.469	.459	4.1
.469	.500	.484	4.3
.500	.528	.514	1.3
.500	.528	.514	3.7
.528	.550	.539	4.1
.550	.600	.575	1.4
		TOTAL:	113.9

STOCKPILE GRADE DISTRIBUTION
YEAR 34

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.5
.250	.271	.261	13.3
.271	.278	.275	4.2
.278	.284	.281	3.8
.284	.293	.289	5.5
.293	.300	.297	3.2
.300	.350	.325	16.6
.350	.400	.375	13.0
.400	.410	.405	2.5
.410	.450	.430	6.2
.450	.469	.459	3.5
.469	.500	.484	3.6
.500	.528	.514	1.1
.500	.528	.514	3.1
.528	.550	.539	3.4
.550	.600	.575	1.2
		TOTAL:	103.9

STOCKPILE GRADE DISTRIBUTION
YEAR 35

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.5
.250	.265	.258	9.4
.265	.271	.268	4.0
.271	.278	.275	4.2
.278	.284	.281	3.8
.284	.293	.289	4.7
.293	.300	.297	2.7
.300	.350	.325	14.0
.350	.400	.375	11.0
.400	.410	.405	2.1
.410	.450	.430	5.2
.450	.469	.459	2.9
.469	.500	.484	3.0
.500	.528	.514	.9
.500	.528	.514	2.6
.528	.550	.539	2.9
.550	.600	.575	1.0
		TOTAL:	93.9

STOCKPILE GRADE DISTRIBUTION
YEAR 36

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.5
.250	.265	.258	9.4
.265	.271	.268	4.0
.271	.278	.275	4.2
.278	.284	.281	3.1
.284	.293	.289	3.8
.293	.300	.297	2.3
.300	.350	.325	11.5
.350	.400	.375	9.1
.400	.410	.405	1.8
.410	.450	.430	4.3
.450	.469	.459	2.4
.469	.500	.484	2.5
.500	.528	.514	.7
.500	.528	.514	2.2
.528	.550	.539	2.4
.550	.600	.575	.8
		TOTAL:	83.9

STOCKPILE GRADE DISTRIBUTION
YEAR 37

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.250	.240	19.5
.250	.261	.256	7.2
.261	.265	.263	2.2
.265	.271	.268	4.0
.271	.278	.275	3.4
.278	.284	.281	2.5
.284	.293	.289	3.1
.293	.300	.297	1.8
.300	.350	.325	9.3
.350	.400	.375	7.3
.400	.410	.405	1.4
.410	.450	.430	3.5
.450	.469	.459	2.0
.469	.500	.484	2.0
.500	.528	.514	.6
.500	.528	.514	1.8
.528	.550	.539	1.9
.550	.600	.575	.7
		TOTAL:	73.9

STOCKPILE GRADE DISTRIBUTION
YEAR 38

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	15.0
.245	.250	.248	4.5
.250	.261	.256	7.2
.261	.265	.263	1.7
.265	.271	.268	3.1
.271	.278	.275	2.7
.278	.284	.281	2.0
.284	.293	.289	2.4
.293	.300	.297	1.4
.300	.350	.325	7.3
.350	.400	.375	5.7
.400	.410	.405	1.1
.410	.450	.430	2.7
.450	.469	.459	1.5
.469	.500	.484	1.6
.500	.528	.514	.5
.500	.528	.514	1.4
.528	.550	.539	1.5
.550	.600	.575	.5
		TOTAL:	63.9

Table 6.41. (Continued).

STOCKPILE GRADE DISTRIBUTION
YEAR 39

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	15.0
.245	.250	.248	4.5
.250	.261	.256	5.5
.261	.265	.263	1.3
.265	.271	.268	2.4
.271	.278	.275	2.1
.278	.284	.281	1.5
.284	.293	.289	1.9
.293	.300	.297	1.1
.300	.350	.325	5.7
.350	.400	.375	4.4
.400	.410	.405	.9
.410	.450	.430	2.1
.450	.469	.459	1.2
.469	.500	.484	1.2
.500	.528	.514	.4
.500	.528	.514	1.1
.528	.550	.539	1.2
.550	.600	.575	.4
		TOTAL:	53.9

STOCKPILE GRADE DISTRIBUTION
YEAR 40

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	12.3
.245	.250	.248	3.6
.250	.261	.256	4.5
.261	.265	.263	1.1
.265	.271	.268	2.0
.271	.278	.275	1.7
.278	.284	.281	1.2
.284	.293	.289	1.5
.293	.300	.297	.9
.300	.350	.325	4.6
.350	.400	.375	3.6
.400	.410	.405	.7
.410	.450	.430	1.7
.450	.469	.459	1.0
.469	.500	.484	1.0
.500	.528	.514	.3
.500	.528	.514	.9
.528	.550	.539	1.0
.550	.600	.575	.3
		TOTAL:	43.9

STOCKPILE GRADE DISTRIBUTION
YEAR 41

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	9.5
.245	.250	.248	2.8
.250	.261	.256	3.5
.261	.265	.263	.8
.265	.271	.268	1.5
.271	.278	.275	1.3
.278	.284	.281	1.0
.284	.293	.289	1.2
.293	.300	.297	.7
.300	.350	.325	3.6
.350	.400	.375	2.8
.400	.410	.405	.5
.410	.450	.430	1.3
.450	.469	.459	.8
.469	.500	.484	.8
.500	.528	.514	.2
.500	.528	.514	.7
.528	.550	.539	.7
.550	.600	.575	.3
		TOTAL:	33.9

STOCKPILE GRADE DISTRIBUTION
YEAR 42

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	6.7
.245	.250	.248	2.0
.250	.261	.256	2.5
.261	.265	.263	.6
.265	.271	.268	1.1
.271	.278	.275	.9
.278	.284	.281	.7
.284	.293	.289	.8
.293	.300	.297	.5
.300	.350	.325	2.5
.350	.400	.375	2.0
.400	.410	.405	.4
.410	.450	.430	.9
.450	.469	.459	.5
.469	.500	.484	.5
.500	.528	.514	.2
.500	.528	.514	.5
.528	.550	.539	.5
.550	.600	.575	.2
		TOTAL:	24.0

STOCKPILE GRADE DISTRIBUTION
YEAR 43

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	3.9
.245	.250	.248	1.2
.250	.261	.256	1.4
.261	.265	.263	.3
.265	.271	.268	.6
.271	.278	.275	.5
.278	.284	.281	.4
.284	.293	.289	.5
.293	.300	.297	.3
.300	.350	.325	1.5
.350	.400	.375	1.1
.400	.410	.405	.2
.410	.450	.430	.5
.450	.469	.459	.3
.469	.500	.484	.3
.500	.528	.514	.1
.500	.528	.514	.3
.528	.550	.539	.3
.550	.600	.575	.1
.431	.444	.438	2.9
		TOTAL:	16.8

STOCKPILE GRADE DISTRIBUTION
YEAR 44

GRADE CATEGORIES %		AVRG	TONNAGE
.230	.245	.238	1.6
.245	.250	.248	.5
.250	.261	.256	.6
.261	.265	.263	.1
.265	.271	.268	.3
.271	.278	.275	.2
.278	.284	.281	.2
.284	.293	.289	.2
.293	.300	.297	.1
.300	.350	.325	.6
.350	.400	.375	.5
.400	.410	.405	.1
.410	.450	.430	.2
.450	.469	.459	.1
.469	.500	.484	.1
.500	.528	.514	.0
.500	.528	.514	.1
.528	.550	.539	.1
.550	.600	.575	.0
.431	.444	.438	2.9
		TOTAL:	8.5

Table 6.42. Final schedule for the alternative B stockpile (Shellman, 1989).

Year	Pit increment	Cutoff grade (% Cu)	Quantity mined (10^6 tons)	Quantity concentrated (10^6 tons)	Quantity refined (10^6 tons)	Profit (10^6 \$)	Total profit (10^6 \$)	Net present value (10^6 \$)
1	1	0.50	17.8	10.0	89.8	26.0	515.5	150.41
2	1	0.50	17.8	10.0	89.8	26.0	489.5	146.96
3	1	0.50	17.8	10.0	89.8	26.0	463.5	142.99
4	1	0.50	17.8	10.0	89.8	26.0	437.5	138.42
5	1	0.50	17.8	10.0	89.8	26.0	411.5	133.18
6	1	0.50	11.2	6.3	56.6	16.4	385.5	127.15
6	2	0.53	7.4	3.7	31.8	8.5	369.1	127.15
7	2	0.53	20.0	10.0	85.9	23.0	360.6	121.34
8	2	0.53	20.0	10.0	85.9	23.0	337.6	116.57
9	2	0.53	20.0	10.0	85.9	23.0	314.6	116.57
10	2	0.53	20.0	10.0	85.9	23.0	291.6	104.78
11	2	0.53	12.8	6.4	55.0	14.7	268.6	97.53
11	3	0.47	7.2	3.6	27.4	6.5	253.9	97.53
12	3	0.47	20.0	10.0	76.1	18.0	247.4	90.96
13	3	0.47	20.0	10.0	76.1	18.0	229.4	86.56
14	3	0.47	20.0	10.0	76.1	18.0	211.4	81.49
15	3	0.47	20.0	10.0	76.1	18.0	193.0	75.67
16	3	0.45	12.8	6.6	49.6	11.8	175.4	68.97
16	4	0.45	6.0	3.4	22.5	4.5	163.6	68.97
17	4	0.41	20.0	10.0	66.5	13.2	159.1	63.06
18	4	0.41	20.0	10.0	66.5	13.2	145.9	59.29
19	4	0.41	19.6	10.0	65.9	13.1	132.7	54.95
20	4	0.39	19.2	10.0	65.2	13.0	119.6	50.07
21	4	0.37	14.5	7.8	50.1	10.0	106.6	44.57
21	5	0.35	4.4	2.2	12.5	1.9	69.6	44.57
22	5	0.35	20.0	10.0	57.2	8.6	94.7	39.37
23	5	0.34	19.7	10.0	56.8	8.6	86.1	36.66
24	5	0.33	19.2	10.0	56.2	8.5	77.5	33.59
25	5	0.32	18.7	10.0	55.5	8.4	69.0	30.14
26	5	0.31	17.8	10.0	54.1	8.2	60.6	26.27
27	5	0.3	0.3	0.2	1.00	0.2	52.4	22.04
27	6	0.27	19.6	9.8	46.7	3.7	52.2	22.04
28	***	0.26	10.0	10.0	32.4	4.0	48.5	21.46
29	***	0.26	10.0	10.0	32.4	4.0	44.5	20.72
30	***	0.26	10.0	10.0	32.4	4.0	40.5	19.87
31	***	0.26	10.0	10.0	32.4	4.0	40.5	19.87
32	***	0.26	10.0	10.0	32.4	4.0	32.5	17.78
33	***	0.26	10.0	10.0	32.4	4.0	28.5	16.5
34	***	0.26	10.0	10.0	32.4	4.0	24.5	15.02
35	***	0.26	10.0	10.0	32.4	4.0	20.5	13.31
36	6	0.26	18.2	10.0	45.6	3.7	16.5	11.36
37	6	0.25	17.8	10.0	45.6	3.6	12.8	9.37
38	6	0.25	17.5	10.0	44.7	3.6	9.2	7.14
39	6	0.24	16.4	10.0	43.3	3.4	5.6	4.62
40	6	0.23	10.5	6.4	27.7	2.2	2.2	1.89

*** Mineral from stockpile.

Table 6.43. Final schedule for the alternative B stockpile (Shellman, 1989).

Year	Pit increment	Cutoff grade (% Cu)	Quantity mined (10^6 tons)	Quantity concentrated (10^6 tons)	Quantity refined (10^6 tons)	Profit (10^6 \$)	Total profit (10^6 \$)	Net present value (10^6 \$)
1	1	0.5	17.8	10	89.8	26	529.6	150.46
2	1	0.5	17.8	10	89.8	26	503.6	147.01
3	1	0.5	17.8	10	89.8	26	477.6	143.06
4	1	0.5	17.8	10	89.8	26	451.6	138.5
5	1	0.5	17.8	10	89.8	26	425.6	133.27
6	1	0.5	11.2	6.3	56.6	16.4	399.6	127.25
6	2	0.53	7.4	3.7	31.8	8.5	383.2	127.25
7	2	0.53	20	10	85.9	23	374.7	121.45
8	2	0.53	20	10	85.9	23	351.7	116.7
9	2	0.53	20	10	85.9	23	328.7	111.24
10	2	0.53	20	10	85.9	23	305.7	104.95
11	2	0.53	12.8	6.4	55	14.7	282.7	97.73
11	3	0.47	7.2	3.6	27.4	6.5	268	97.73
12	3	0.47	20	10	76.1	18	261.5	91.19
13	3	0.47	20	10	76.1	18	243.5	86.82
14	3	0.47	20	10	76.1	18	225.5	81.79
15	3	0.47	20	10	76.1	18	207.5	76.01
16	3	0.45	12.8	6.7	49.8	11.8	189.5	69.36
16	4	0.41	6.7	3.3	22.2	4.4	177.7	69.36
17	4	0.41	20	10	66.5	13.2	173.2	63.52
18	4	0.41	20	10	66.5	13.2	160.1	59.82
19	4	0.4	19.5	10	65.6	13.1	146.9	55.57
20	4	0.38	18.8	10	64.6	12.9	133.8	50.82
21	4	0.36	15.1	8.5	53.4	10.7	120.9	45.55
21	5	0.35	3.00	1.5	8.6	1.3	110.2	45.55
22	5	0.34	19.5	10	56.5	8.8	108.9	40.43
23	5	0.33	19	10	55.9	8.4	100.4	37.95
24	5	0.32	18.5	10	55.1	8.3	92	35.2
25	5	0.3	17.8	10	54.1	8.2	83.7	32.14
26	5	0.29	17.3	10	53.4	8.0	75.5	28.79
27	5	0.27	5.0	3.0	15.8	2.4	67.5	25.07
27	6	0.27	31.1	7	32.4	2.6	65.1	25.07
28	6	0.26	18.3	10	45.8	3.7	62.5	23.85
29	6	0.26	17.8	10	45	3.6	58.8	23.72
30	6	0.26	17.7	10	44.9	3.6	58.8	23.72
31	6	0.24	17.1	10.0	44.2	3.5	51.6	23.57
32	6	0.23	15.9	9.7	42.0	3.3	48.1	23.57
32	***	0.30	0.3	0.3	0.9	0.1	44.8	23.57
33	***	0.30	10.0	10.0	36.2	5.8	44.7	23.69
34	***	0.30	10.0	10.0	35.4	5.4	38.9	21.41
35	***	0.29	10.0	10.0	34.9	5.2	33.5	19.18
36	***	0.28	10.0	10.0	33.9	4.7	28.3	16.87
37	***	0.27	10.0	10.0	34.4	4.9	23.6	14.70
38	***	0.27	10.0	10.0	33.3	4.4	18.7	11.97
39	***	0.26	10.0	10.0	31.7	3.6	14.3	9.35
40	***	0.25	10.0	10.0	29.8	2.6	10.7	7.14
41	***	0.24	10.0	10.0	27.6	1.6	8.1	5.58
42	***	0.23	10.0	10.0	27.9	1.7	6.5	4.86
43	***	0.23	10.0	10.0	28.5	2.0	4.8	3.87
44	***	0.23	9.7	9.7	29.3	2.8	2.8	2.42

*** Mineral from stockpile.

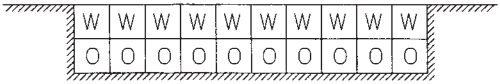


Figure 6.28. Simple sequencing example.

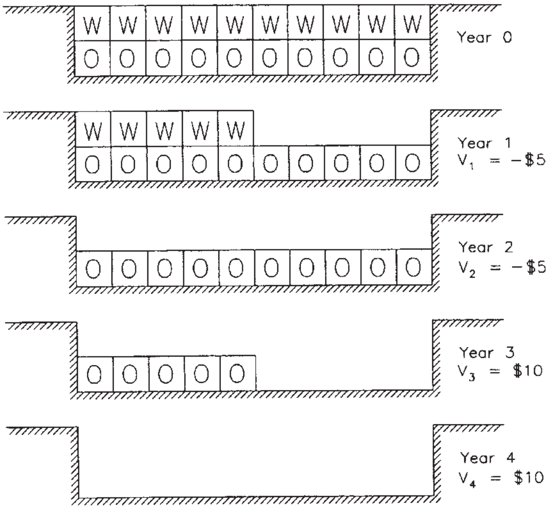


Figure 6.29. Mining sequence – scenario 1.

Scenario 1. Removal of waste followed by ore mining

For operational simplicity it would be best if all of the waste could be stripped (removed) first followed later by ore mining. This is shown in Figure 6.29.

The net present value for this sequence assuming an interest rate of 10% is

$$\begin{aligned}\text{NPV} &= \frac{-\$5}{(1.10)^1} + \frac{-\$5}{(1.10)^2} + \frac{\$10}{(1.10)^3} + \frac{\$10}{(1.10)^4}\\ &= -\$4.55 - \$4.13 + \$7.51 + \$6.83 = \$5.66\end{aligned}$$

Scenario 2. One year of pre-stripping followed by both ore (3 blocks/year) and waste (2 blocks/year) mining

This alternative would require mining 5 blocks of waste in year 1. In years 2 and 3 three blocks of ore would be mined for every two blocks of waste. The final year would have 1 block of waste and 4 blocks of ore. The sequencing is shown in Figure 6.30.

The net present value is now

$$\begin{aligned}\text{NPV} &= \frac{-\$5}{(1.10)^1} + \frac{\$4}{(1.10)^2} + \frac{\$4}{(1.10)^3} + \frac{\$7}{(1.10)^4}\\ &= -\$4.54 + \$3.31 + \$3.01 + \$4.78 = \$6.56\end{aligned}$$

Scenario 3. Mining of waste is maintained one block ahead of ore

Comparing scenarios 1 and 2, there was an improvement in the net present value when the time lag between stripping and mining was shortened. In this third scenario, the stripping

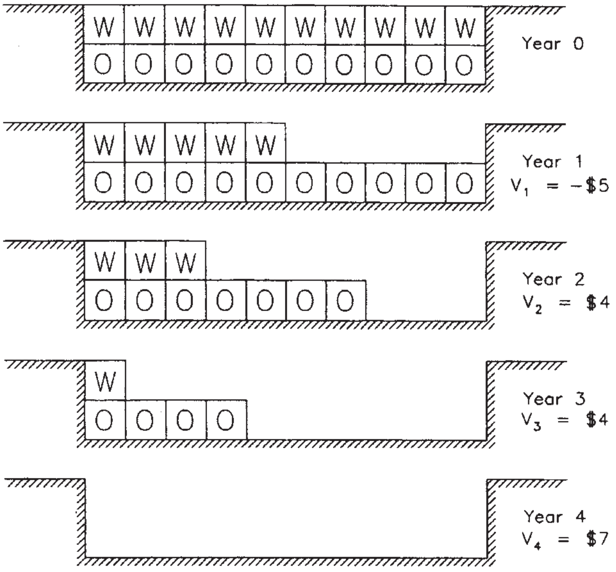


Figure 6.30. Mining sequence – scenario 2.

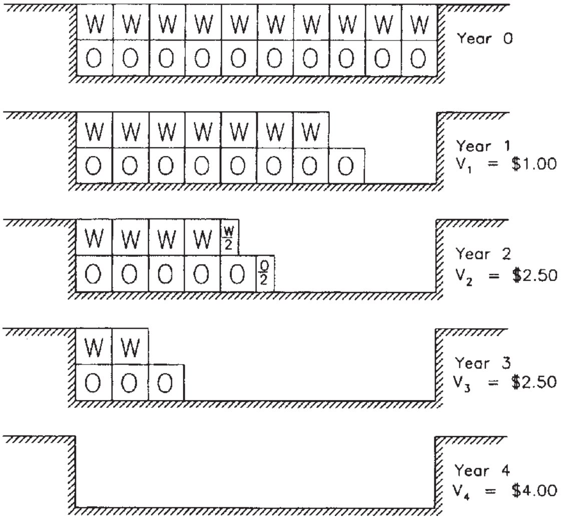


Figure 6.31. Mining sequence – scenario 3.

will be kept only one block ahead of ore mining, to make the time lag even shorter. This is shown in Figure 6.31.

The net present value is

$$\begin{aligned}\text{NPV} &= \frac{\$1}{(1.10)^1} + \frac{\$2.50}{(1.10)^2} + \frac{\$2.50}{(1.10)^3} + \frac{\$4}{(1.10)^4} \\ &= \$0.91 + \$2.07 + \$1.88 + \$2.73 = \$7.59\end{aligned}$$

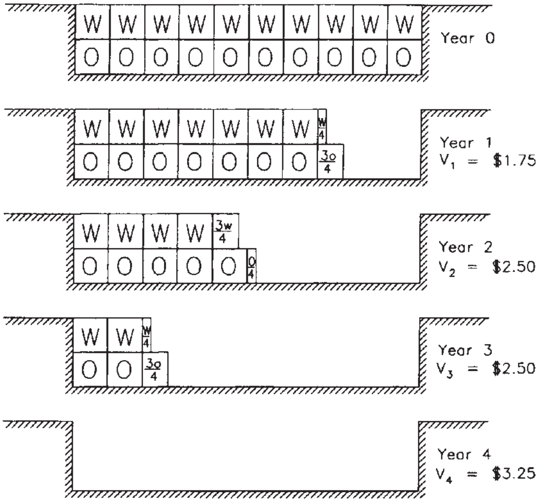


Figure 6.32. Mining sequence –scenario 4.

Scenario 4. Stripping is maintained one half block ahead of ore mining
A major improvement in the NPV was observed between scenarios 2 and 3. To explore this further, consider the situation when the stripping lead is cut to one-half block (Figure 6.32).

The NPV is

$$\begin{aligned}\text{NPV} &= \frac{\$1.75}{(1.10)^1} + \frac{\$2.50}{(1.10)^2} + \frac{\$2.50}{(1.10)^3} + \frac{\$3.25}{(1.10)^4}\\ &= \$1.59 + \$2.07 + \$1.88 + \$2.22 = \$7.76\end{aligned}$$

This would appear to be the most favorable alternative of the four scenarios. However, suppose that in reducing the stripping lead it is found that the operating costs for both ore and waste increase by $0.05/block, perhaps through lack of sufficient working space or through neglect of drilling precision because of time pressures. Hence the cost for waste removal increases to $1.05/block and the net ore revenue drops to $1.95/block.

The actual NPV is

$$\begin{aligned}\text{NPV} &= \frac{\$1.50}{(1.10)^1} + \frac{\$2.25}{(1.10)^2} + \frac{\$2.25}{(1.10)^3} + \frac{\$3.00}{(1.10)^4}\\ &= \$1.36 + \$1.86 + \$1.69 + \$2.05 + \$6.96\end{aligned}$$

In this case scenario 3 remains the most attractive.

Scenario 5. Mining rate doubled
It has been suggested that with the purchase of more equipment, the mining rate could be increased to 10 blocks per year. There would be an increase in the equipment ownership costs to be charged against both ore and waste however. The resulting values are:

$$\text{Waste cost} = \$1.10/\text{block}$$

$$\text{Ore revenue} = \$1.90/\text{block}$$

Stripping will be kept one block ahead of ore mining as in scenario 3.

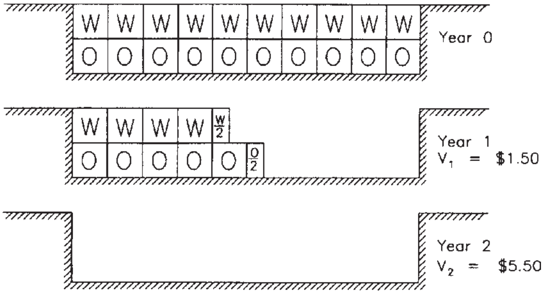


Figure 6.33. Mining sequence – scenario 5.

The scheduling is shown in Figure 6.33.

The NPV is

$$\text{NPV} = \frac{\$2.50}{(1.10)^1} + \frac{\$5.50}{(1.10)^2} = \$2.27 + \$4.55 = \$6.82$$

As can be seen scenario 3 remains the most favorable.

If there had been no additional cost then

$$\text{NPV} = \frac{\$3.50}{(1.10)^1} + \frac{\$6.50}{(1.10)^2} = \$3.18 + \$5.37 = \$8.55$$

and scenario 5 would have been the most favorable.

In summary, this very simple example has demonstrated some important aspects of production scheduling. The NPV is dependent upon

1. The time interval between stripping and ore mining. It is highest when the lead time is short. With added costs associated with shortening the lead time, however, there may or may not be an improvement in NPV.

2. The production rate. For the same unit cost, the highest NPV is achieved with the highest production rate. With added costs with increasing production rate, there may or may not be an improvement in NPV.

6.9.2 *Phase scheduling*

Several mining areas or mine phases, typically three or more, are active at any given time during the life of a mine. Of these, one or two would be in the process of being stripped, another being mined for ore and the last nearing exhaustion. This section will describe a procedure which can be used to help sequence the phases so that the desired ore stream is produced. The procedure and the illustrative example have been adapted from Mathieson (1982).

The hypothetical deposit is shown in section in Figure 6.34. The orebody, located in rock type 2, is overlain by waste (rock type 1). The planning scheduling will be described in a step-by-step manner.

1. The phases are first designed. The slope angles used are selected based upon initial geotechnical investigations. For this example it is assumed that the orebody is of uniform grade. Hence the phases A through F (shown in Figure 6.35) have been designed, subject to access constraints, to progressively mine the 'next best' ore in terms of annual stripping ratio.

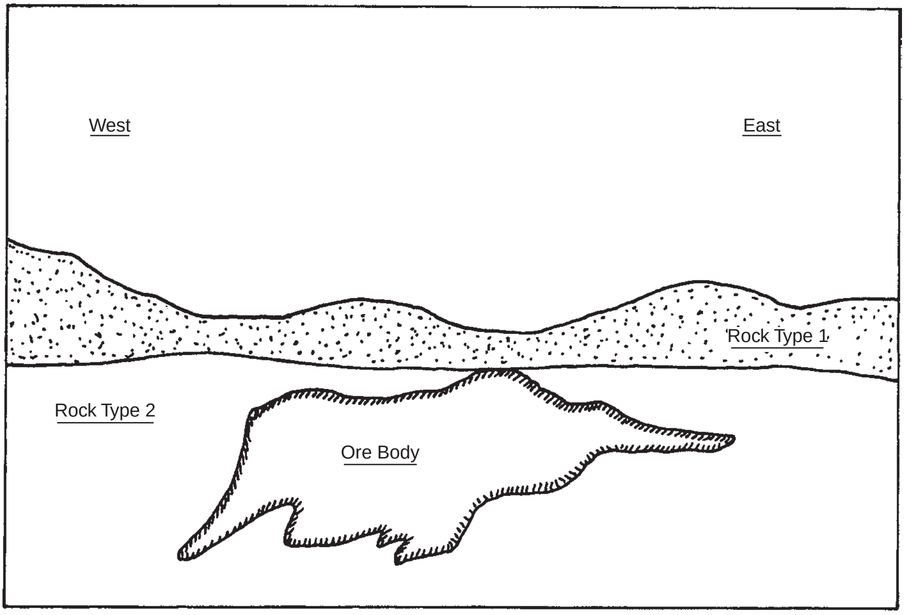


Figure 6.34. Hypothetical deposit for the sequencing study (Mathieson, 1982).

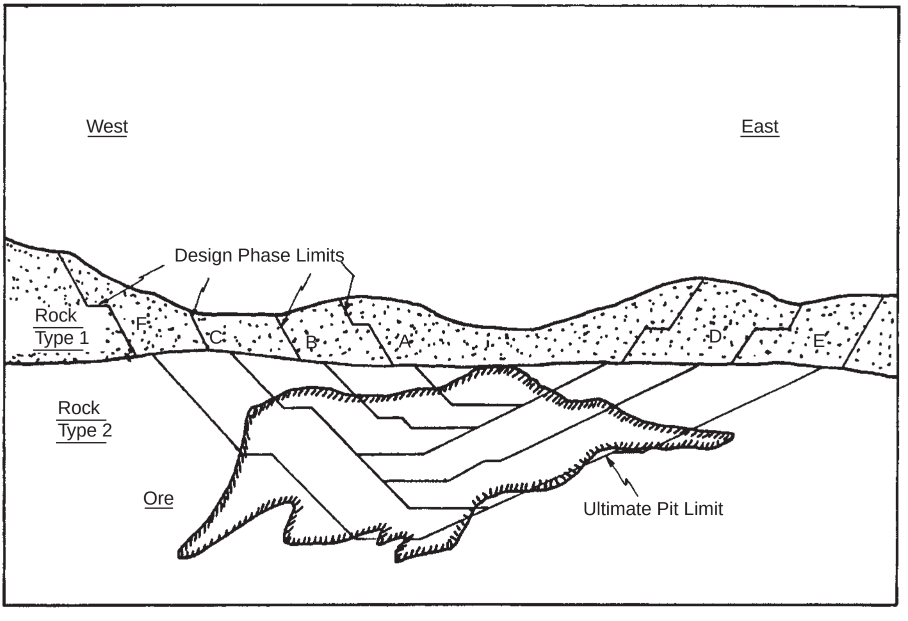


Figure 6.35. Development scheme for the hypothetical deposit (Mathieson, 1982).

Table 6.44. Tonnage-grade inventory by phase and bench (Mathieson, 1982).

Bench	Thousands of tons					
	'A' phase		'B' phase		'C' phase	
	Waste	Ore	Waste	Ore	Waste	Ore
5100	1500					
5050	3200					
5000	5000		200		400	
4950	3800		1800		1500	
4900	1500		2000		1800	
4850	400	1000	1500		2200	
4800	300	900	400	900	1600	
4750	200	800	300	900	300	1000
4700			200	700	500	2000
4650			100	600	800	2200
4600					300	1700
4550					100	700
	15,900	2700	6500	3100	9500	7600

Table 6.45. Summary of phase quantities (Mathieson, 1982).

Phase	Thousands of tons				
	Waste above first ore bench	Waste on ore benches	Ore	Ore life* (yrs)	Cumulative ore life* (yrs)
A	15,000	900	2700	1.08	1.08
B	5500	1000	3100	1.24	2.32
C	7500	2000	7600	3.04	5.36
D	12,800	3800	12,500	5.00	10.36
E	18,200	4900	15,100	6.04	16.40
F	22,000	4500	13,000	5.20	21.60
Total	81,000	17,100	54,000	21.60	

*Assuming an annual milling rate of 2,500,000 tons.

2. The ore-waste tonnage inventory by bench and phase is determined. The detailed results from the first 3 phases are given in Table 6.44. At this point the mining engineer would 'mine' the ore on each successive bench gradually stepping, in order, through the phases to meet the required annual mill production. Such an ore schedule would typically be done using a hand calculator or an interactive desk top computer program. Each phase would have a different ore life since they were defined on operating rather than schedule constraints. The first trials would be based upon a fixed cutoff grade.

3. For this simple example the overall phase quantities given in Table 6.45 will be used. Normally those broken down bench-by-bench (Table 6.44) would be considered.

4. These phases will be mined in sequence and no inter-phase blending will be considered in this simple schedule. Each phase contains a tonnage of ore which must be exposed or developed prior to the exhaustion of ore from the previous phase. A 2.5 million ton per year milling rate is assumed. This is divided into the ore tons available in each phase to determine

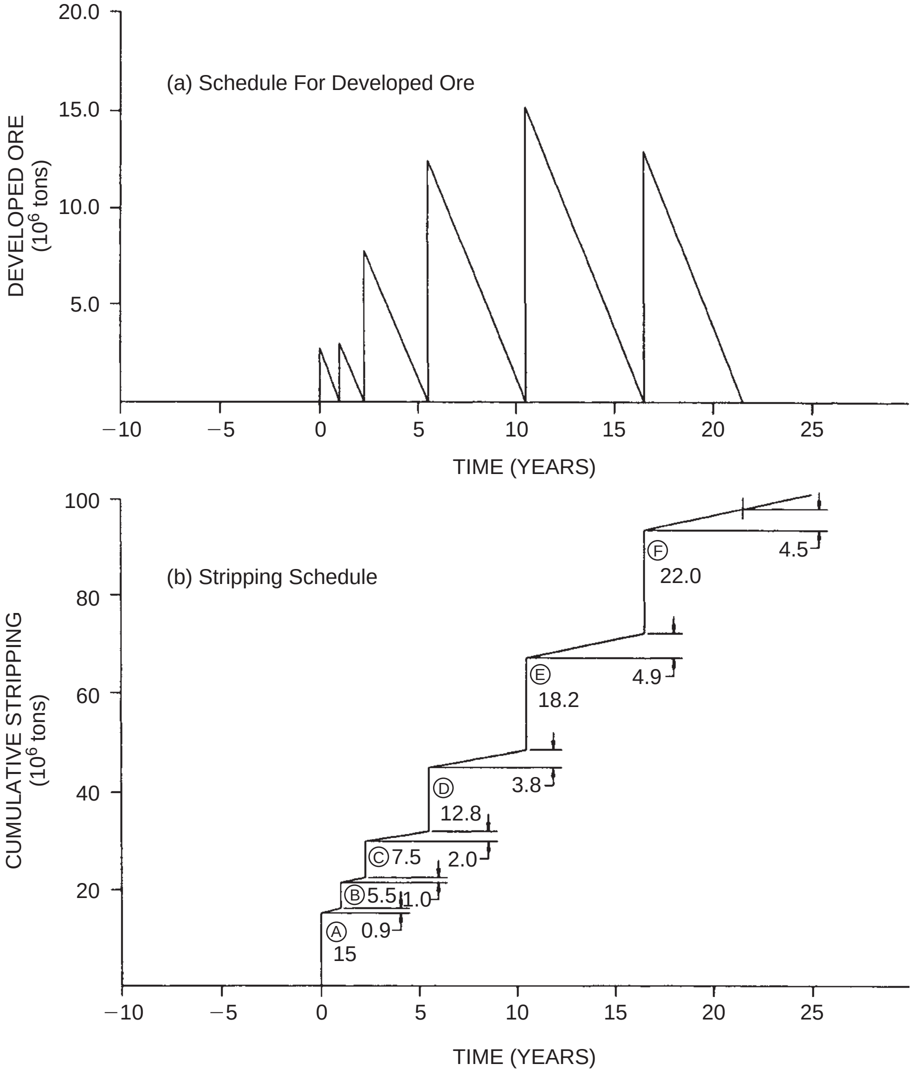


Figure 6.36. Developed ore and stripping schedules.

phase life. A saw-tooth plot of available ore versus time is made such as is shown in Figure 6.36a. This illustrates the availability and depletion of ore from the various phases.

5. The planner is now able to define the points in time at which the waste stripping must be completed for any given phase in order to sustain the ore supply. For phase A ore production to commence, the 15 million tons of waste lying above the first bench must have first been removed. During the mining of Phase A there is an additional 900,000 tons of internal waste assumed to be evenly distributed and 'locked-up' with the ore. In order for the phase B ore to be available, 21,400,000 tons of waste must be removed. The cumulative waste tons versus time plot is shown in Figure 6.36b. The vertical steps in the plot correspond to the respective

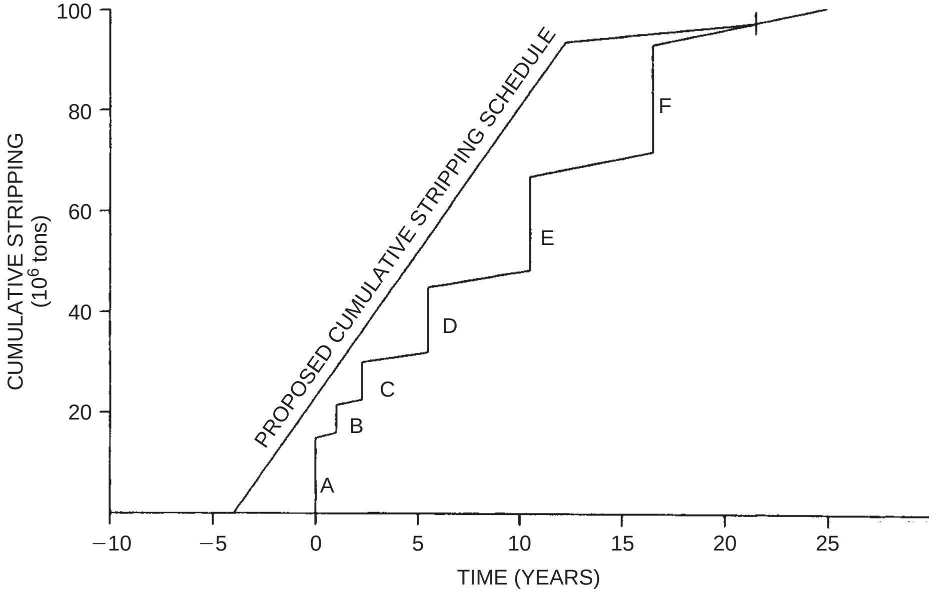


Figure 6.37. Proposed cumulative stripping schedule for the example orebody.

overburden stripping quantities above the first ore bench of each phase. The latter segments represent the progressive mining of internal waste.

6. The next step is to arrive at a 'smoothed' stripping schedule which exceeds the minimum. A possible schedule is shown as the straight line superimposed on Figure 6.37. It consists of a 4 year pre-production period totalling 20 million tons followed by a constant stripping rate of 5 million tons per year through year 15. Beyond this only the internal waste of phase F remains.

The detailed preproduction stripping schedule is shown in Figure 6.38. It consists of 2.5 million tons during the first year when the crews are being trained and equipment is being delivered. During years 2 and 3 the stripping rate is 5 million tons/year. Finally in year 4 the rate is increased to the total material rate (ore plus waste) which will be sustained nearly throughout the remaining life. It can be seen from the figure that the required stripping is completed prior to the required time by various amounts.

Phase	Early completion (months)
A	8
B	9
C	4
D	5
E	9
F	16

7. The initial curve of developed ore versus time is now adjusted to reflect the early completion of the stripping. This construction is shown in Figure 6.39. The cumulative stripping and developed ore curves are used. For phase B one moves horizontally at the required

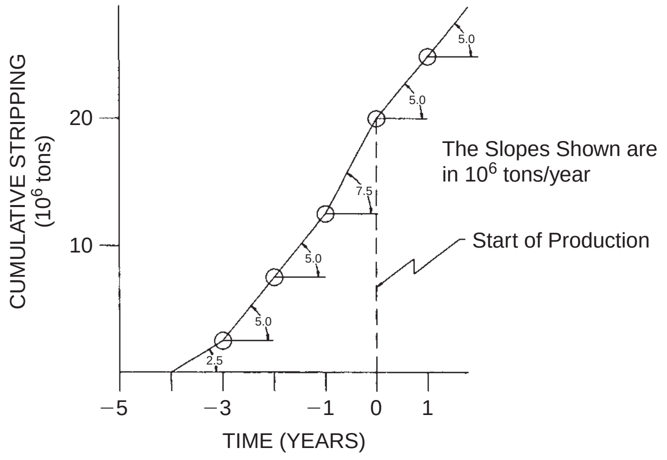


Figure 6.38. Preproduction stripping schedule.

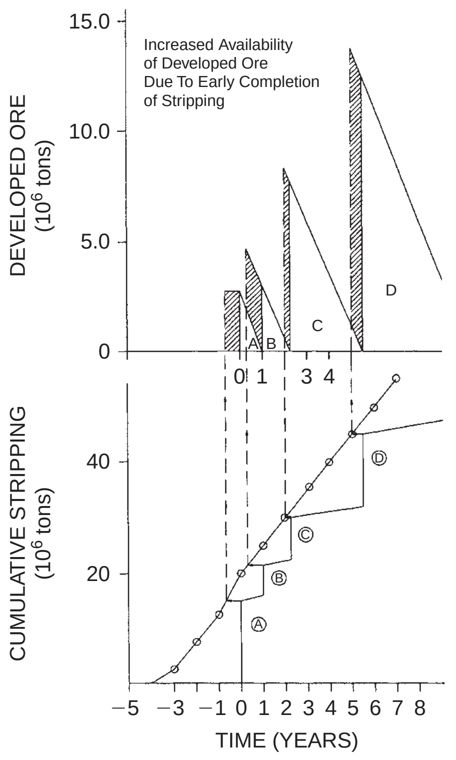


Figure 6.39. Construction showing modification of ore availability.

cumulative stripping level to the actual stripping curve. Then one proceeds vertically to the developed ore curve. As can be seen there are approximately 2 million tons of ore from phase A remaining. Due to the early development of phase B, an additional 3.1 million tons of ore become available. The construction is simply to extend the slanted portion of the phase B ore reserve upward until it meets the vertical line from the stripping graph. This process is repeated for phases C through F. For phase A, the ore in the phase simply becomes available earlier. The cross hatched areas indicate contingency ore available in the case that the stripping schedule falls behind. The modified developed ore curve represents the predicted inventory balance with time. A further acceleration of stripping is sometimes done in the trial scheduling process to guard against possible surprises in mineable reserves. This can help to avoid:

– an unexpected crash stripping program,
– a forced reduction in mill feed,
– a temporary lowering of the cutoff grade to sustain planned concentrate production.

8. With this thorough understanding of the orebody and its development options, the pit planner presents his/her findings to management.

9. Final mine plan period maps are drawn up to test the viability of the plan. Some refinements in the ramping and phasing strategy, etc., may be needed but major changes are unlikely.

10. To this point, a series of logical pit development phases have been defined based on the 'next best' profitable ore and a fixed cutoff. The plan can now be fine tuned. Alternative ore and waste schedules based on variable production rate and cutoff grade strategies can be developed using the computed tonnage – grade inventories within each successive phase. Such schedules can then be compared economically through standard internal ROR analysis. A visual comparison can be achieved by plotting cumulative operating cash flow with time.

11. Once a 'final' production schedule has been decided upon, the planner generates a series of period end plans. These might be for example:

(a) end of preproduction,
(b) years 1–5 in yearly increments,
(c) years 10, 15, etc.

These plans would be based on both the phase designs and the paper schedules. They would constitute a vital test on the mineability of the proposed plan. Shovel and drill deployment and any internal temporary ramping would also be considered in detail.

6.9.3 *Block sequencing using set dynamic programming*

Introduction

In 1974, Roman (1974) described an algorithm for determining the optimum mining sequence and pit limits patterned after one originally presented by Lerchs & Grossmann (1965). The process will be demonstrated through the use of a 2-dimensional example. Figure 6.40 is a schematic representation of a slice through a block model.

An index number representing the column and row position for each block is assigned. The first step in the process is to convert the grade block model into an economic block model. To assign the appropriate costs and revenues a decision must be made at this point regarding the destination of each block. Three possibilities might be:

– mill,
– leach dump, and
– waste dump.

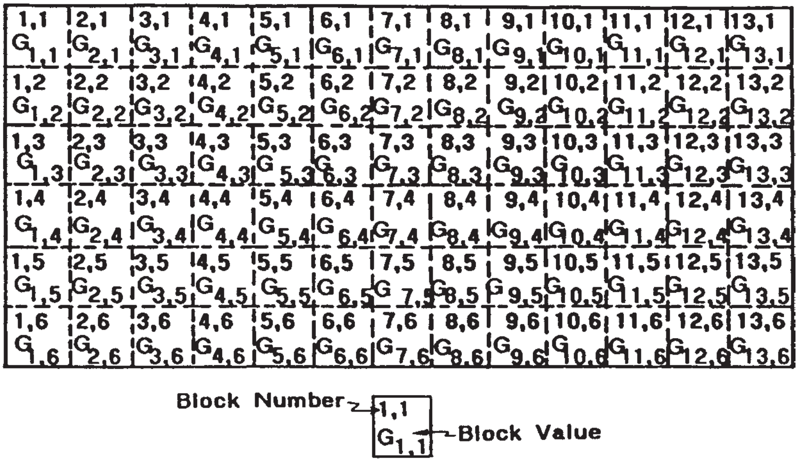


Figure 6.40. Schematic of the ore deposit showing block numbers and grades (Roman, 1974).

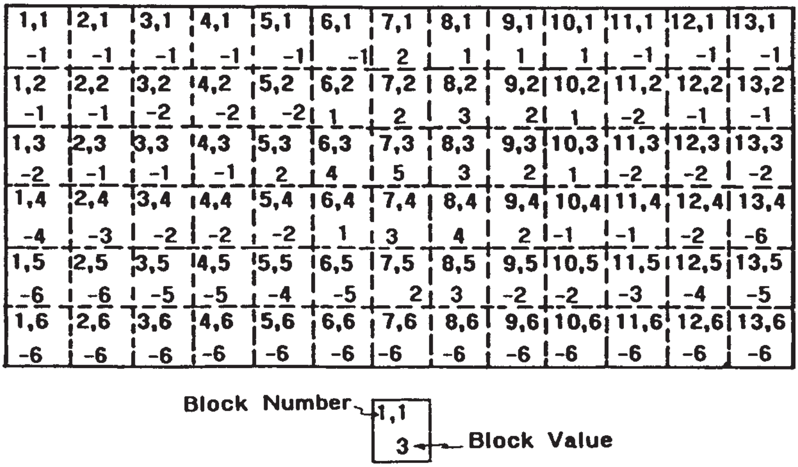


Figure 6.41. Schematic of the ore deposit showing block numbers and block values (Roman, 1974).

The net block value is determined by subtracting the mining and processing costs from the revenues. The mining costs are for the block alone and do not include stripping costs. Figure 6.41 shows the resulting economic block model. At this point a constraint relating to the final pit slope is introduced.

Constraint 1: *The pit wall slope may not exceed 1:1 at any point.*
Applying the floating cone procedure introduced in the previous chapter one would arrive at the final pit shown in Figure 6.48. This same result will be achieved using the technique described in this section. The problem is to determine the sequence in which the blocks should be mined so that the net present value for the section is a maximum.

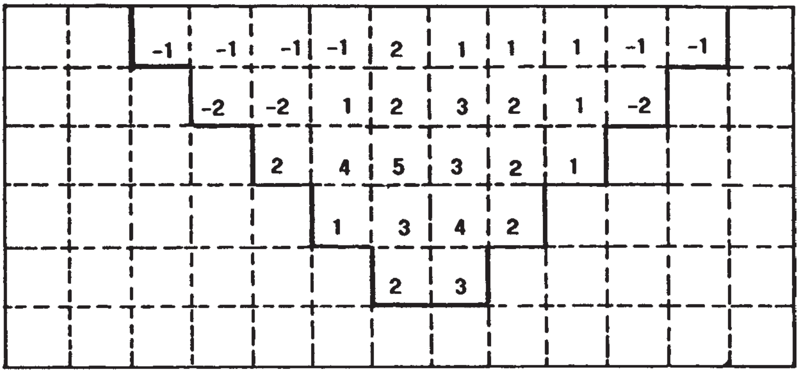


Figure 6.42. Schematic of the ore deposit with the final pit limits as determined using the floating cone superimposed.

The optimum sequence. To begin the sequence optimizing process, the economic block model is scanned to determine the maximum outline which the future pit could assume. Obviously all of the positive blocks on the section must be included and the pit limit slopes obeyed. The objective is to identify the location of the last block which might be mined. In this example, the last block has been selected as the bottom vertice of the inverted triangle containing all blocks of positive value. This triangle shown in Figure 6.43 has been constructed in accordance with constraint 1. An alternative procedure would be to select a hypothetical block (a block that does not exist) lying on or below the lowest level of positive blocks. To simplify the discussion, the triangle approach has been used. There are 36 blocks included within the triangle. If each block corresponds to a unit time period (of unspecified length), 36 time periods are required to mine all of the blocks. Block (7,6) as seen in Figure 6.44 is the last one to be mined. It is mined in period 36. In order to mine block (7,6) one must first mine blocks (6,5), (7,5) and (8,5). At this point a second constraint, one regarding sequencing, will be introduced.

Constraint 2: *Each mining level may be entered at only one point.*
With this constraint in place there are only two sequencing options for the mining of blocks in time periods 35 and 36:
Option 1. Mine block (6,5) followed by block (7,6) (Figure 6.45a).
Option 2. Mine block (8,5) followed by block (7,6) (Figure 6.45b).
The third option (shown in Figure 6.45c) of mining block (7,5) in period 35 means that both blocks (8,5) and (6,5) had been mined earlier. This requires that two separate entries be made on level 5 thus violating constraint 2. Hence this is not an option.

In time period 34 there are several choices for the block to be mined depending upon the block mined in period 35. If the last two blocks mined are (6,5) and (7,6) then the possible sequences for the last 3 periods are

$$(1)\ (7,5) \rightarrow (6,5) \rightarrow (7,6)$$

$$(2)\ (8,5) \rightarrow (6,5) \rightarrow (7,6)$$

$$(3)\ (5,4) \rightarrow (6,5) \rightarrow (7,6)$$

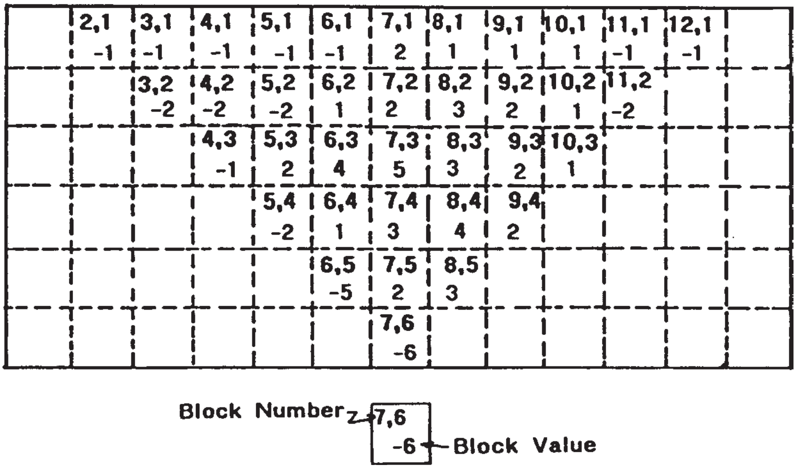


Figure 6.43. Schematic of the ore deposit showing the triangle containing the maximum pit superimposed (Roman, 1974).

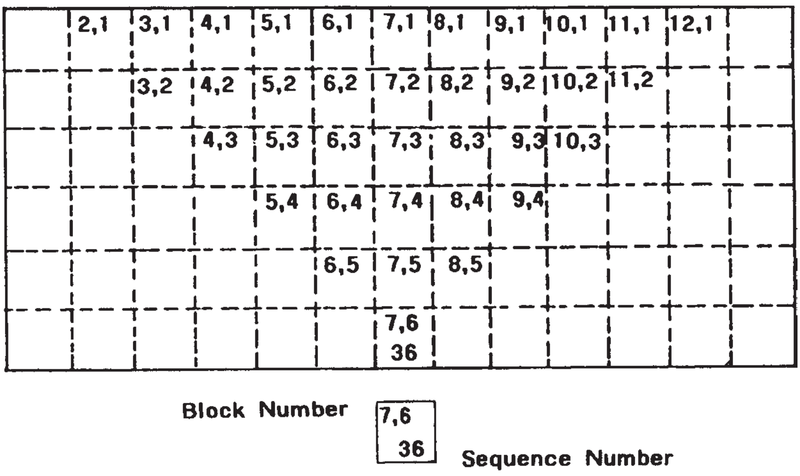


Figure 6.44. Blocks which must be removed prior to mining block 36.

On the other hand if blocks (8,5) and (7,6) are mined last then the possible sequences are:

(4) $(6, 5) \rightarrow (8, 5) \rightarrow (7, 6)$

(5) $(7, 5) \rightarrow (8, 5) \rightarrow (7, 6)$

(6) $(9, 4) \rightarrow (8, 5) \rightarrow (7, 6)$

These 6 possibilities are shown in Figure 6.46.

Sequences 2 and 4 however involve mining the same 3 blocks in just a different order. An economic evaluation is performed to determine the most attractive of the two alternatives.

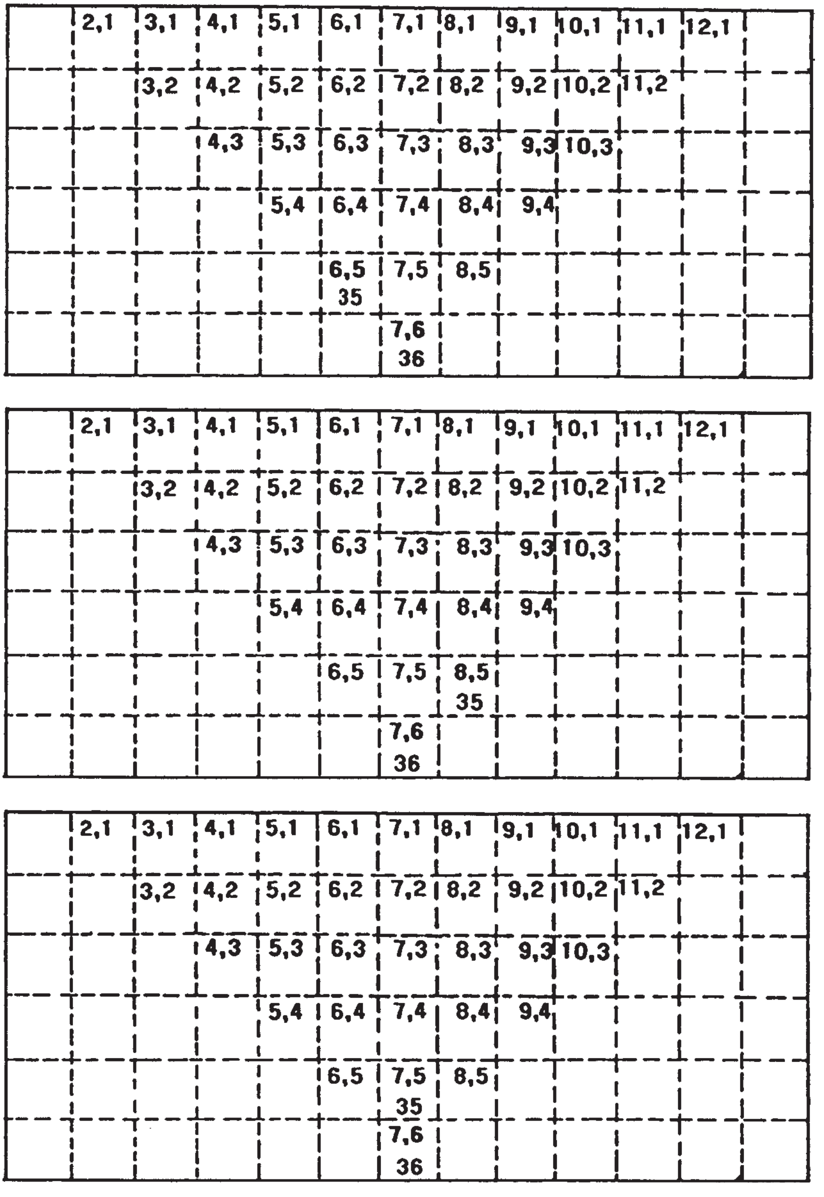


Figure 6.45. Possible sequences for mining blocks 35 and 36.

The least attractive is dropped from further consideration. Choosing an interest rate of 10% and discounting to the beginning of time period 34 one finds:

Sequence 2: $(8, 5) \rightarrow (6, 5) \rightarrow (7, 6)$

$$\text{NPV}_2 = \frac{\$3}{(1.1)^1} - \frac{\$5}{(1.1)^2} - \frac{\$6}{(1.1)^3} = -\$5.91$$

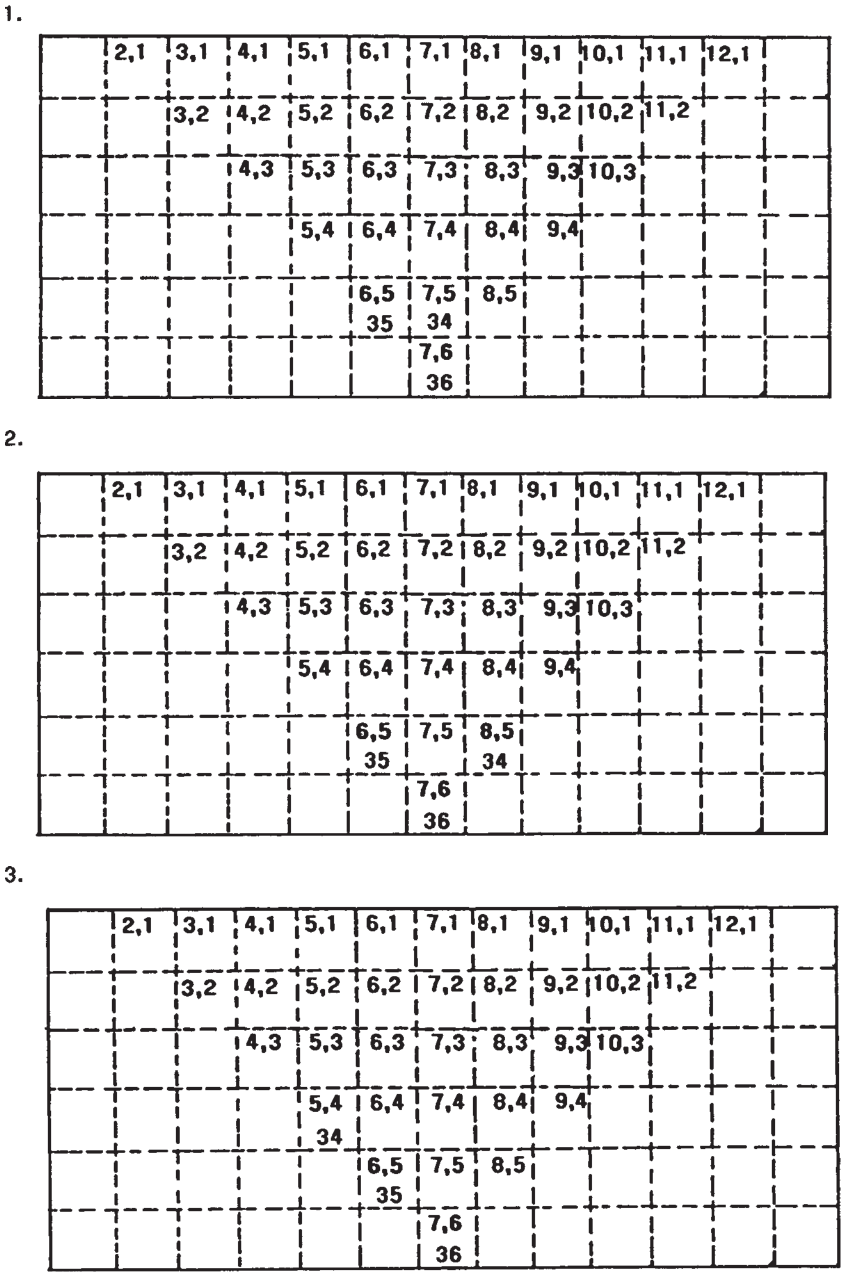
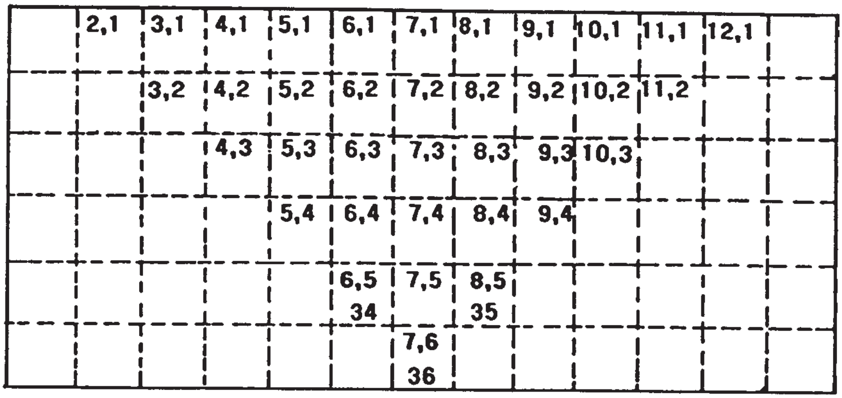


Figure 6.46. Possible sequences for mining blocks 34, 35, and 36.

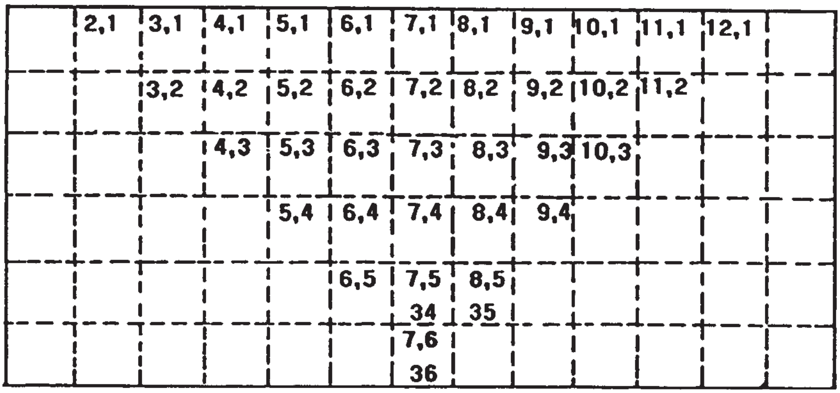
Sequence 4: $(6,5) \rightarrow (8,5) \rightarrow (7,6)$

$$\text{NPV}_2 = \frac{-\$5}{(1.1)^1} + \frac{\$3}{(1.1)^2} - \frac{\$6}{(1.1)^3} = -\$6.57$$

4.



5.



6.

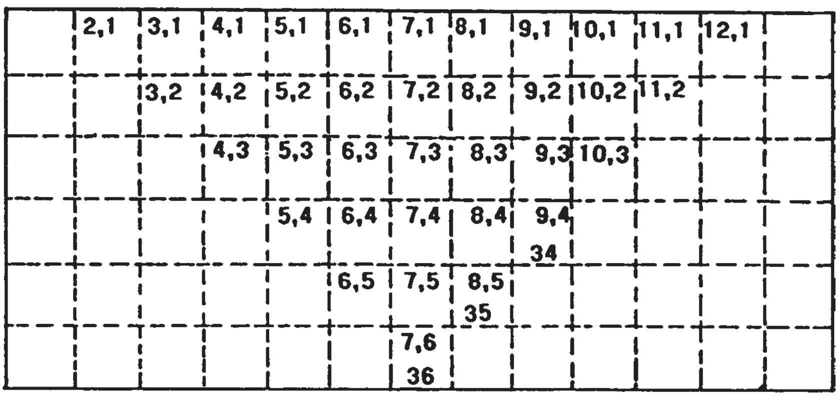


Figure 6.46. (Continued).

Sequence 4 is the least attractive of the two and is dropped. The five block combinations for periods 34 through 36 which must be included when sequencing the remaining 33 periods are:

Sequence	Mining order
1	(7,5) → (6,5) → (7,6)
2	(8,5) → (6,5) → (7,6)
3	(5,4) → (6,5) → (7,6)
4	(7,5) → (8,5) → (7,6)
5	(9,4) → (8,5) → (7,6)

The remaining choices for sequencing the final four blocks after eliminating duplicate combinations of blocks by the present value analysis are:

Sequence	Mining order
1	(8,5) → (7,5) → (6,5) → (7,6)
2	(9,4) → (8,5) → (6,5) → (7,6)
3	(7,5) → (5,4) → (6,5) → (7,6)
4	(8,5) → (5,4) → (6,5) → (7,6)
5	(4,3) → (5,4) → (6,5) → (7,6)
6	(9,4) → (7,5) → (8,5) → (7,6)
7	(10,3) → (9,4) → (8,5) → (7,6)

This process is continued until all 36 blocks have been included. In this final stage the various sequences will just be permutations of the same combination. Consequently the optimum sequence can be determined through a present value calculation. Figure 6.47 shows the section with the 36 blocks numbered in the order that they are to be removed.

Determination of the optimum pit

Once the optimum sequence has been found, the pit outline can be developed. The procedure for determining which of the 36 blocks in the optimum sequence are actually to be mined is as follows:

1) Identify the last block in the optimum sequence with a positive value. Drop all blocks after this one.

2) Examine the remaining sequence of blocks and identify the latest negative block scheduled for mining. If there are a number of negative blocks in a row select the earliest in the row. Determine the present worth for the sequence extending from identified negative block to the end.

3) If the present value is negative, drop all these blocks from the optimum sequence and repeat step 2. If the subsequence has a positive present value, replace the subsequence by an equivalent block value at the end of the sequence.

4) Repeat steps 2 and 3 until the first mined block is included in the subsequence. The final present value is that of the optimum pit.

Applying these rules to the example problem, it is seen that blocks 31 through 36, which are all negative, are dropped by inspection. Continuing along the sequence it is seen that two adjacent blocks (13,1) and (4,2) corresponding to mining periods 26 and 27 respectively, are negative. The net present value for this subsequence (blocks 26 to 30) discounted to the

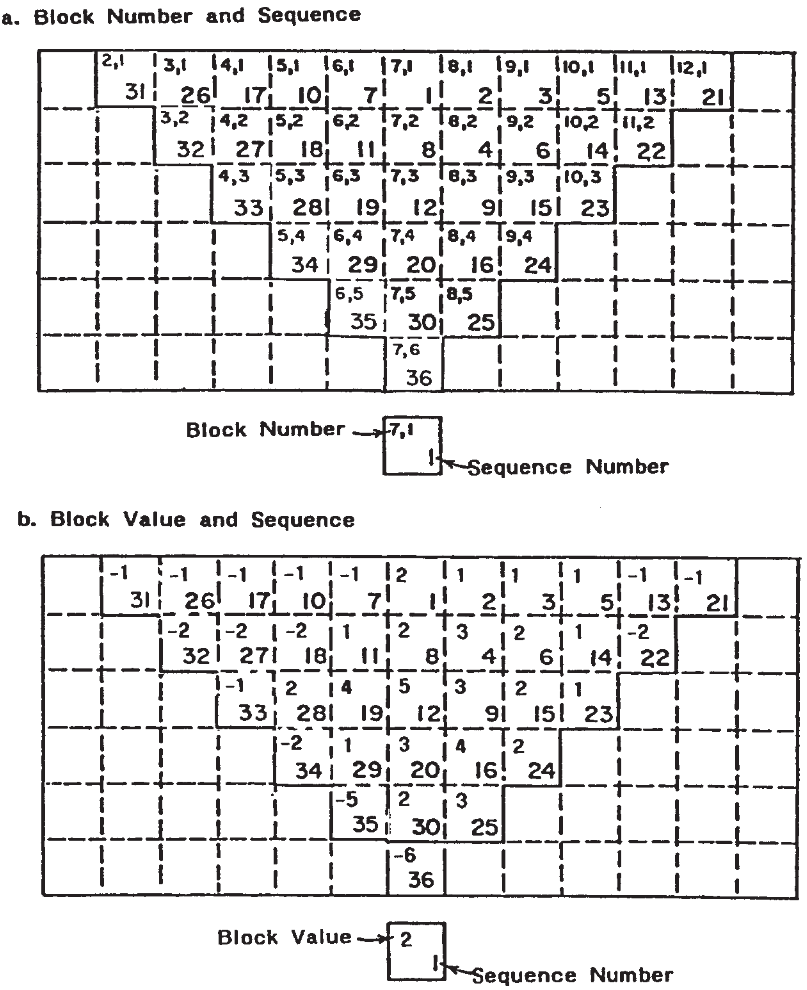


Figure 6.47. Schematic of the deposit showing the optimum block mining sequence (Roman, 1974).

beginning of period 26 is

$$\begin{aligned}\text{NPV} &= \frac{-\$1}{(1.1)^1} - \frac{\$2}{(1.1)^2} + \frac{\$2}{(1.1)^3} + \frac{\$1}{(1.1)^4} + \frac{\$2}{(1.1)^5} \\ &= \$0.87\end{aligned}$$

Since it is positive, this subsequence, referred to as subsequence SS1 is retained. The next switch between negative and positive blocks occurs after block 21. The net present value for the subsequence SS2 through to the beginning of period 21 is

$$\begin{aligned}\text{NPV} &= \frac{-\$1}{(1.1)^1} - \frac{\$2}{(1.1)^2} + \frac{\$1}{(1.1)^3} + \frac{\$2}{(1.1)^4} + \frac{\$3}{(1.1)^5} + \frac{\$0.87}{(1.1)^6} \\ &= \$1.91\end{aligned}$$

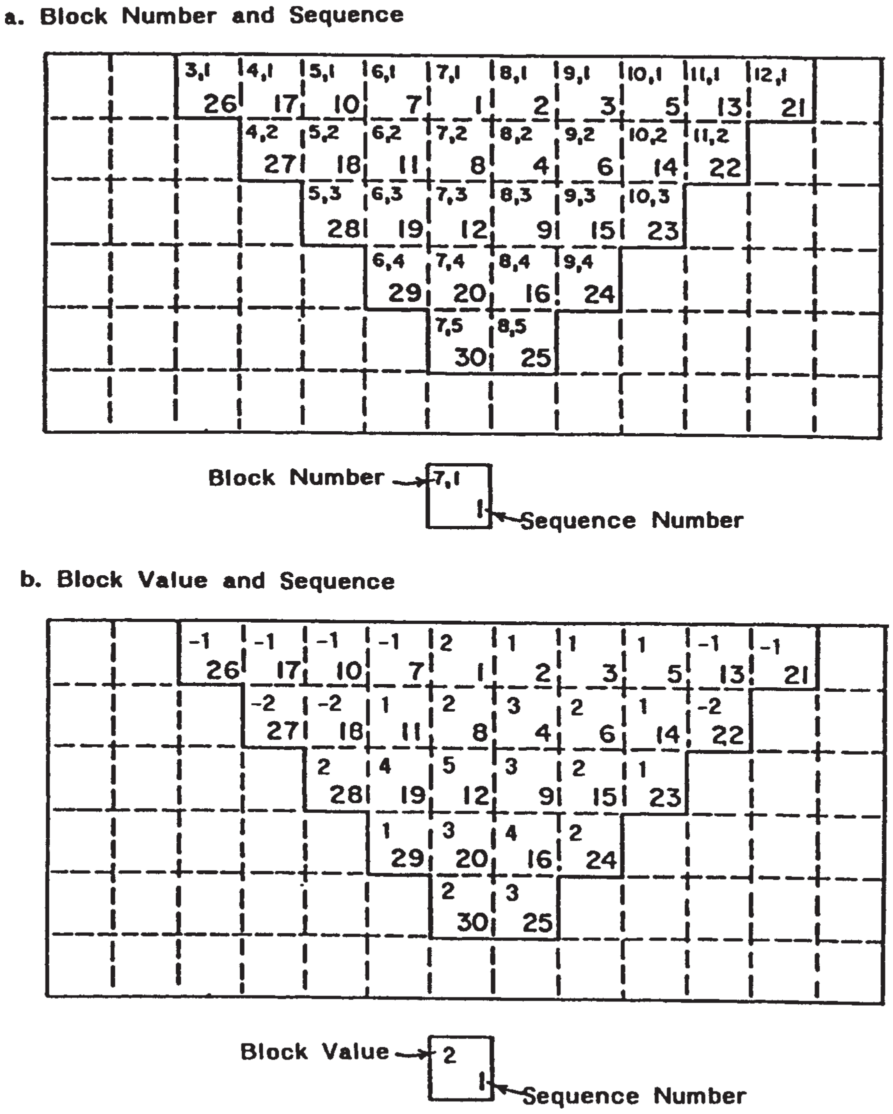


Figure 6.48. The final pit outline and optimum mining sequence (Roman, 1974).

It is positive and hence retained. This process of calculating net present value is continued until the first block to be mined is included. The final pit is shown in Figure 6.48. The overall NPV for the pit on this section is

$$\text{NPV (overall)} = \$12.60$$

Simply summing the values of the blocks included within the outlined yield

$$\text{Sum} = \$34$$

This would be the result if the pit could be mined instantaneously at time zero.

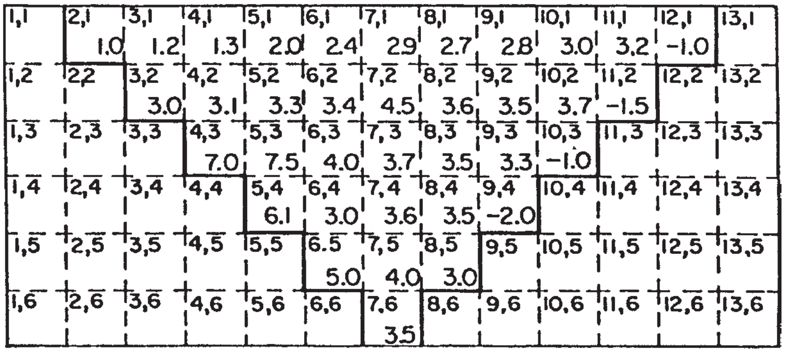


Figure 6.49. Hypothetical deposit showing block values and maximum pit limits (Roman, 1974).

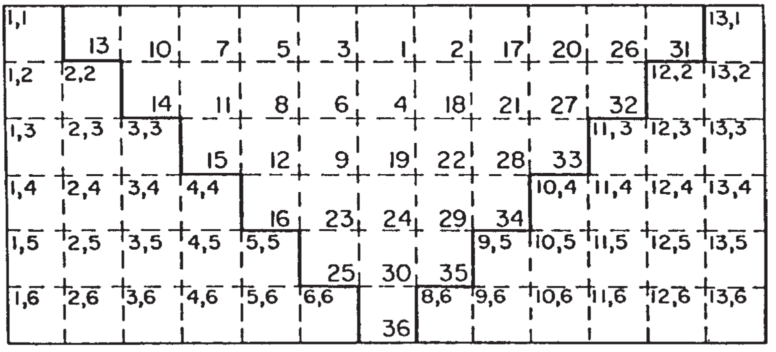


Figure 6.50. Mining sequence and pit limit for an interest rate of 5% (Roman, 1974).

Time value of money influence

In the previous example, a 10% discount rate was applied in arriving at the optimum sequence and pit. The question arises as to the influence this rate has both on the sequence and the final pit. Figure 6.49 shows the section to be used with the maximum pit superimposed. Using the techniques and constraints previously described the pits corresponding to the use of 5%, 10%, 20%, and 50% discount rates are shown in Figures 6.50 through 6.53 respectively. As can be seen, a definite effect is observed. At first glance it would appear that the reduction in pit size between discount rates of 5% and 10% is incorrect since the strip of blocks

$$(12, 1) \rightarrow (11, 2) \rightarrow (10, 3) \rightarrow (9, 4) \rightarrow (8, 5) \rightarrow (7, 6)$$

when summed has a positive value (+1). However when taking sequencing into account the discounted value of this strip (at a rate of 10%) is

$$\begin{aligned} \text{NPV} &= \frac{\$3.5}{(1.1)^{36}} - \frac{\$3.0}{(1.1)^{35}} - \frac{\$2.0}{(1.1)^{34}} - \frac{\$1}{(1.1)^{33}} - \frac{\$1.5}{(1.1)^{32}} - \frac{\$1.0}{(1.1)^{31}} \\ &= -\$0.025 \end{aligned}$$

For higher discount rates this sub-sequence is even more negative.

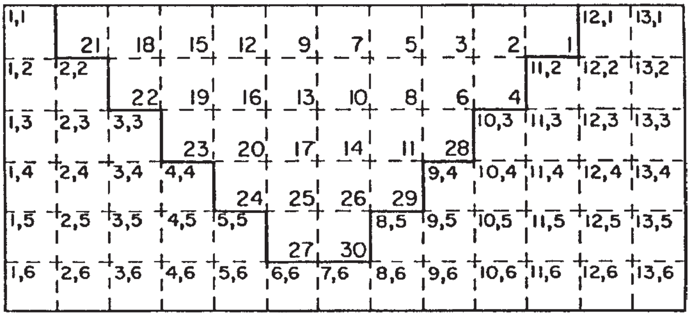


Figure 6.51. Mining sequence and pit limit for an interest rate of 10% (Roman, 1974).

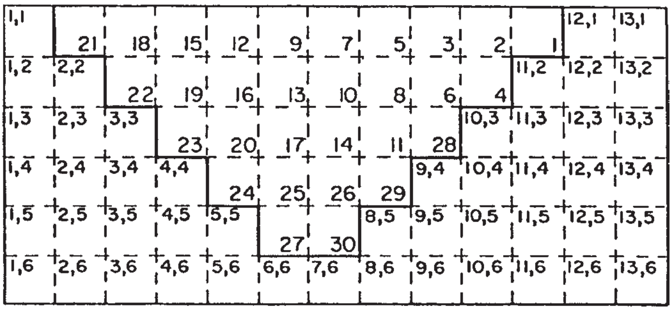


Figure 6.52. Mining sequence and pit limit for an interest rate of 20% (Roman, 1974).

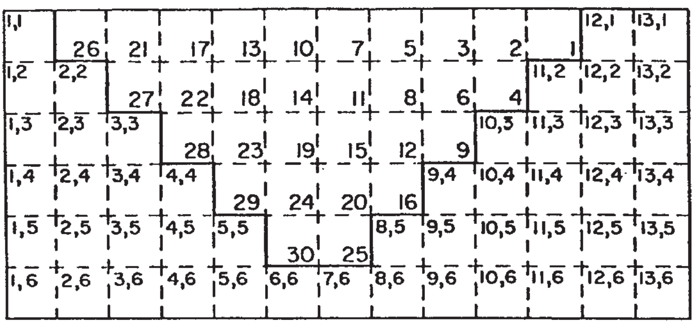


Figure 6.53. Mining sequence and pit limit for an interest rate of 50% (Roman, 1974).

The rate to be used depends upon a number of factors, the principal one being that of company policy. In the early stages of a property, a high rate of return is generally desired in order to repay the investment as quickly as possible. Later when the investment has been repaid, another value may apply. Thus the procedure outlined is not intended to be used just once in the life of a property and the derived sequence to be followed without change.

Rather with changing conditions, it can be rerun on the blocks remaining at any given time and a new optimum sequence developed.

Summary
The described sequencing procedure is quite simple. Although the examples were 2-dimensional in nature, programs containing the algorithm have been written for the true 3-dimensional pit problem. Changes in constraints on the mining sequence can be incorporated relatively easily. In theory, the size of problem which can be handled is unlimited. However as was seen in the 2-dimensional hand example, the number of sequences to be considered increases rapidly as the number of blocks contained in the deposit increases. A relatively large and fast computer is required to perform the calculations for even a small deposit. A deposit for which ore characteristics fluctuate radically and continually along any direction will require more computer time and storage to evaluate. On the other hand, adding constraints on the mining sequence tends to simplify the problem. If in the problem of sequencing the 36 blocks constraint 2 is dropped, the required computer time increases by a factor of 30 and the storage by 20 times.

Rather than attempting to provide an overall optimum sequence for an entire deposit consisting of many thousands of blocks, the mine plan may be first broken down into a series of phases. The sequencing procedure can then be applied to each of the phases in turn. In this way a series of sub-optimizations is realized. Being sub-optimizations, due care should be taken in putting these phases together, but it is possible in practice to build up a good mine plan in this way.

6.9.4 *Some scheduling examples*

Dagdelen (1985) has applied the Lagrangian parametrization technique for optimum open pit mine production scheduling. A full discussion of the procedure is beyond the scope of this textbook. An ultimate pit limit contour is first obtained using Lerchs-Grossmann's 3-D algorithm. A series of constraints such as: (a) mining capacity, (b) milling capacity, (c) mill feed grade, (d) geometric (i.e. slope limitation), are then introduced.

The program searches through the set of candidate blocks and selects those yielding a maximum net present value for the deposit while meeting the constraints. The results of applying this procedure to a small high grade copper deposit are included here to demonstrate the effect of changing constraints.

The grade block model (25×20 blocks) for benches 1 through 6 of this deposit is shown in Figure 6.54. The pit limits obtained using a 45 degree slope constraint have been superimposed. Each of the blocks has plan dimensions 100 ft $\times$ 100 ft and the block (bench) height is 45 ft. The assumptions made in the development of the economic block model are given in Table 6.46. The level by level statistics of reserves in the ultimate pit limit contour are given in Table 6.47. The tonnage factor for both ore and waste is 12.0 ft^3/ton, hence there are 37,500 tons of material per block. The overall material in the pit consists of

Material	Blocks	Tons	Grade
Waste	505	18,937,500	0
Ore	63	2,362,500	3.68% Cu
Total	568	21,300,300	NA

If all the material in the pit could be mined instantaneously then the net value (revenues-costs) would be about \$57,632,000. It will be assumed to be mined over a 3 year period

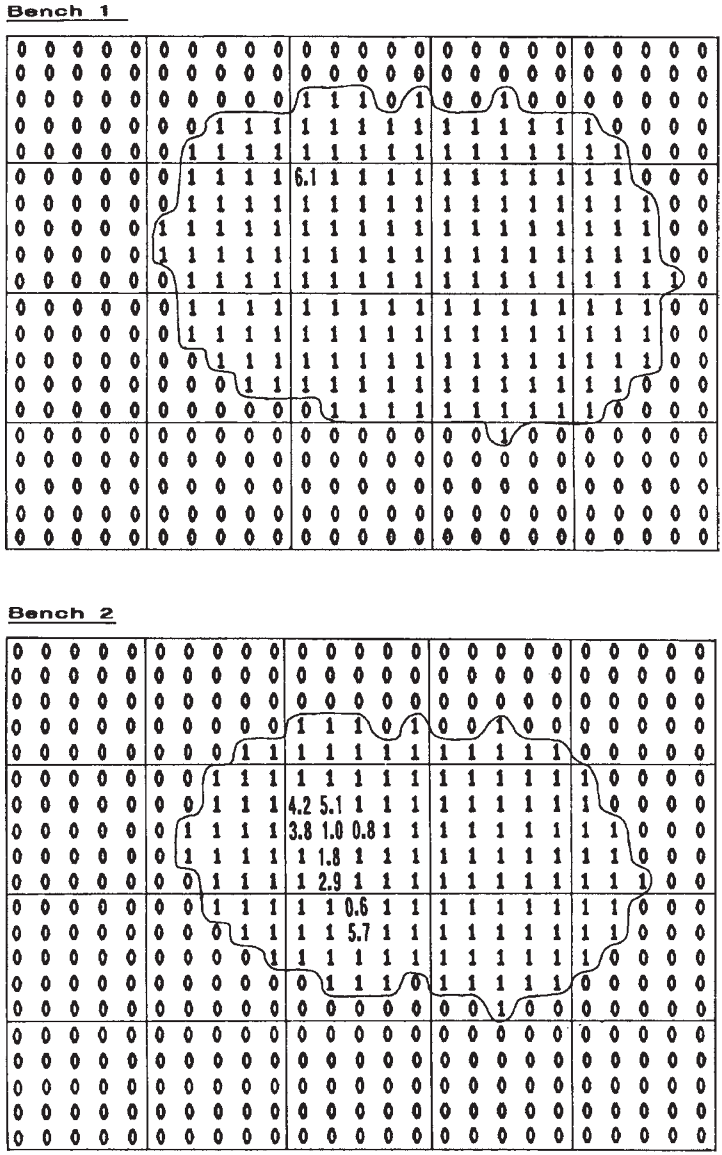


Figure 6.54. Grade block model with final pit limits superimposed. The number 1 denotes waste with grade 0 (Dagdalen, 1985).

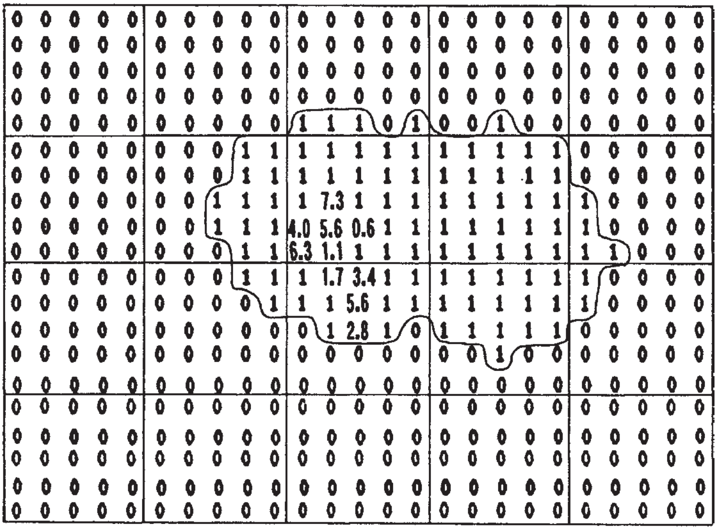
instead. A discounting rate of 12.5% will be assumed. The precedence (order of mining) and slope constraints which apply are respectively

– In order to remove a given block, overlying blocks in the cone of influence must be removed first.

– Pit slopes may not exceed the given maximum values (45° in this case).

– The bench faces are considered to be vertical.

Bench 3



Bench 4

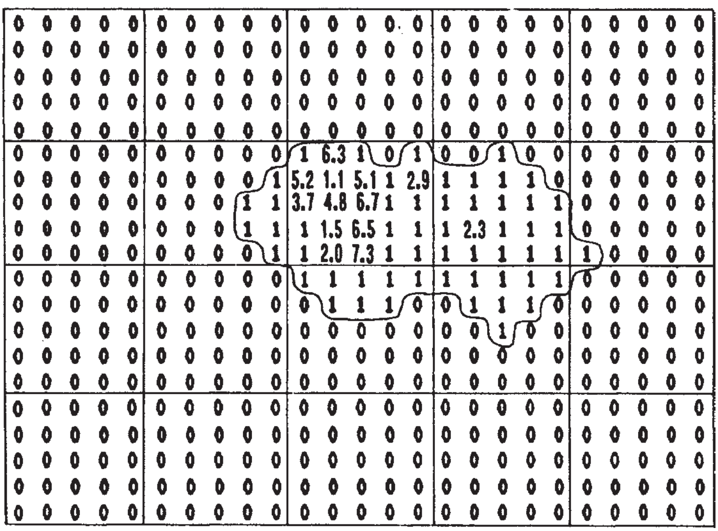


Figure 6.54. (Continued).

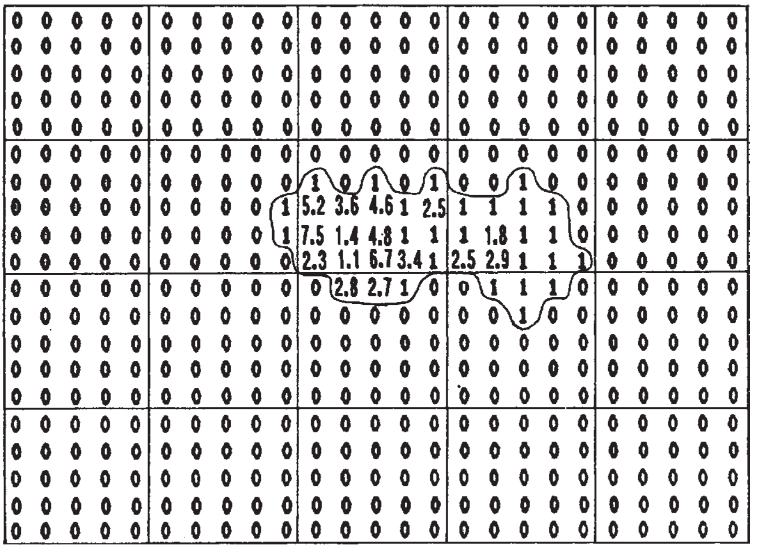
Constraint example 1

The first schedule will be made subject to the following ore mining (milling) capacity constraint:

1. Mine 19 ore blocks in period 1.
2. Mine 21 ore blocks in period 2.
3. Mine 23 ore blocks in period 3.

The problem is to schedule the operation such that the above restrictions are not violated and, at the same time, the total discounted profits before tax are maximized. The resulting

Bench 5



Bench 6

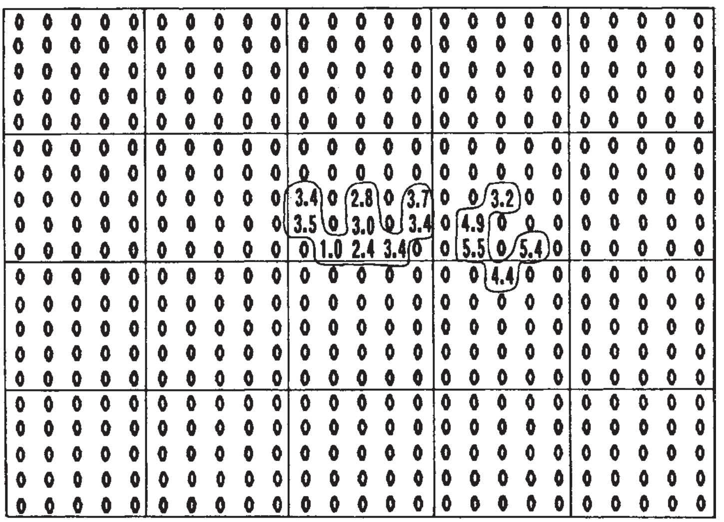


Figure 6.54. (Continued).

Table 6.46. Values used in the economic block model calculation (Dagdalen, 1985).

Price of copper	\$0.80 per lb
Mining costs	\$0.85 per ton of material
Milling costs	\$2.20 per ton of ore
Smelting, refining and marketing	\$0.25 per lb of Cu
G&A	\$0.15 per ton of ore
Overall metal recovery	85%

Table 6.47. Summary of the reserves within the ultimate pit limits by level (Dagdalen, 1985).

Bench	Number of ore blocks	Ave. grade	Number of waste blocks
1	1	6.10	196
2	9	2.88	134
3	10	3.84	95
4	13	4.26	55
5	16	3.49	25
6	14	3.57	0
Total	63	3.68	505

schedules are shown in Figure 6.55 and in Table 6.48. In Figure 6.55, the numbers indicate the period in which each block is scheduled to be mined.

The average grade of the ore mined is 3.7% Cu in the first year, goes up to 4.4% Cu in the second year and decreases to 3.0% Cu in year 3 (Table 6.49).

According to the schedule, very little stripping is required in year 1; the amount of stripping is more than doubled in year 2; and most of the stripping is done in year 3. No cost adjustments have been made to reflect this schedule. The net present value of this schedule is

$$\begin{aligned}\text{NPV} &= \frac{\$20,352,206}{1.125} + \frac{\$25,341,619}{(1.125)^2} + \frac{\$11,857,556}{(1.125)^3}\\ &= \$18,090,850 + \$20,023,008 + \$8,327,941\\ &= \$46,441,800\end{aligned}$$

Imposing yearly ore tonnage constraints on the system together with discounting reduced the net present value by $11,190,225.

Constraint example 2

For this example, an additional restriction on mill feed grade is imposed together with the ore mining capacity constraints of the previous example; specifically:

1. Mine 19 ore blocks averaging 3.7% Cu in year 1.
2. Mine 21 ore blocks averaging 3.7% Cu in year 2.
3. Mine 23 ore blocks averaging 3.7% Cu in year 3.

As indicated in the discussion of the ultimate pit reserves, the average grade of the total reserve within the ultimate pit contour was 3.68% Cu. Hence, this new restriction is to force the operation to mine as close to this average grade as possible.

The optimum solution to the above mining system is depicted in Figure 6.56. The numbers on the different benches indicate the years in which the block will be mined.

The summary statistics by bench and year are given in Table 6.50.

All of the constraints are satisfied except for the average grade requirement in year 2.

In year 2 the average grade is 3.6% as compared to the required 3.7%. The reason for this is a lack of available blocks. Table 6.51 summarizes the results. The NPV for this schedule, $45,929,335, is slightly less than that obtained in the previous example.

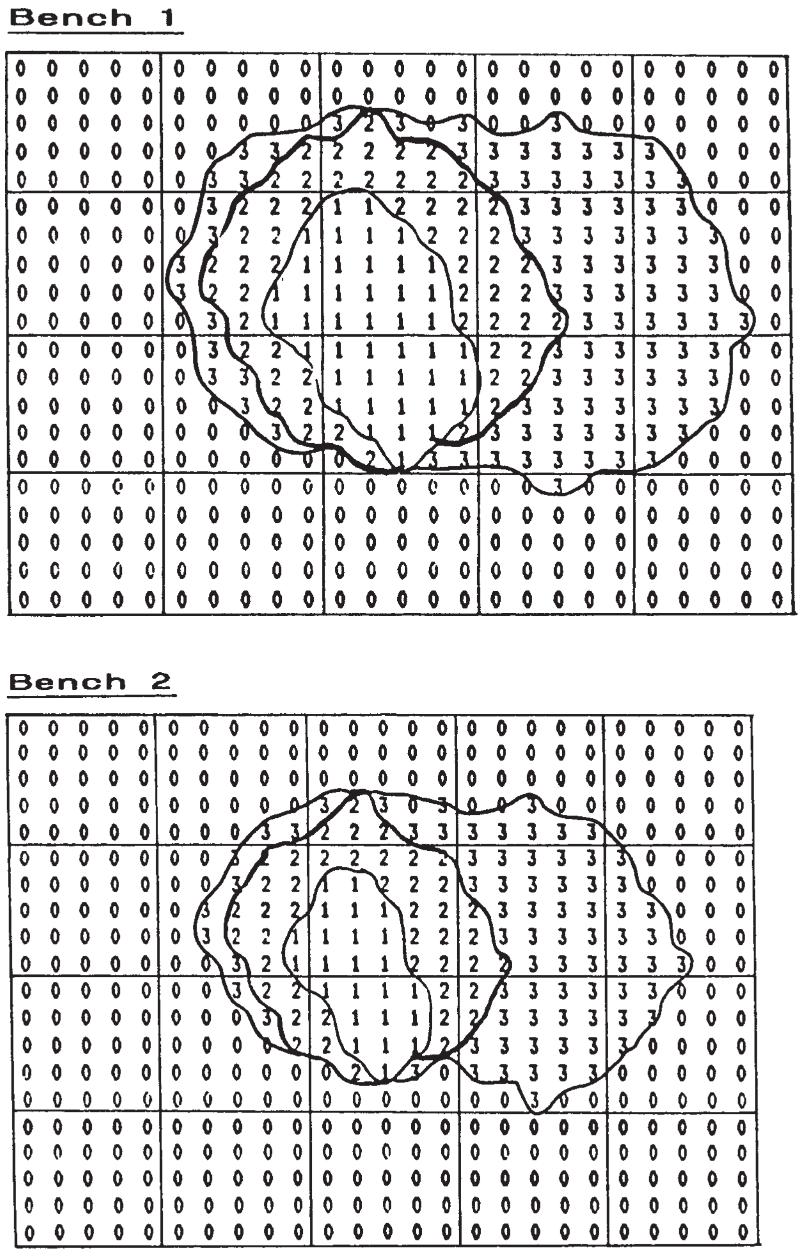


Figure 6.55. Proposed mining sequence for years 1, 2 and 3 under constraint set 1 (Dagdalen, 1985).

This reduction is caused by blending some of the low grade material with high grade in period 2.

These example studies can be expanded to include other conditions to determine the effects and costs of different constraints on the system.

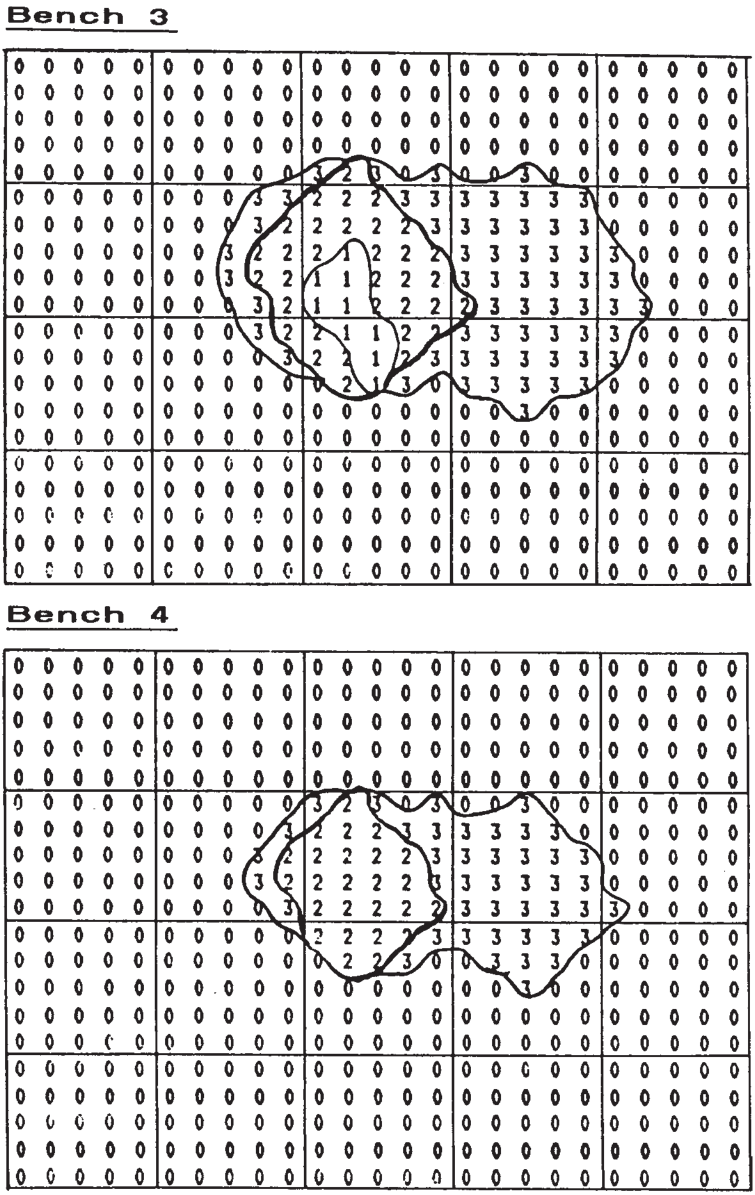


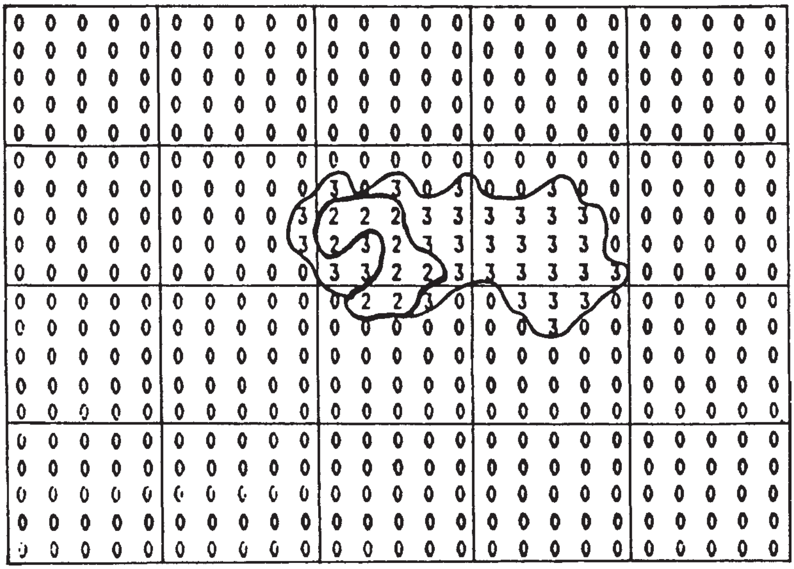
Figure 6.55. (Continued).

6.10 PUSH BACK DESIGN

6.10.1 *Introduction*

The first step in the practical planning process is to break the overall pit reserve into more manageable planning units. These units are commonly called sequences, expansions, phases,

Bench 5



Bench 6

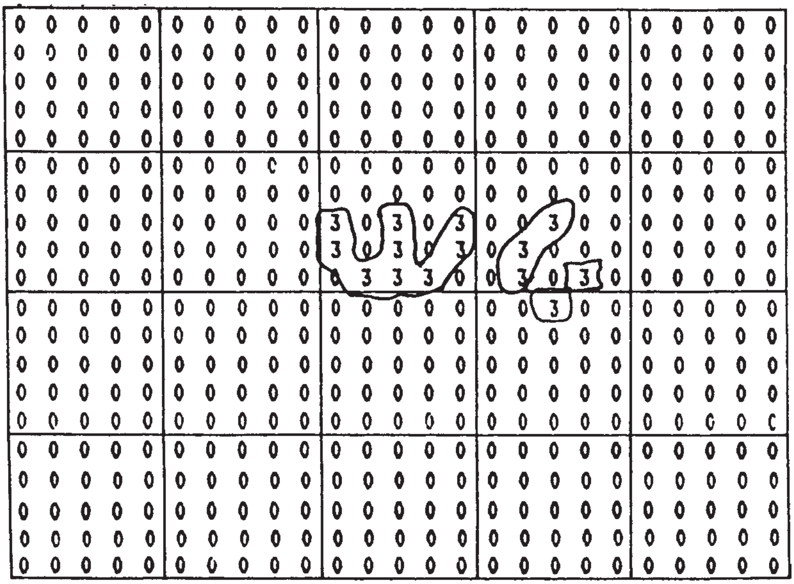


Figure 6.55. (Continued).

working pits, slices or push backs. In the beginning quite coarse divisions covering periods of several years may be used. They are a preliminary attempt to relate

the geometry of mining

to

the geometry of the ore distribution.

Table 6.48. Level by level statistics for the optimum schedule under constraint set 1 (Dagdalen, 1985).

	Year 1			Year 2			Year 3		
Bench	Ore blocks	Ave. Grade	Waste blocks	Ore blocks	Ave. grade	Waste blocks	Ore blocks	Ave. grade	Waste blocks
1	1	6.10	42	0	0	50	0	0	104
2	9	2.88	15	0	0	44	0	0	75
3	9	4.20	0	1	0.6	32	0	0	63
4	0	0	0	11	4.56	14	2	2.60	41
5	0	0	0	9	4.60	0	7	2.07	25
6	0	0	0	0	0	0	14	3.57	0
Total	19	3.67	57	21	4.39	140	23	3.03	308

Table 6.49. Production summary under constraint set 1 (Dagdalen, 1985).

Year	Ore grade (% Cu)	Ore	Tonnage waste	Total	SR
1	3.67	712,500	2,137,500	2,850,000	3.0
2	4.39	787,500	5,250,000	6,037,500	6.7
3	3.03	862,500	11,550,000	12,412,500	13.4
Total		2,362,500	18,937,500	21,300,000	8.0

Phase planning should commence with mining that portion of the orebody which will yield the maximum cash flow. Succeeding phases are ordered with respect to their cash flow contribution. Eventually the ultimate pit limits are reached.

By studying the ore grade distribution (particularly as depicted on bench plans) and the topography the mining engineer can, in most cases, arrive at a logical pit development strategy in a relatively short time. Figure 6.57 shows a two dimensional representation of the phases used to extract an ore reserve (Mathieson, 1982). The extraction sequence proceeds from that phase having the highest average profit ratio (APR) to the lowest. In this case they proceed in alphabetical order A to G.

$$\text{Profit ratio} = \frac{\text{Revenue}}{\text{All costs}}$$

The incremental profit ratio, computed at the final pit boundary is 1.

A basic overview of the steps involved in phase planning (Mathieson, 1982) are indicated below. These will be expanded upon in the succeeding sections.

1. Before design work is initiated some preliminary judgements must be made regarding
   – the probable maximum ore and waste mining rate required in a given phase,
   – the size and type of equipment to be used. This determines the required minimum operating bench width necessary,
   – appropriate working, interramp, and final slope angles.

2. Using the constraints given in step 1, the mining engineer then proceeds to design a series of phases in some detail, complete with haul roads. He ensures that ramp access

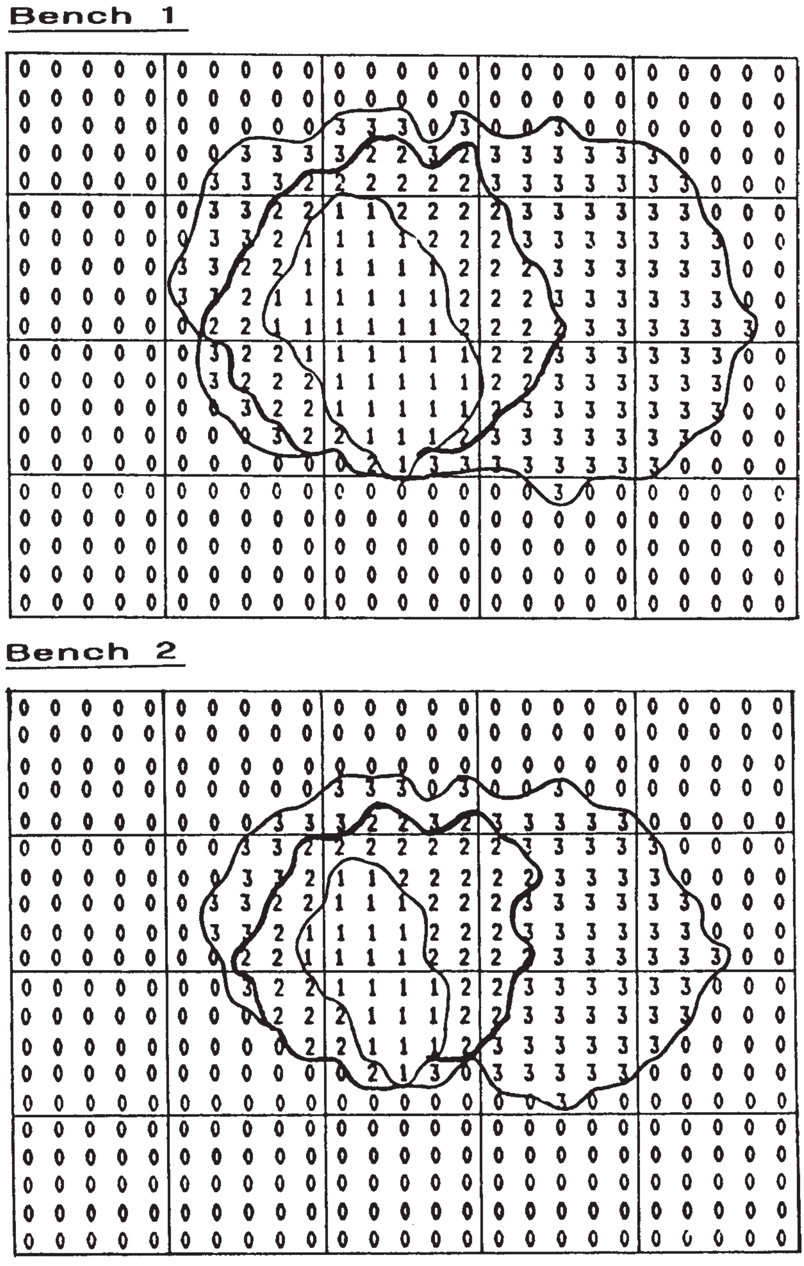


Figure 6.56. Proposed mining sequence for years 1, 2 and 3 under constraint set 2 (Dagdalen, 1985).

to each active bench is provided. The transition between phases is carefully planned. The designer is not constrained by having to include a certain quantity or number of years of ore supply in each phase. The variable ore and waste quantities by phase will be subsequently scheduled.

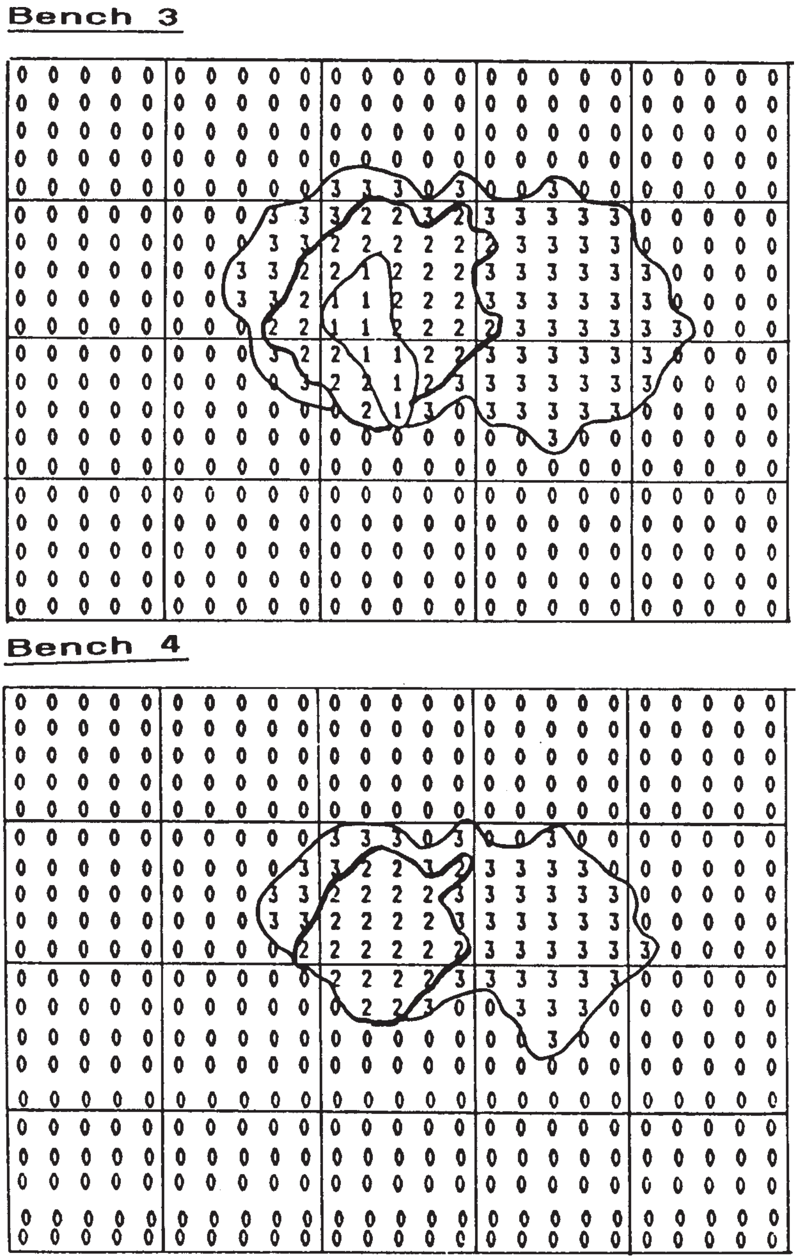
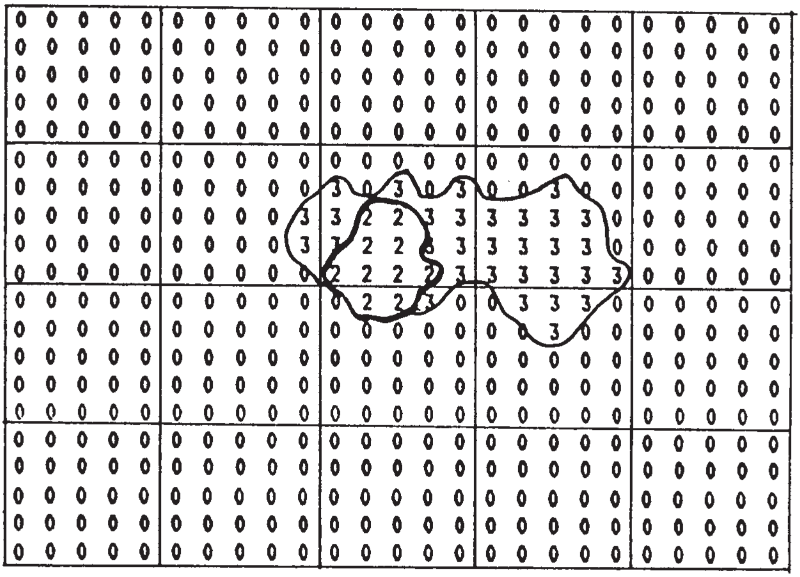


Figure 6.56. (Continued).

3. Once complete, the phase designs are transferred onto the bench plans. Bench by bench tonnages and grades are evaluated. A tonnage-grade inventory for each phase of the logically defined pit development sequence is then available.

4. The last task is to determine annual mining schedules based on mill feed or product requirements.

Bench 5



Bench 6

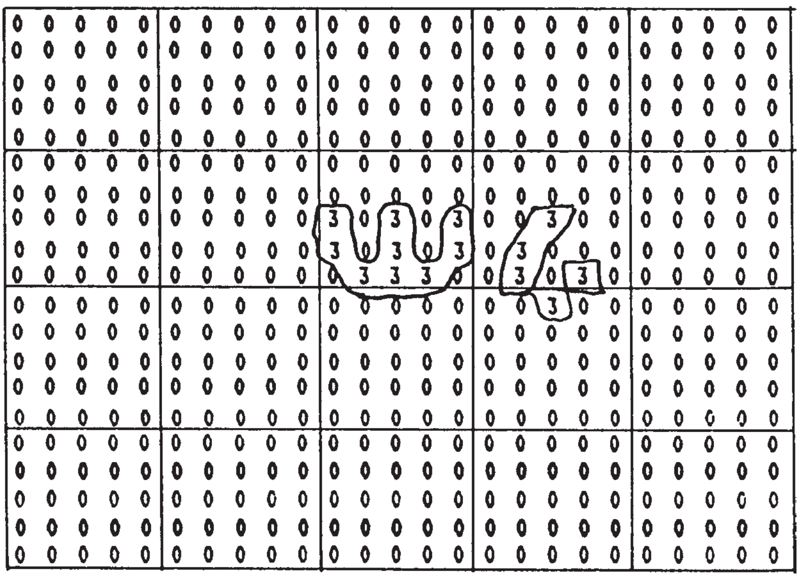


Figure 6.56. (Continued).

It may be necessary to repeat this process several times before an adequate plan is developed.

The development of phases makes it possible to determine the amount of pre-production stripping required. Although the natural tendency is to keep the amount of such stripping

Table 6.50. Level by level statistics for the optimum schedule under constraint set 2 (Dagdalen, 1985).

	Year 1			Year 2			Year 3		
Bench	Ore blocks	Ave. Grade	Waste blocks	Ore blocks	Ave. grade	Waste blocks	Ore blocks	Ave. grade	Waste blocks
1	1	6.1	42	0	0	50	0	0	104
2	9	2.88	15	0	0	44	0	0	75
3	9	4.20	0	1	0.6	31	0	0	64
4	0	0	0	10	4.16	13	3	4.60	42
5	0	0	0	10	3.34	0	6	3.73	25
6	0	0	0	0	0	0	14	3.57	0
Total	19	3.67	57	21	3.60	138	23	3.75	310

Table 6.51. Production summary under constraint set 2 (Dagdalen, 1985).

Year	Ore grade (% Cu)	Ore	Tonnage waste	Total	SR
1	3.67	712,500	2,137,500	2,850,000	3.0
2	3.60	787,500	5,175,000	5,962,500	6.6
3	3.75	862,500	11,625,000	12,487,500	13.5
Total		2,362,500	18,937,500	21,300,000	8.0

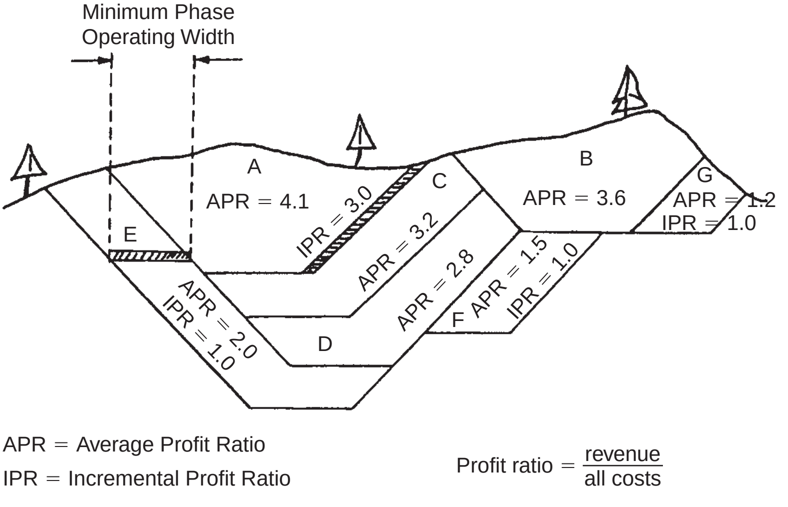


Figure 6.57. Pit sequencing in order of decreasing value (Mathieson, 1982).

to a minimum since it appears as a major negative cash flow early in the mining process, it is important that enough work be done to (1) expose a sustaining ore supply, and (2) keep the mine in a condition that allows it to be operated efficiently at all times. Sometimes the preproduction work is contracted. However, in new large tonnage truck-shovel mines, the pre-production period provides an opportunity to build an organization and to gain operating experience.

6.10.2 *The basic manual steps*

This series of steps in manual push back design has been provided by Crawford (1989a).

1. Start with the ultimate pit limit design.

– Develop detailed data of ore grade and stripping distributions for various cutoff grades in zones around the designed pit circumference and in pit shell progression between the beginning surface topography (or pit surface) and the design pit limit. These data should include locations of ore zones (these vary with cutoff) and the impact of the differences between operating and ultimate pit slopes. Of particular interest should be locating high ore grade and low stripping zones on level plan maps and cross-sections.

– Manual planning methods are essentially trial and error approaches.

2. Planning goals typically comprise one or more of the following:

– Maximize NPV economics.

– Provide stable cash flow patterns.

– Uniform ore grade, grade decline curve, or high grading. Frequently high grading is used during the initial investment pay back period. The level(s) of ore grade will be related to the cutoff criteria.

– Uniform stripping ratio or classic stripping ratio curve.

– Uniform total tonnage rate (ore + waste).

– Uniform ore tonnage rate.

– Uniform or variable rate of product output.

3. Operating design criteria for push back design:

– Operating and remnant bench widths.

– Slopes between operating benches and roads.

– Overall operating slope.

– Road widths and grades.

– Bench height.

Some typical values are as follows:

– Remnant bench width equal to bench height.

– Push back widths normally 200–500 ft depending on size of pit and orebody characteristics.

– Minimum push back widths (single cut passes) about 80 ft for small equipment and 135–150 ft for 25–30 yd shovels and 150–200 ton trucks.

– Road widths 50–80 ft depending on width of equipment. Maximum road grade 8–12 percent.

4. Laying out of one to several push backs. Evaluating whether they satisfy the goals, individually and collectively, is a more or less cut and try process. Normally a push back is laid out according to the operating criteria in plan and cross-section views. The plan view approach is used here. The operating criteria are expressed in geometric parameters (feet and degrees). A useful tool is to make a scaled bench crest and toe pattern including operating bench and road widths.

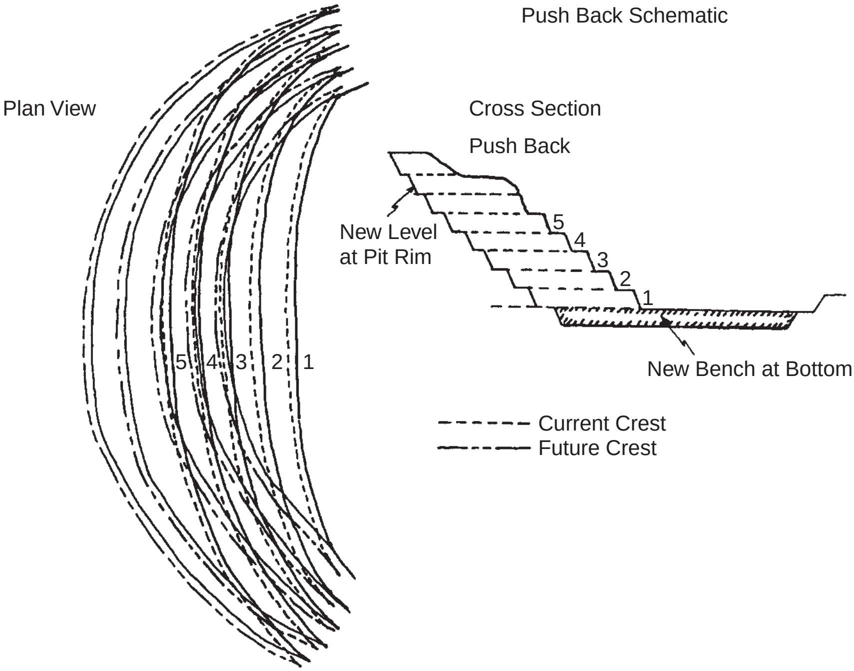


Figure 6.58. Push back schematic in plan and section views (Crawford, 1989a).

5. The push backs are shown on plan view maps as a progression of movements of bench level toes and crests from initial topography to ultimate pit limit (see Fig. 6.58). New levels are created as the push back progresses at the pit rim and at the bottom. New bottom levels are established on the basis of minimum level size and ore grades. Normally, new bottom levels are encouraged by the need to hold stripping at reasonable levels. In addition to pit geometry, ore/waste interface lines for the selected ore cutoff must be plotted.

6. Calculations of volumes of ore and waste are done using a planimeter to measure areas, and the average grades within push backs are determined. The volumes of material to be removed from each bench are based on the average of the areas encompassed by the movements of the bench crest and toe from initial to new position by a push back, and the average bench height for the zone covered. Ore and waste volumes are calculated separately. The average ore grade is determined by averaging the block values within the ore zones. In multiple push backs a push back serves as the initial location for a subsequent push back. The calculated values are evaluated against the various goals for acceptability of the individual push back or series of push backs.

7. For plan view calculations, the planimeter is used to determine the areas of crest and toe movements; commonly called crest to crest and toe to toe calculations. If the pit geometry is sufficiently regular only the toe to toe calculation may be necessary. To achieve accurate results, the calculation of volumes for new levels at the pit rim or at the bottom, along irregular pit rim elevations, for roads, at the ultimate pit intercept and for irregular bench heights require special planimeter techniques. The key is to divide up the volumes to be

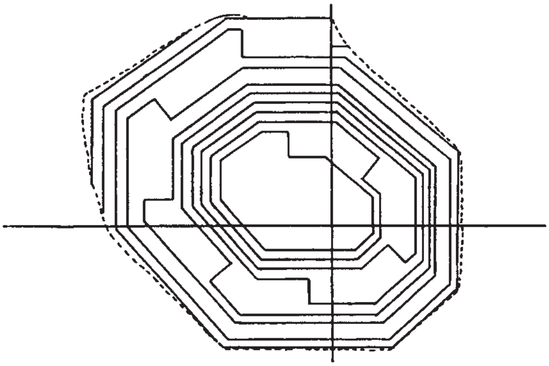


Figure 6.59. The Phase I pit with haulage road (Mathieson, 1982).

calculated into rectangular, parallelogram type solids with flat tops and bottoms and reliable average heights. The areas to be calculated must be closed polygons. The geometric layout prior to calculation is critical for accurate results. All the benches and roads must be described in the form of crests, toes, and average heights. Each bench and its related parts are calculated separately. Frequently, the drawing of a few cross-sections is helpful to keep the plan view drawing from becoming too confusing.

6.10.3 *Manual push back design example*

This section describes the construction procedure involved in designing a push back together with the layout of the main haulage road. An initial Phase I pit (Figure 6.59) already exists (Mathieson, 1982). The following information applies:

- The bench toe lines are shown
- Bench height = 45 ft
- Toe-toe distance = 40 ft
- Road width = 120 ft
- Road grade = 10%

Careful examination of the bench plans and the sections have indicated that the push back should involve the south and east portion of the pit. The north and northeast sections of the pit together with their portion of the haulroad will remain unchanged. The final pit design for Phase II must conform with this current geometry. It has been decided that the width of the push back should be 320 ft. This is a multiple of the basic 40 ft dimension and provides the desired

- tonnage for the phase, and
- operating space.

Figure 6.60 shows the basic push back area and the region where no changes will occur. The toe of the bottom bench has been described by a series of straight line segments. It is not necessary that they be straight lines but this facilitates the construction for this example. In Figure 6.61, that part of the Phase I pit involved directly in the push back has been removed. Sectors A and B will be modified for a smooth transition into the new pit. In Figure 6.62 the initial construction lines are shown. Lines a–b and c–d are drawn at the desired push back

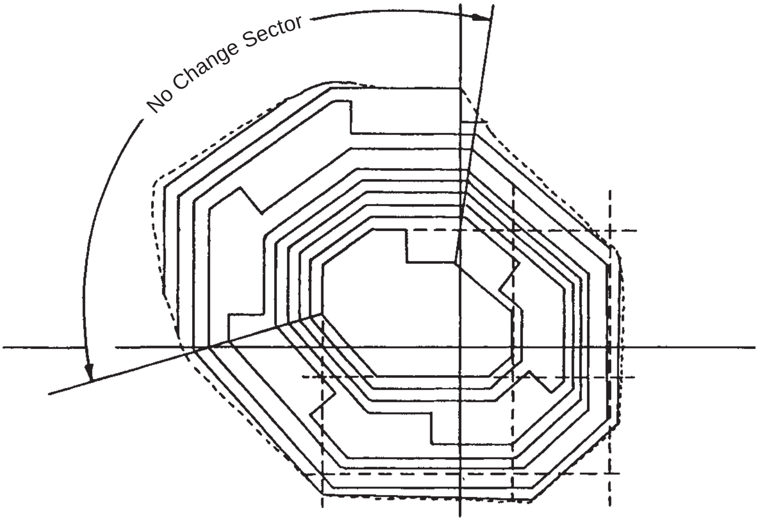


Figure 6.60. Construction lines for the push back superimposed.

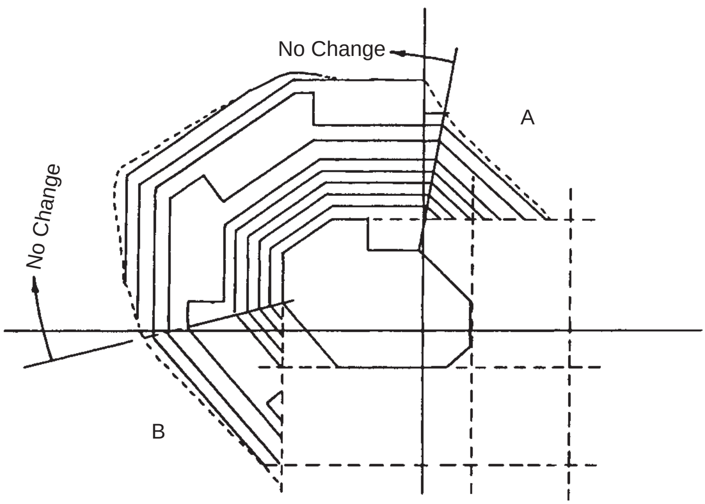


Figure 6.61. The push back section removed. Sectors A and B are transition regions.

distance of 320 ft parallel to the existing pit bottom lines. Lines f–a and b–c which form the pit corners are drawn at 45°. They are parallel to the corresponding lines of the current pit and at a distance of 320 ft. Line d–e has been drawn as shown recognizing that the pit will become more elongated in the east-west direction on that side. In Figure 6.63 the initial attempt at a pit bottom design is shown. The transitional sectors A and B have been removed

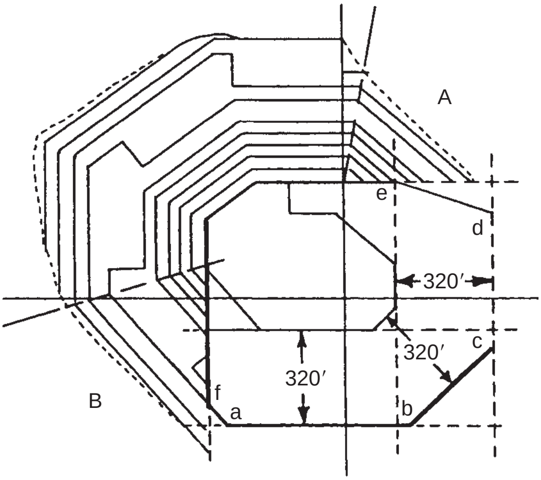


Figure 6.62. The dimensions used for constructing the new pit bottom.

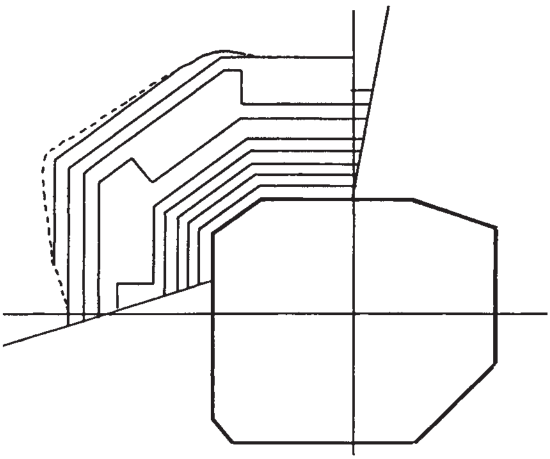


Figure 6.63. The initial design for the bottom of the Phase II pit.

from the drawing. To assist in constructing the upper bench from this base pit, lines have been drawn at the ends of the line segments perpendicular to the segment. This is shown in Figure 6.64. From each corner a line bisecting the angle described by the two radiating lines is drawn.This is shown by the heavy lines in Figure 6.65. A series of lines spaced at 40 ft (the toe-toe distance) are drawn parallel to those of the pit bottom. They are extended to join those of the existing pit (if appropriate) or to the corner lines. The drawing of the lines for the right hand side of the pit has been delayed pending the extension of the haulroad. In Figure 6.66, haulroad extension is shown. Due to the 10% grade and the 45 ft bench height, the distance between the adjacent road elevation lines shown is 450 ft. Figures 6.67 and 6.68 show the extension of the road down to the pit floor. In Figure 6.68 it can be seen that material in sector

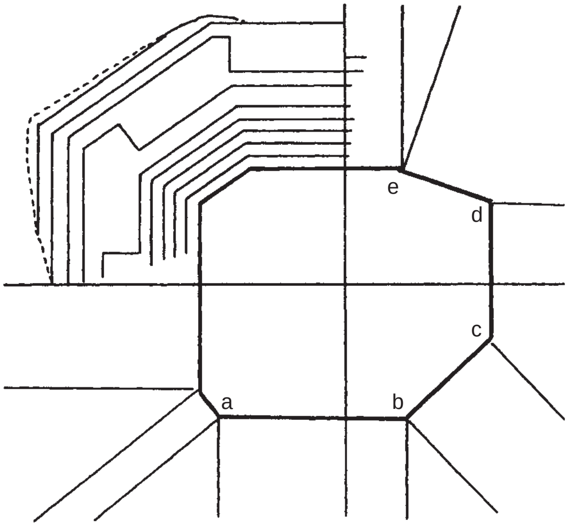


Figure 6.64. Construction of normals at the segment end points.

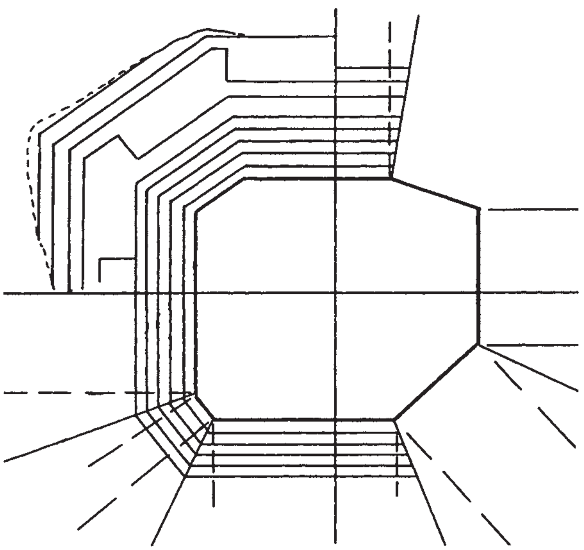


Figure 6.65. Construction of toe lines representing higher benches.

C must be mined to allow the road to daylight. A straight stretch having a length of one road width (120 ft) is added (Fig. 6.69). From this point (g), three segments g–h, h–i, and i–d have been drawn connecting to segment d–e. This provides the trucks with smooth access to the pit bottom and facilitates future road extension as the pit is deepened. As before, the bisecting lines are drawn from the segment corners (Fig. 6.70). In Figure 6.71, the parallel bench lines are added and the lower design is completed. As can be seen in Figure 6.71, there is no access

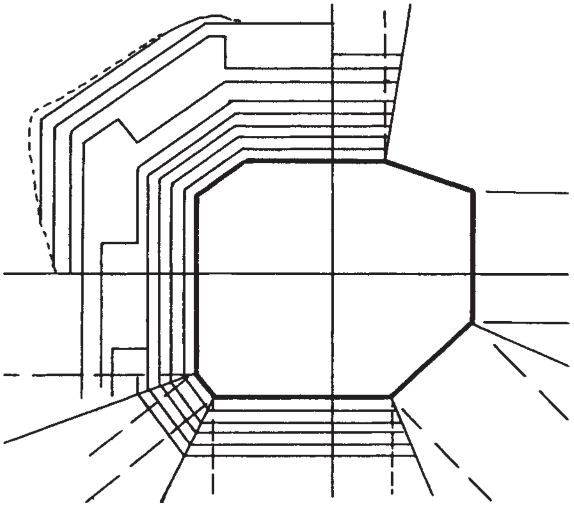


Figure 6.66. Extension of the road into the new phase.

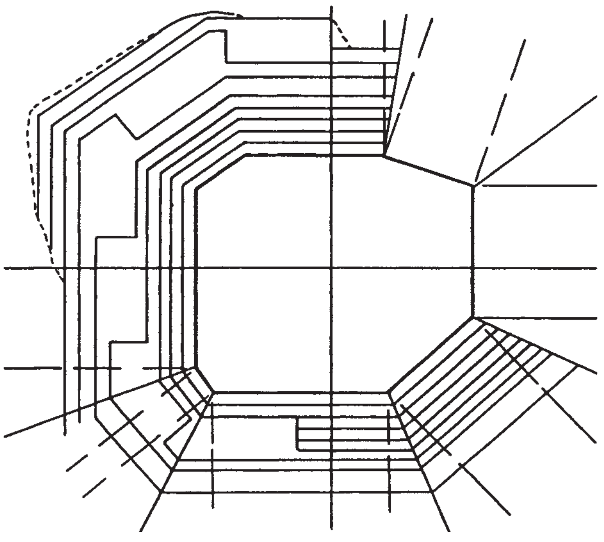


Figure 6.67. Extension of the road along the south side of the pit.

to the upper benches on the southeast side of the pit. This is corrected in Figure 6.72 by the addition of a road. The intersection of the Phase II pit with the surface topography is also shown.

With a workable design completed, one can now examine the tonnages involved. This is done on a bench by bench basis. Figure 6.73 shows the resulting bench geometry changes between the Phase I and Phase II pits. Figure 6.74 is the grade block model for bench 3835 with the toe lines for the Phase I and II pits superimposed. The next step in the process is to determine the grade-tonnage distribution for each level. Each of the complete blocks in the block model has plan dimensions 100 ft × 100 ft. For these, since the scale used is

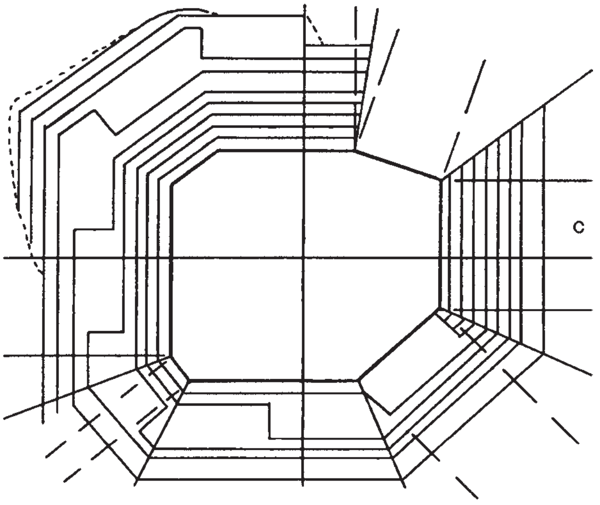


Figure 6.68. The road reaches the pit bottom elevation.

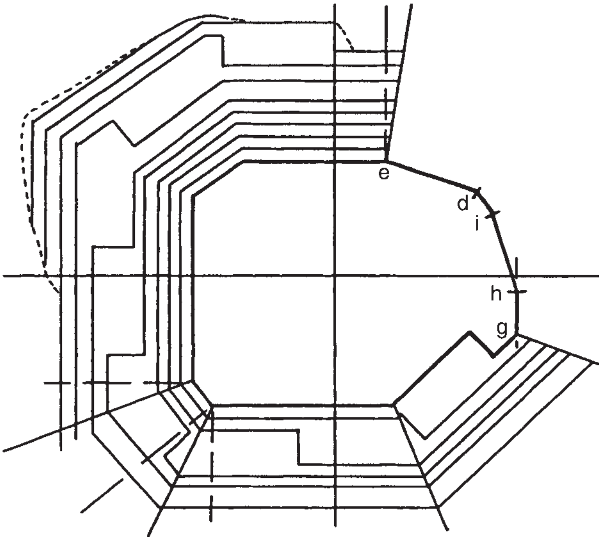


Figure 6.69. Final segments added to the pit bottom.

$1'' = 200$ ft, the block plan area is 0.25 in^2. However, in viewing Figure 6.74 it can be seen that

– the toe lines create many partial blocks,
– there are often several adjacent block with the same grade.

To save time, the contiguous blocks of the same grade are first identified. The area involved is determined using a planimeter. The number of square inches obtained is written on the bench map (see Fig. 6.75). Next the areas of the individual whole or partial blocks are found. These values are also entered on to the map. A summary sheet such as shown in Table 6.52 is prepared for the bench. The planimetered areas (in^2) are converted into square feet (ft^2)

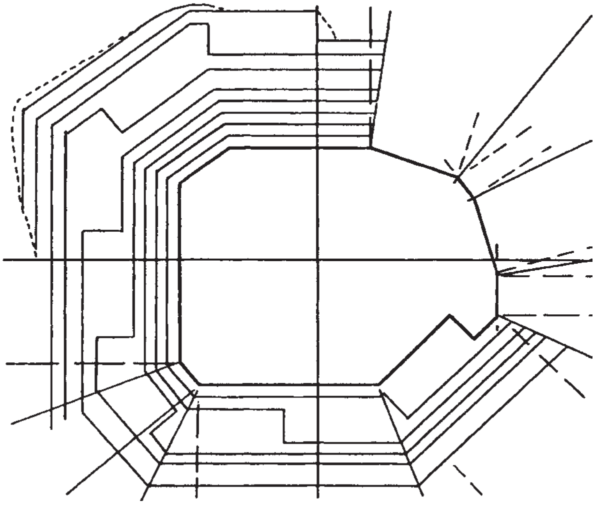


Figure 6.70. Construction at the segment corners.

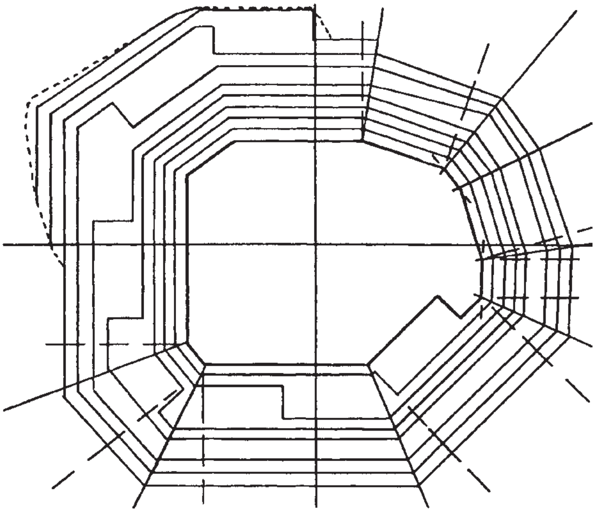


Figure 6.71. Road to pit bottom completed.

using the scale factor and then into tons via the bench height (45 ft) and the tonnage factor (12.5 ft^3/ton). Hence

$$1\ \text{in}^2\text{area} = \frac{200\ \text{ft} \times 200\ \text{ft} \times 45\ \text{ft}}{12.5\ \text{ft}^3/\text{ton}} = 144{,}000\ \text{tons}$$

This conversion factor is applied to the areas in Table 6.52.

The total area on this bench involved in phase II mining is 17.57 in^2 hence the tonnage is

$$\begin{aligned}\text{Tons} &= 17.57\ \text{in}^2 \times 144,000\ \text{tons/in}^2\\ &= 2,530,080\ \text{tons}\end{aligned}$$

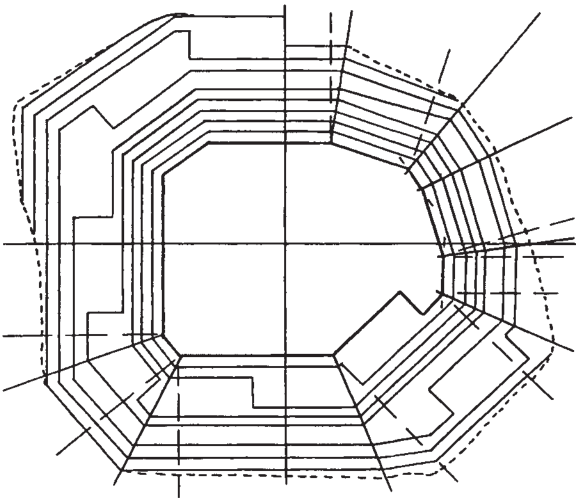


Figure 6.72. Road access to upper benches added.

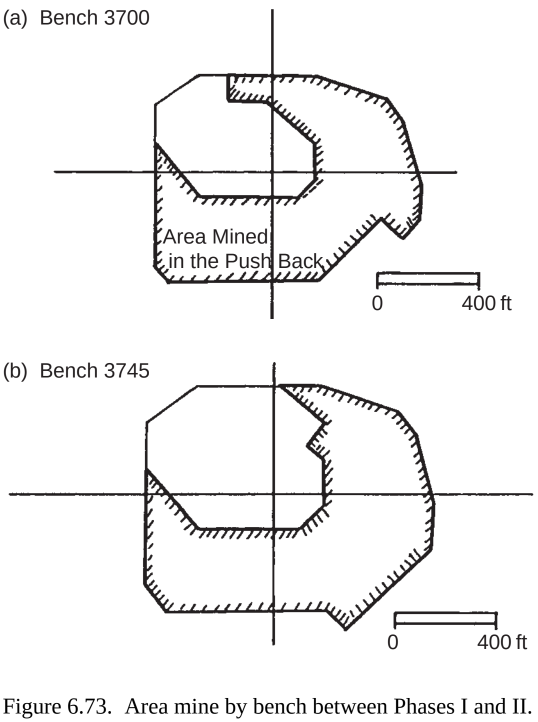


Figure 6.73. Area mine by bench between Phases I and II.

Sometimes grade classes are used. In Table 6.53 an interval of 0.05% Cu has been employed.

Expressed in this form it is quite easy to summarize the overall results from the entire push back. Applying one or more cutoff grades, the amount of material falling in each category is found.

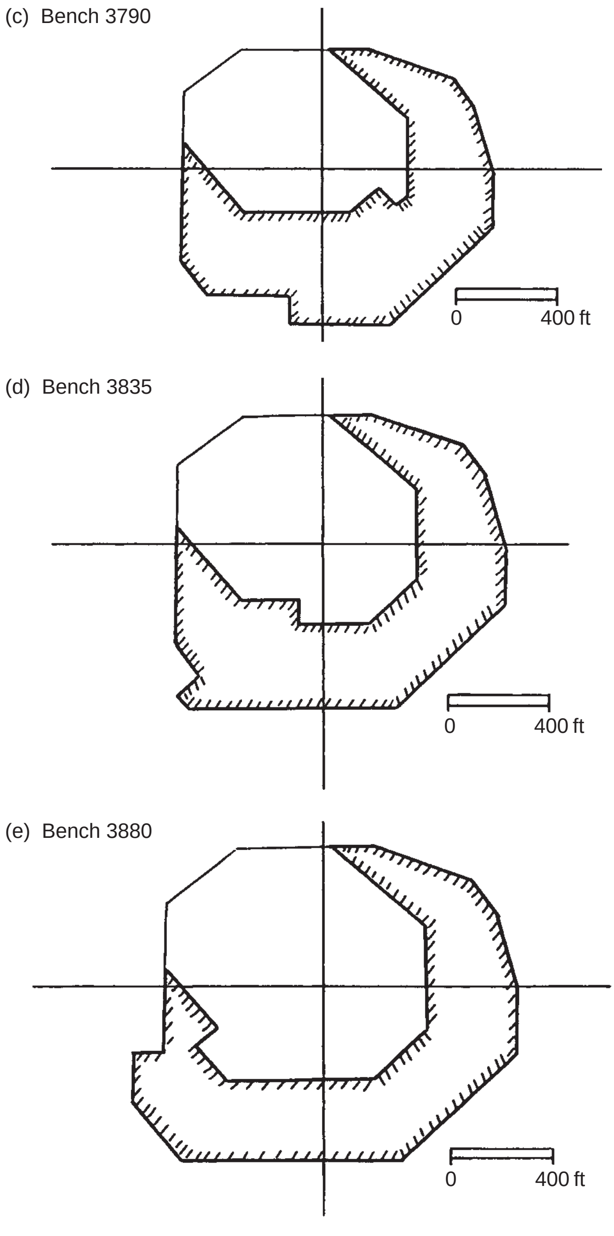


Figure 6.73. (Continued).

These results unfortunately may be unacceptable. For example, the overall stripping ratio might be too high, the average grade too low, the tonnage too low, etc. Another push back design must then be done in the same fashion. Eventually an acceptable plan will emerge.

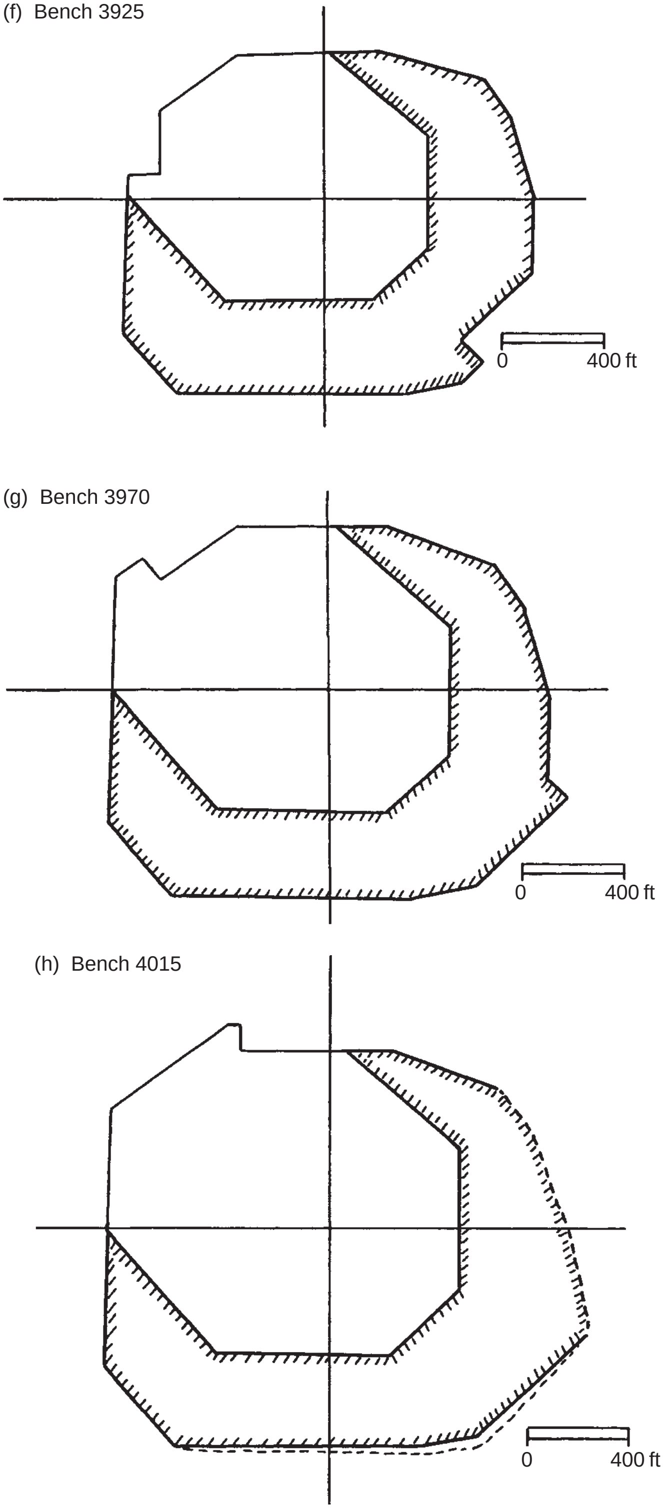


Figure 6.73. (Continued).

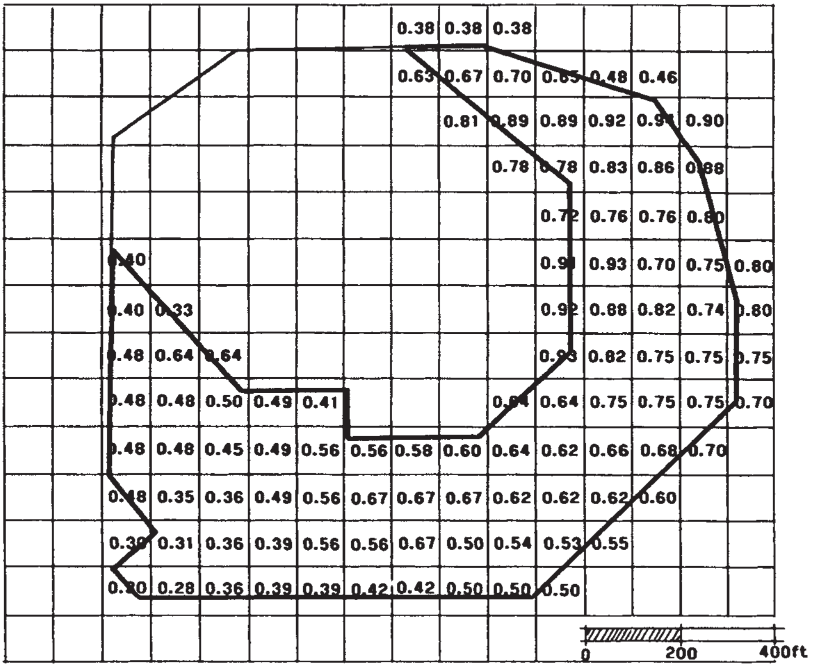


Figure 6.74. The final pit outline has been superimposed on the grade block model for bench 3835.

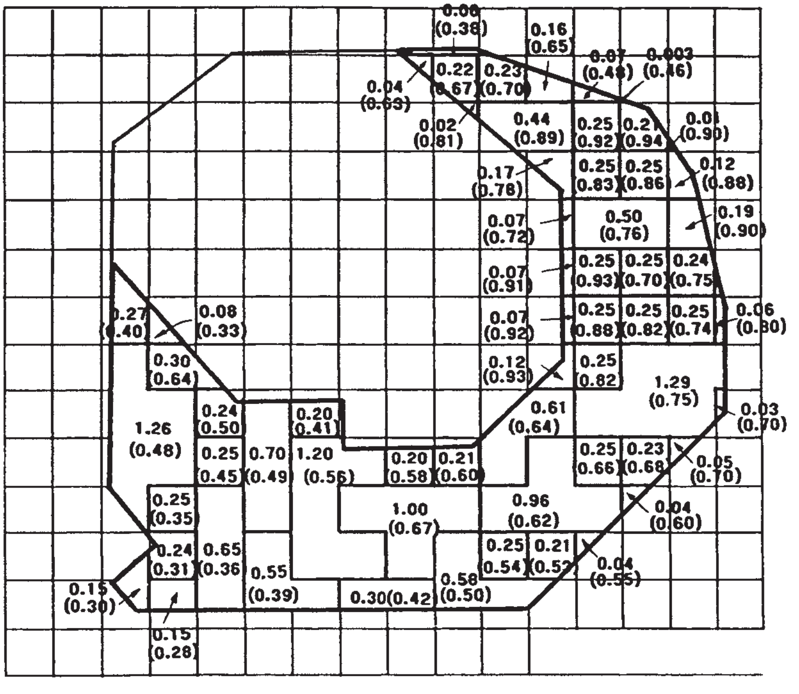


Figure 6.75. The areas are calculated.

Table 6.52. Summary tonnage-grade for the Phase II mining of bench 3835.

Grade (% Cu)	Plan area (in^2)	Tons (10^3)	Grade (% Cu)	Plan area (in^2)	Tons (10^3)
0.28	0.15	21.6	0.63	0.04	5.8
0.30	0.15	21.6	0.64	0.91	131.0
0.31	0.24	34.6	0.65	0.16	23.0
0.33	0.08	11.5	0.66	0.25	36.0
0.35	0.25	36.0	0.67	1.22	175.7
0.36	0.65	93.6	0.68	0.23	33.1
0.38	0.06	8.6	0.70	0.56	80.6
0.39	0.55	79.2	0.71	0.07	10.1
0.40	0.27	38.9	0.74	0.25	36.0
0.41	0.20	28.8	0.75	1.53	220.3
0.42	0.30	43.2	0.76	0.53	76.3
0.45	0.25	36.0	0.78	0.17	24.5
0.46	0.01	1.4	0.80	0.25	36.0
0.48	1.33	191.5	0.81	0.02	2.9
0.49	0.70	100.8	0.82	0.50	72.0
0.50	0.82	118.1	0.83	0.25	36.0
0.53	0.21	30.2	0.86	0.25	36.0
0.54	0.25	36.0	0.88	0.37	53.3
0.55	0.04	5.8	0.89	0.44	63.4
0.56	1.20	172.8	0.90	0.01	1.4
0.58	0.20	28.8	0.91	0.07	10.1
0.60	0.04	5.8	0.92	0.32	46.1
0.61	0.21	30.2	0.93	0.37	53.3
0.62	0.96	138.2	0.94	0.21	30.2

Table 6.53. The data from Table 6.51 reorganized into grade intervals.

Grade interval	Area (in^2)	Tons
0.26–0.30	0.30	43,200
0.31–0.35	0.57	82,080
0.36–0.40	1.53	220,320
0.41–0.45	0.75	108,000
0.46–0.50	2.86	411,840
0.51–0.55	0.50	72,000
0.56–0.60	1.44	207,360
0.61–0.65	2.28	328,320
0.66–0.70	2.26	325,440
0.71–0.75	1.85	266,400
0.76–0.80	0.42	60,480
0.81–0.85	0.77	110,880
0.86–0.90	1.07	154,080
0.91–0.95	0.97	139,680
Total	17.57	2,530,080

It is obviously possible to evaluate various push back options without going to the trouble of including the haulroads. However, a considerable amount of material is involved in later adding such a road to the design and the results can change markedly.

6.10.4 *Time period plans*

Most of the pit planner's work (Couzens, 1979) is done on plan or bench maps. These show:
- – topography or surface contour,
- – location of ore,
- – geologic boundaries, and
- – design limits.

Pit composite maps showing the shape of the mine at the end of each planning period should be kept up. These enable the planner to:
- – avoid conflicts between features of the plan,
- – provide a picture of the access at each stage of development,
- – illustrate the actual working-slopes, operating room and spacial relationships between ore and waste.

The transition from phase plans to time period plans should be made as soon as the phase designs are complete enough to set the overall pattern. The yearly plans:
- – Enable definite production goals to be set in space as well as in quantities of material to be moved.
- – Allow better economic evaluations than the phase average provide.
- – Give a better definition of the relationship of the phases to each other as they overlap in the complete mine operation. They show actual operating slopes and haulage routes.

Figures 6.76, 6.77, and 6.78 show such a 3-year progression of benches in an open pit mine (Couzens, 1979).

The midbench contours have been plotted. The haulroads, stripping areas and part of the waste dumps are also shown. In this system the labelling of elevations is as follows (Couzens, 1979):

(a) Outside of the pit the contours are labelled with their true elevations.

(b) Inside of the pit:
- – The labelled elevations refer to the bench toe elevations.
- – The elevations of bench centerlines are one-half the bench height above the bench toe elevation. Thus it is the flat areas between center lines that are labelled.
- – On ramps, the bench centerlines cross the ramp halfway between benches. The labels are positioned at would be the actual bench elevation on the road.
- – For more explanation of the labelling, (see Section 4.7).

At an operating mine there will be a number of different plans covering different periods. The engineering staff is generally responsible for:
- – annual ore reserve estimation,
- – yearly or multi-yearly plans regarding the progression of the pit, changes in haul roads, etc.,
- – quarterly plans, and
- – monthly plans.

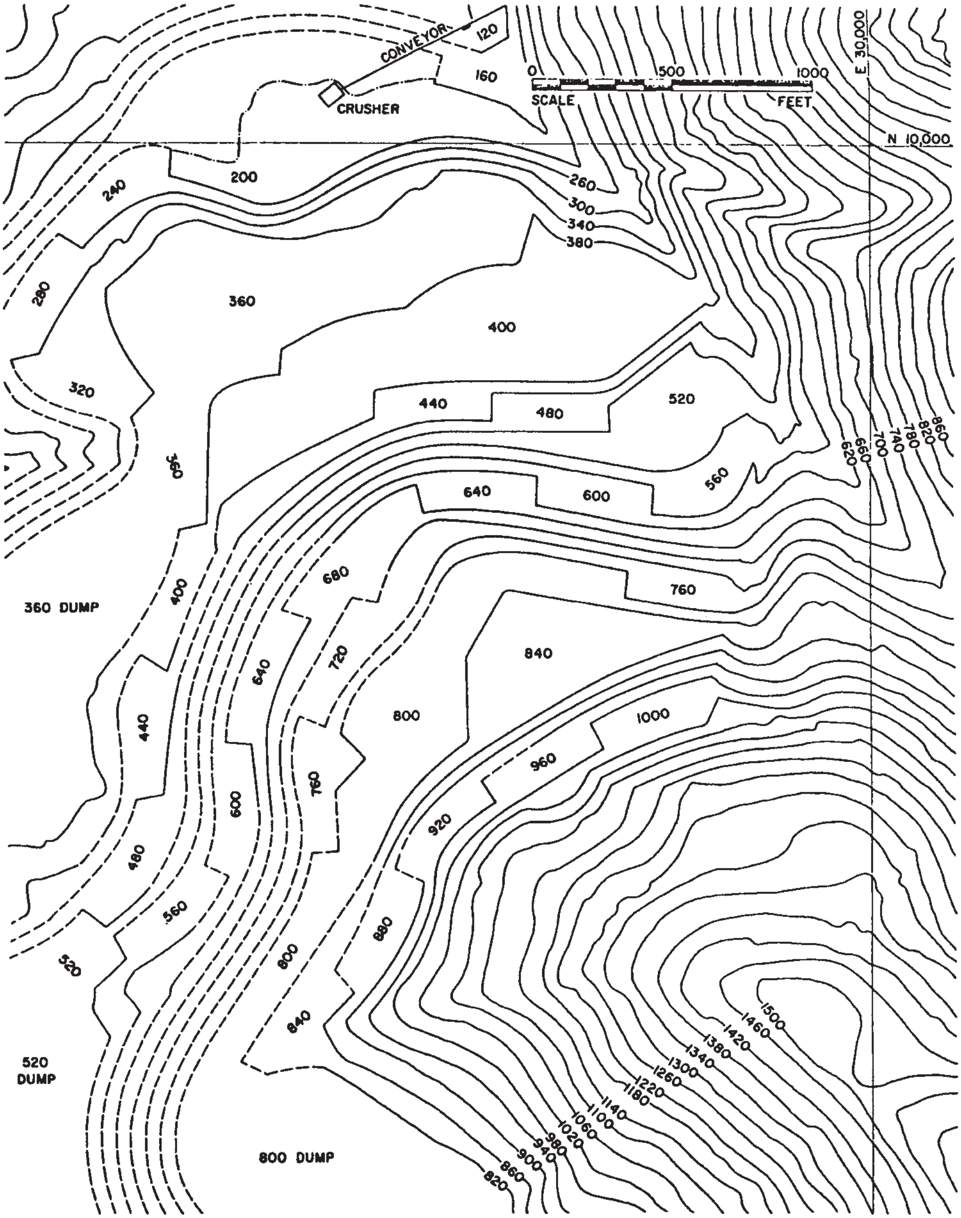


Figure 6.76. Bench composite for year 1 (Couzens, 1979).

The operating staff develop:
- weekly plans, and
- daily plans

within the longer range framework.

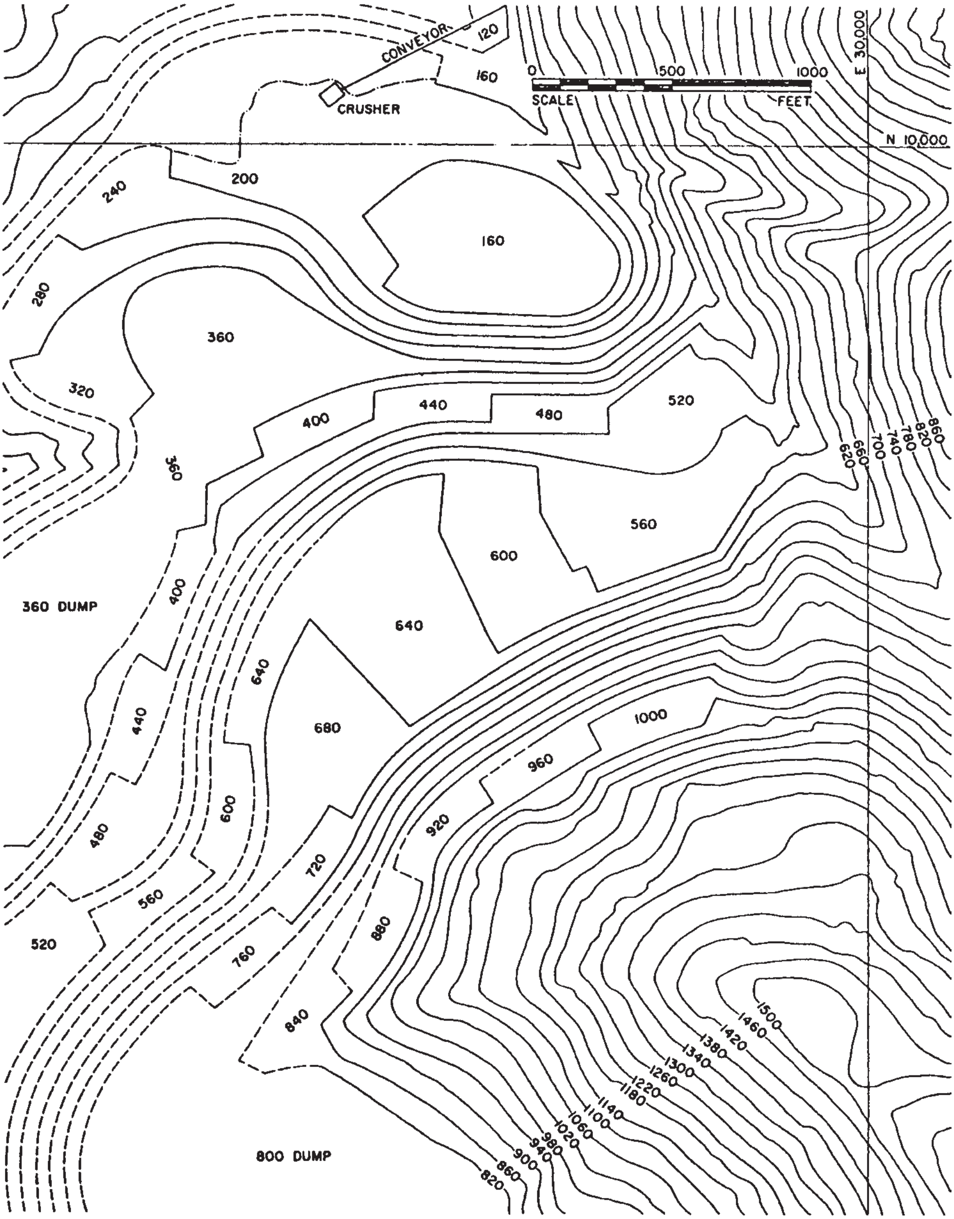


Figure 6.77. Bench composite for year 2 (Couzens, 1979).

6.10.5 *Equipment fleet requirements*

Once phase plans have been developed, equipment fleet requirements can be examined (Couzens, 1979). A graph showing the total tonnage movement and waste/ore ratios can be prepared. On such a graph, the planner can see what must be done to adjust or smooth out

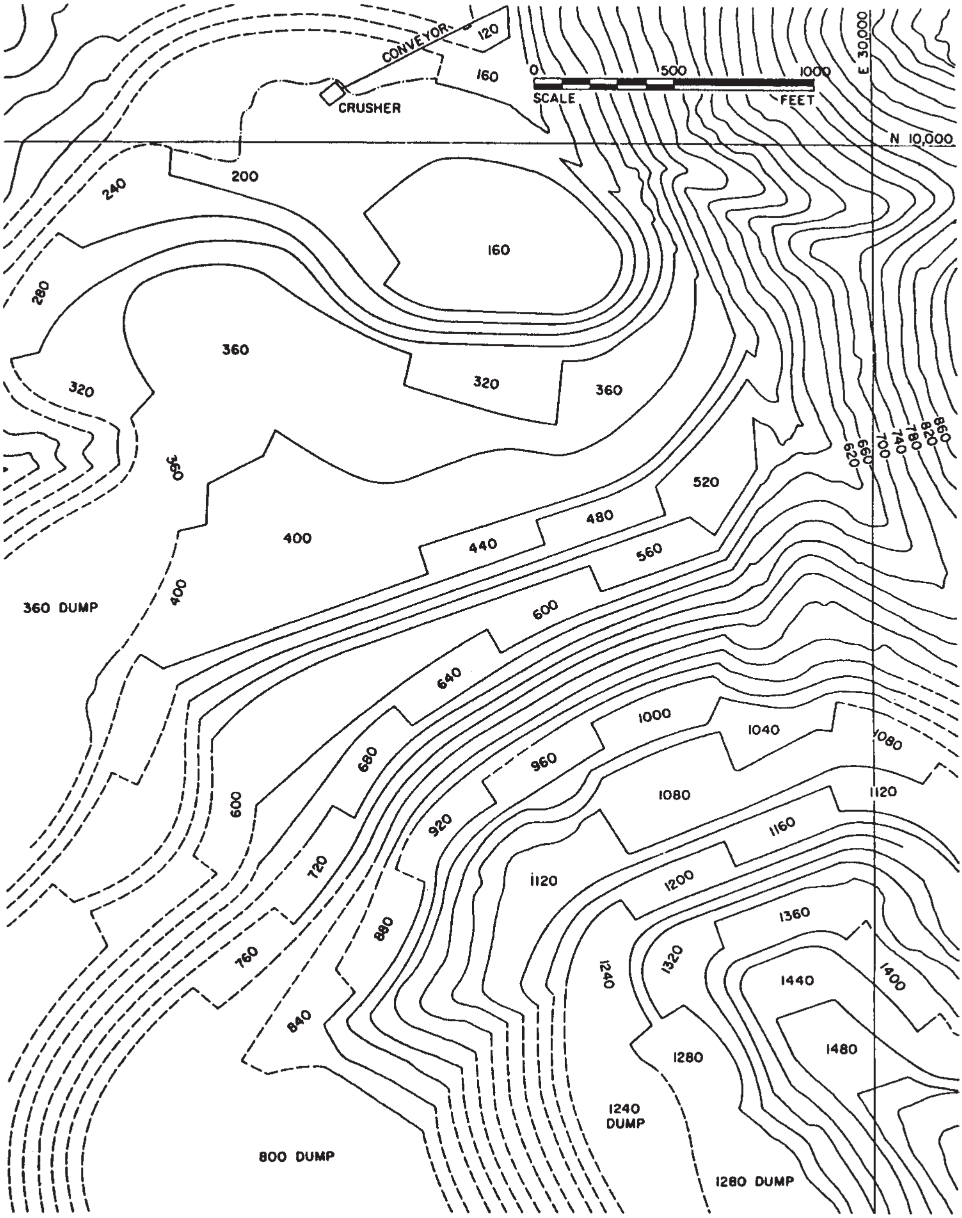


Figure 6.78. Bench composite for year 3 (Couzens, 1979).

the production. Figure 6.79 shows such a graph for a trial mining plan before smoothing has occurred. In this case the milling rate was constant and the plan was worked out to achieve:

- a good blend of ore,
- good ore exposure, and
- good operating conditions.

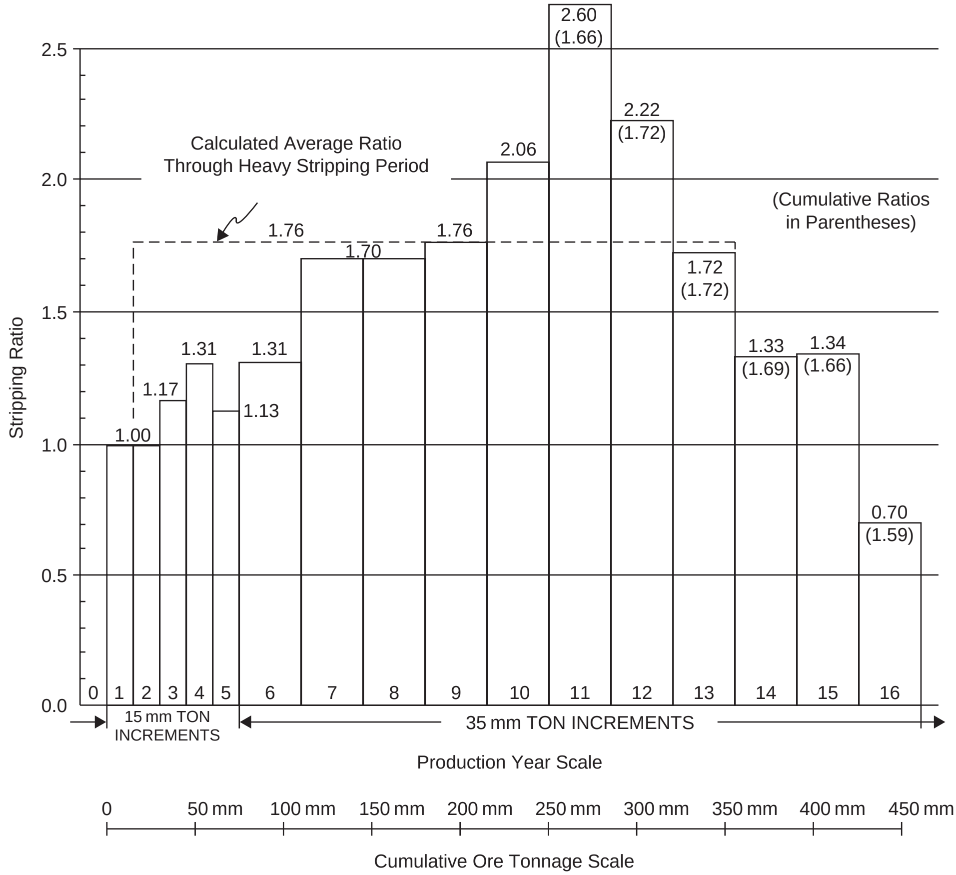


Figure 6.79. Waste/ore ratio versus time in a trial mining plan before smoothing (Couzens, 1979).

The amount of waste to be removed was determined by these conditions and as a result the waste/ore ratios vary. For the heavy stripping period, years 2–13, the average waste/ore ratio is 1.76. One attempt at replanning has been done in Figure 6.80. The peaks have been redistributed both earlier and later from their original positions.

Figure 6.81 illustrates a type of production scheduling graph. The various relationships between total tonnage movement, ore requirements, waste ratio, and shovel shifts are shown. A presentation to management of this type makes it possible to communicate the mining schedule better than just bare statements of tonnage and waste/ore ratios.

6.10.6 *Other planning considerations*

Dump planning is also a part of the mine planners job (Couzens, 1979). There are a number of factors which enter the scene at this stage:

– length of hauls,
– the required lifts,

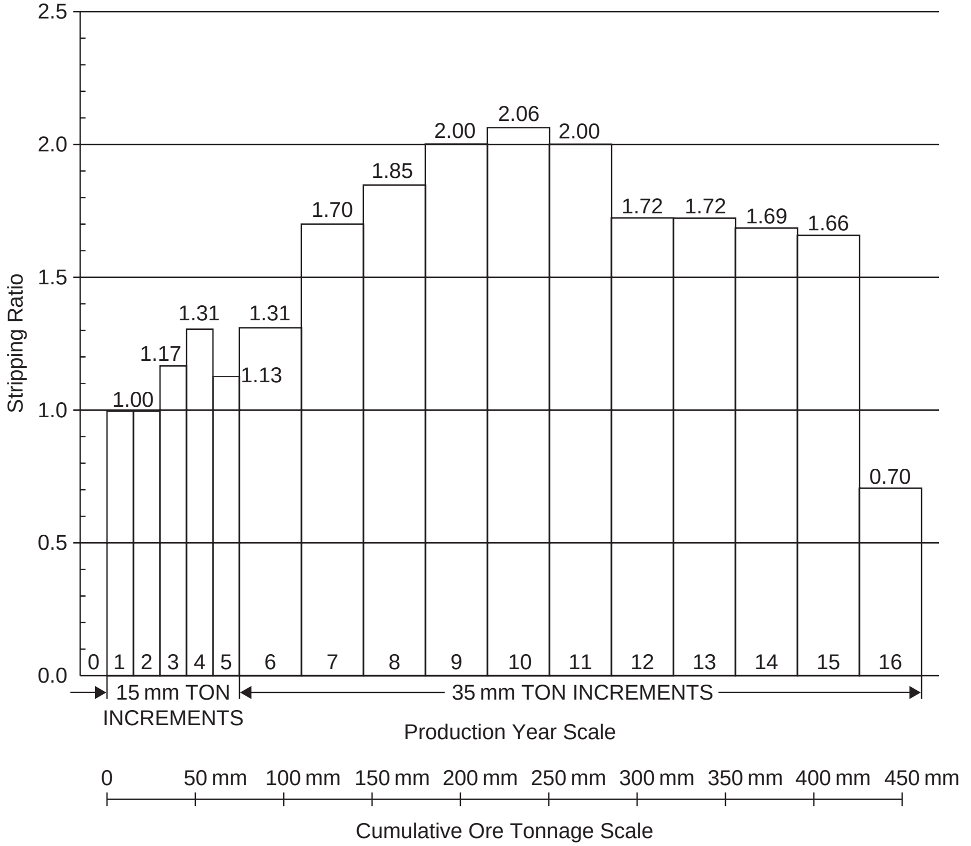


Figure 6.80. Waste/ore ratios versus time in a trial mining plan after smoothing (Couzens, 1979).

– the relationship between dumps and property constraints, other installations, drainage, etc., and

– reclamation and environmental requirements.

Pit planning should include an estimate of where the dumps are going to be at each stage. This should be done in connection with the haulage study. The planner can look at the trade-off between an additional lift and a longer haul. Dump planning can have an important bearing on pit planning, particularly in the haulage layout, scheduling and equipment estimating areas.

Water management planning forms a part of overall mine planning. In new pit design it is important to estimate the amount of expected water. This can be done by looking at rainfall records, drill logs and hydrologic reports.

Streams or even dry arroyos coming into the pit area may have to be diverted in order to avoid bringing surface water into the pit.

For environmental reasons the planner may have to consider what happens to water that is removed from the pit. Can it be discharged into natural drainage or must it be

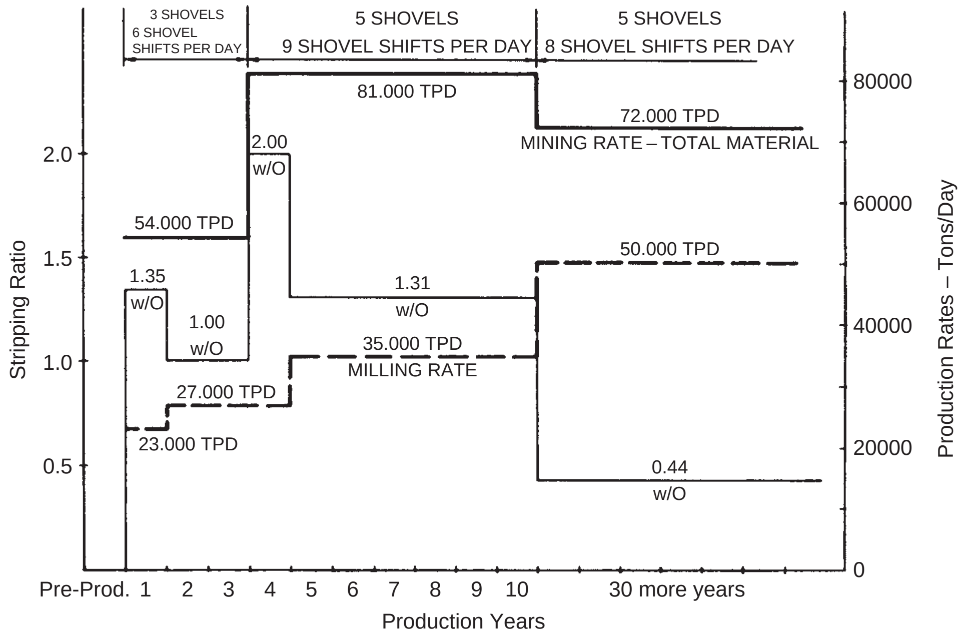


Figure 6.81. Example of a final production schedule graph based upon a balanced mining plan (Couzens, 1979).

impounded, treated, or recycled? Water affects blasting, equipment operation and maintenance, road construction and sometimes even ore quality. It is difficult to operate with much water in the pit. Water also has an adverse effect on slope stability. To manage the water, ditches, drains and/or dewatering systems may have to be included in the planning.

In summary, mine planning includes a rather wide spectrum of activities. It must be done carefully and thoroughly if the mine operation is to be successful.

6.11 THE MINE PLANNING AND DESIGN PROCESS – SUMMARY AND CLOSING REMARKS

In closing this chapter, it is appropriate and necessary to try and provide a perspective or helicopter view of the mine planning and design process. The individual component parts described in the preceding chapters and sections must be knit together if they are to function as a whole and produce the desired mine design.

Table 6.54 developed by Crawford (1989b) is a very useful summary of the steps that one goes through with a new orebody. In examining an expansion of a current mine, some simplifications arise due to the availability of better initial data. The steps are, however, basically the same. The iterative nature of the planning and design process has been very aptly termed circular analysis by Dohm (1979). The process and the included components are simply and rather elegantly presented in Figure 6.82.

Table 6.54. Logical sequenced approach for evaluating greenfield open pit ore deposits (after Crawford, 1989b).

1. Establish a long term price forecast and maximum practical marketable volume per year.
2. Make a geologic reserve assessment. Develop a grade versus tonnage curve for the contiguous portion of the orebody including overburden. Select a typical cut-off grade for the commodity (gold, copper, iron, etc.), type of deposit (deep, shallow, etc.), and likely mining/processing system. Determine an average grade and strip ratio and then determine average mill and mining rates for the maximum marketable volume per year or 10 year life whichever is lower.
3. Develop block net values and costs for both ore and waste using mining and milling rates ranging above and below those determined in No. 2. Incorporate capital costs and return considerations in the ore and waste block evaluations as feasible. Because timing and various inter-block interactions cannot be directly addressed, these will be only approximate.
4. Develop sets ultimate of pit designs flexing (varying) prices and/or milling/mining rates and costs to get logical concentric nests of pits. Use a block value ≥ 0 as cutoff.
5. Develop short range plans, using average operating slopes, within each ultimate pit. Holding costs and rates constant, flex the price starting above the price of ultimate pit and work down in increments. This approach should mine the best ore/waste combinations first. A set of short-range mining segments will be generated in this process and thereby fix the feasible mine geometry options for a given ultimate pit.
6. Develop production, revenue, operating and capital cost schedules over time for each short-range segment and over full mine life from development through closure. These data will not normally be derived from the block data directly but they must be related for consistency.
7. Optimize the value of each short range segment by flexing cutoff criteria and mining/ore processing rate being careful not to exceed the marketable maximum volume, using NPV methods. The production and economic schedules in No. 6 will be iterated to accomplish this. The choice of NPV discount rate will impact the results of the total evaluation process significantly.
8. Evaluate the NPV of the total pit plan segment sequence to confirm optimum.
9. Presumably the combination of ultimate pit, short-range pit plan segment sequence, and mining/ore processing rate yielding the maximum NPV regardless of mine life would be the preferred option for developing the deposit under evaluation.

Except for the simplest cases, there are often a large number of different combinations which are both possible and realistic. Of these, there is a small subset which would be acceptable. Eventually one must arrive at a ‘best’ few. Based upon technical, political, company cultural, or other reasons, one of these is eventually selected and the development phase is entered.

The rapid advances which have been made in the power, speed and friendliness of computerized design tools over the past few years have produced major changes in the activities performed by mining engineers in modern mining companies.

Largely relieved of routine number crunching and drafting, the design/planning engineer can focus on the task – problem solving – for which he/she has been educated/trained. Creative alternative solutions must be identified, developed and evaluated and recommendations made regarding an action path, if companies are to survive in a very competitive world. There should and must be some time for dreaming, perhaps not the ‘Impossible Dream’ but of innovative solutions to the tough problems facing mining today, and the even tougher ones coming. The continuing challenge of supplying the world’s mineral requirements at

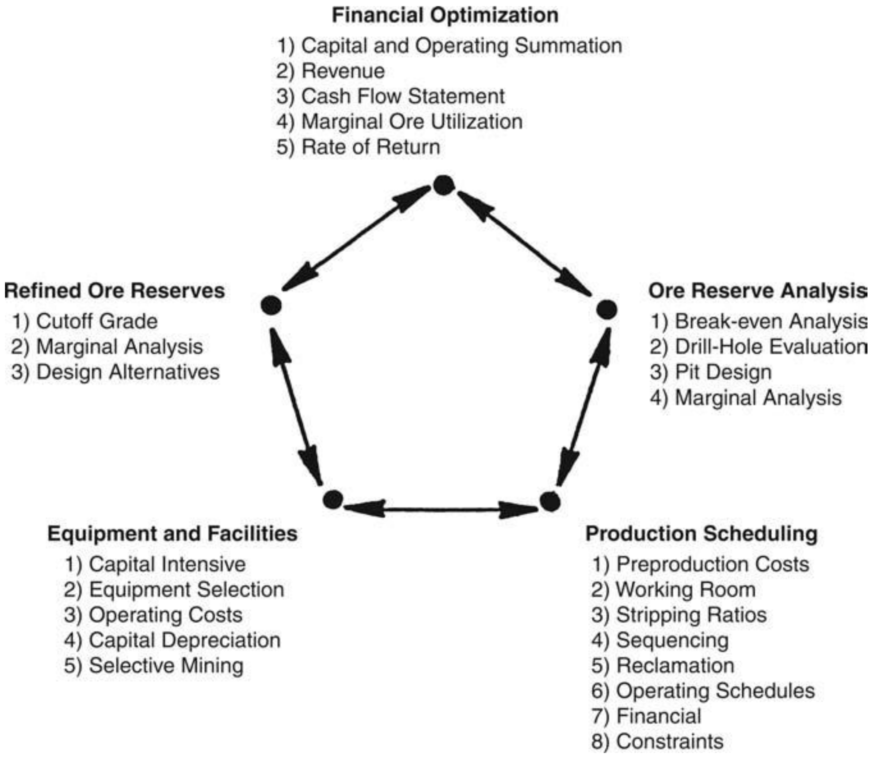


Figure 6.82. Circular analysis (Dohm, 1979).

reasonable cost under ever more stringent regulations will require the best minds and mines. Glückauf!

REFERENCES AND BIBLIOGRAPHY

Abzalov, M.Z., Menzel, B., Wlasenko, M., and J. Phillips. 2007. Grade Control at the Yandi Iron Ore Mine, Pilbara Region, Western Australia – Comparative Study of the Blastholes and Reverse Circulation Holes Sampling, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 37–43.

Akaike, A., and K. Dagdelen. 1999. A strategic production scheduling method for an open pit mine. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 729–738. Colorado School of Mines: CSM.

Allen, L., & M. Baumann. 2007. Interaction of grade and metallurgy simulations in assessing uncertainty for ore routing determinations in the Gold Quarry deposit. Paper 07-108. SME Annual Meeting. Feb 25–28. Denver, CO.

Anonymous. 1990. Orebody modeling and mine planning software. *Mining Engineering* 42(8): 976–981.

Anonymous. 2006. Tools of the trade – Software: Model behaviour (Part 1). Mining Magazine. February. pp. 12–16.

Anonymous. 2006. Tools of the trade – Software: Fingertip planning (Part 2). Mining Magazine. March. pp. 18–22.

Anonymous. 2006. Tools of the trade – Software: Screen control (Part 3). Mining Magazine. April. pp. 42–45.

Anonymous. 2007. Equipment – GPS: Pole precision. Mining Magazine. March. pp. 38–41.

Anonymous. 2010. Management in action – Mine scheduling: Planning for the future. Mining Magazine. January/February. pp. 22–28.

Appiah, P.B. & J.R. Sturgul. 1992. Optimal mine operating strategy for a fluctuating price. *Int. J. Surface Mining and Reclamation* 6(3): 121–128.

Arioglu, E. 1988. Examination of empirical formulae for predicting optimum mine output. *Trans. Instn. Min. Metall. (Sect. A*: *Min. Industry*) 97(1): A54–A55.

Armstrong, M., & A. Galli. 2011. A new approach to flexible open pit optimisation and scheduling. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 253–263.

Armstrong, M., Galli, A., & A.A. Ndiaye. 2009. A case study on the impact of hedging against foreign exchange risk and energy price risk. Project Evaluation Conference. Melbourne, Vic. April 21–21. pp. 27–33.

Asad, M.W.A., & R. Dimitrakopoulos. 2012. Performance evaluation of a new stochastic network flow approach to optimal open pit mine design – application at a gold mine JSAIMM. 112(7): 649–655.

Asad, M.W.A., & E. Topal. 2011. Net present value maximization model for optimum cut-off grade policy of open pit mining operations. JSAIMM. 111(11): 741–750.

Askari-Nasab, H., & E. Ben-Awuah. 2011. Integration of oil sands mine planning and waste management using goal programming. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 329–350.

Askari-Nasab, H., Eivazy, M. Tabesh, & M. Badiozamani. 2011. A mathematical programming model for open pit short-term production scheduling. Paper 11-038. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Baird, B.K., & P.C. Satchwell. 1999. Application of economic parameters and cutoffs during and after pit optimization. Paper 99-069. SME Annual Meeting. Mar 1–3. Denver, CO.

Baird, B.K., & P.C. Satchwell. 2001. Application of economic parameters and cutoffs during and after pit optimization. *Mining Engineering*. 53(2): 33–40.

Barnes, R.J. & L. Bertrand. 1990. Operational ore/waste classification at a surface gold mine. *Int. Journal of Surface Mining and Reclamation* 4: 26–29.

Bascetin, A., & A. Nieto. 2007. Determination of a mining cutoff grade strategy based on an iterative factor. Paper 07-088. SME Annual Meeting. Feb 25–28. Denver, CO.

Belanger, M. 1993. Application of ORK to a secondary enriched porphyry copper deposit (Chile). 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(241–248). Montreal: CIMM.

Belobgraidich, W., Trumbly, R.G., & R.K. Davey. 1979. Computer-assisted long-range mine planning practices at Ray Mines Division – Kennecott Copper Corporation. *Computer Methods for the 80's.* (A. Weiss, editor): 349–358. New York: SME.

Benito, R., & G. Sanner. 2011. Life of mine schedule from multiple pits: A case study for the Tenke Fungurume Mining District. Paper 11-064. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Benito, R., Sanner, G., & W. Cai. 2011. Life-of-mine schedule from multiple pits: case study for the Tenke Fungurume mining district. Mining Engineering. 63(8): 39–47. August.

Bernabe, D., and K. Dagdelen. 2002. Comparative analysis of open pit mine scheduling techniques for strategic mine planning of the Tintaya Copper mine in Peru. Pre-print Paper 02-125. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.

Bertinshaw, R., and I. Lipton. 2007. Estimating Mining Factors (Dilution and Ore Loss) in Open Pit Mines, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 13–17.

Blackwell, G., Anderson, M., and K. Ronson. 1999. Simulated grades and open pit mine planning – Resolving opposed positions. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 205–216. Colorado School of Mines: CSM.

Blackwell, G.H. 2000. Open pit mine planning with simulated gold grades. CIM Bulletin 93(1039): 31–37.

Blackwell, M.R.L. 1971. Some aspects of the evaluation and planning of the Bougainville copper project. *Decision-Making in the Mineral Industry.* CIM Special Volume No. 12. Montreal: 261–269.

Bofinger, C., and J. Field. 2006. An Occupational Health and Safety Perspective on the Water Systems in Mining, Water in Mining Conference Brisbane, QLD, November 2006, pp. 159–164.

Botin, J., Del Castillo, M., Guzman, R., & M. Smith. 2012. Real options: A tool for managing technical risks in a mine plan. Paper 12-121. SME Annual Meeting. Feb 19–22. Seattle, WA.

Botin, J., Guzman, R., & M. Smith. 2011. A methodological model to assist on the optimization and risk management of mining investment decisions. Paper 11-024. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Cai, W., and A.F. Banfield. 1993. Long range open pit sequencing – a comprehensive approach. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(11–18). Montreal: CIMM.

Cai, W.L. 2001. Design of open pit phases with consideration of schedule constraints. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 217–222. Beijing: Balkema.

Cai, W-L., and A.F. Banfield. 1996. Some practical aspects of open pit mine planning. 26th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 277–284. Penn. State University: SME.

Camus, J.P. 1993. Optimization of the production schedule in the case of several orebodies feeding a stand alone plant. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(19–26). Montreal: CIMM.

Camus, J.P., and S.G. Jarpa. 1996. Long range planning at Chuquicamata mine. 26th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 237–242. Penn. State University: SME.

Carew, F., & C. Forrest. 2011. Integrated mine production – The key to increased production and cost control. Paper 11-143. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Carter, R.A. 2008. Location, location, location: The steadily improving performance of GPS machine location and guidance systems means that production equipment can rul – with greater precision and productivity – but it can't hide. E/MJ. 209(1): 44–47. January/February.

Carter, R.A. 2008. Vendors boot up a new generation of mining software. E/MJ. 209(8): 50–52. October.

Carter, R.A. 2009. Mesh technology knits tighter, more flexible mine communications systems. E/MJ. 210(6): 48–51. July/August.

Cavender, B. 1992. Determination of the optimum lifetime of a mining project using discounted cash flow and option pricing techniques. *Mining Engineering* 44(10): 1262–1268.

Chadwick, J. 2005. Software products – Softly smart. Mining Magazine. February. pp. 31–34.

Chuanqing, C. & Q. Wu. 1989. Chapter 50. An approach to the mining design system based on a deposit model. *Proceedings APCOM 21.* (A. Weiss, editor): 521–528. SME.

Clark, L., & K. Dagdelen. 2007. Finding added value at Newmont through large-scale mining sequence, cutoff grade and process optimization using mixed integer linear programming. Paper 07-004. SME Annual Meeting. Feb 25–28. Denver, CO.

Costa, S.D., and M.J. Scoble. 2006. Mine Accommodation Strategies – Case Studies in Developing Countries, AUSIMM Conference Proceedings, International Mine Management Conference Melbourne, Vic, October 2006, pp. 17–23.

Couzens, T.R. 1979. Aspects of production planning: Operating layout and phase plans. *Open Pit Mine Planning and Design.* (J.T. Crawford and W.A. Hustrulid, editors): 219–231. New York: SME-AIME.

Couzens, T.R. 1990. Personal communication. Cyprus Miami Mining, October 1990.

Crawford, D., Leevers, P., Parsons, S., Solly, G., and S. Nielsen. 2005. Grade Control in the Southern Middleback Ranges – A Case Study, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 271–280.

Crawford, J.T. 1989a. Push back design steps. Personal communication.

Crawford, J.T. 1989b. Logical sequenced approach for evaluating greenfield open pit ore deposits. Personal communication.

Crawford, J.T., III. & R.K. Davey 1979. Case study in open-pit limit analysis. *Computer Methods for the 80's*. (A. Weiss, editor): 310–318. New York: SME-AIME.

Dagdalen, K. 1985. Optimum multi-period open pit mine production scheduling. PhD Thesis T-3073. Colorado School of Mines.

Dagdalen, K. 1992. Cutoff grade optimization. In: *Proceedings 23rd APCOM Proceedings* (Y.C. Kim, editor): 157–165. Littleton, CO: SME.

Dagdelen, K., & K. Kawahata. 2007. Value creation through strategic mine planning and cutoff grade optimization. Paper 07-085. SME Annual Meeting. Feb 25–28. Denver, CO.

Dance, A., Valery, W., Jankovic, A., La Rosa, D., and J. Colacioppo. 2007. Process Integration and Optimisation – A Case Study in Productivity Improvement, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 47–53.

Davey, R.K. 1979a. Short-range mine planning. In: *Computer Methods for the 80's* (A. Weiss, editor): 600–609. New York: SME-AIME.

Davey, R.K. 1979b. Mine production reporting. In: *Computer Methods for the 80's* (A. Weiss, editor): 653–661. New York: AIME.

Davey, R.K. 1979c. Mineral block evaluation criteria. In: *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors): 83–96. SME-AIME.

David, P.A. & A.H. Onur. 1992. Optimizing open pit design and sequencing. In: *Proceedings 23rd APCOM* (Y.C. Kim, editor): 411–422. Littleton, CO: SME.

Davis, R.E. & C.E. Williams. 1973. Optimization procedures for open pit mine scheduling. In: *11th International Symposium on Computer Applications in the Minerals Industry* (J.R. Sturgul, editor) 2(4): C1-C18. University of Arizona.

Denby, B., Schofield, D., and T. Surme. 1998. Genetic algorithms for flexible scheduling of open pit operations. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 605–616. London: IMM.

De Souza e Silva, K., Moura, M. Mourão, Lanna, O., Saliby, E., and J. Fleurisson. 1999. Short term mine planning: Selection of working sites in iron ore mines. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 763–770. Colorado School of Mines: CSM.

Desbarats, A., & M. David. 1984. Influence of selective mining on optimum pit design. *CIM Bulletin* (867)77(7): 49–56.

Deutsch, C.V., E. Magri V., & K. Norrena. 1999. Optimal grade control using geostatistics and economics: Methodology and examples. Paper 99-018. SME Annual Meeting. Mar 1–3. Denver, CO.

Discussion to Halls, Bellum, Lewis paper 1970. (Bulletin. No. 758)79: A26–A30. Reply. 1970. (Bulletin No. 764) 79(7): A107–108.

Dohm, G.C., Jr. 1979. Circular analysis – open pit optimization. In: *Open Pit Mine Planning and Design* (J.T. Crawford and W.A. Hustrulid, editors). New York: Society of Mining Engineers of the AIME.

Dohm, G.C., Jr. 1992. Open pit feasibility studies. In: *SME Mining Engineering Handbook.* 2nd Edition (H.L. Hartman, editor) 2: 1278–1297. Littleton, CO: SME.

Dohmen, A., and M. Dohmen. 2001. Visualization of opencast mine planning. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 319–322. Beijing: Balkema.

Donskoi, E., Holmes, R.J., Manuel, J.R., Campbell, J.J., Poliakov, A., Suthers, S.P., and T. Raynlyn. 2008. Utilisation of iron ore texture information for prediction of downstream process performance. Ninth International Congress for Applied Mineralogy, ICAM 2008 Proceedings, AusIMM Publication Series 8/2008. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 687–694.

Douglas, B., Mercer, S., Wright, S., & D. Barclay. 2009. Pit water management in a mine planning cycle, Olympic Dam case study. Water in Mining Conference. Perth, WA. Sept 15–17. pp. 211–225.

Dowd, P.A. 2011. Formulating and solving complex minerals industry optimisation problems – A systems approach to complex systems problems. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 29–38.

Dowd, P.A., and A.H. Onur. 1992. Optimizing open pit design and sequencing. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 411–422. Tucson: SME.

Dowd, P.A., and C. Xu. 1995. Financial evaluation of mining projects. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 1247–254. Brisbane: AusIMM.

Dowd, P.A., and C. Xu. 1999. The financial evaluation of polymetallic mining projects. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 385–392. Colorado School of Mines: CSM.

Dowson, N., Connelly, D., and D. Yan. 2009. Trends in magnetite ore processing and test work. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 231–242.

Drebenstedt, C., and R. Grafe. 2001. Stereoscopic 3D-display for mine planning and geological modeling. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 323–328. Beijing: Balkema.

Dreger, T., Simpson, C., & D. Lacey. 2012. Overburden mining improvements at Suncor Energy's Millennium mine. Paper 12-017. SME Annual Meeting. Feb 19–22. Seattle, WA.

Dube, T.Y. 2011. Development and application of a holistic quantitative deterministic technique for risk modelling (Quadrim) in mine valuation. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 517–533.

Dubnie, A. 1972. *Surface Mining Practice in Canada*. Mines Branch Information Circular 292(10): 109 pp.

Elbrond, J. & P.A. Dowd 1975. The sequence of decisions on cutoff grades and rates of production. *13th APCOM Symposium*: 13–113. Germany, Verlag Gluckauf.

Elevli, B. 1988. Open pit mine production scheduling. MS Thesis T-3531. Colorado School of Mines.

Elevli, B. 1992. Open pit mine production scheduling using operations research and artificial intelligence. 189 pp. PhD Thesis T-4125. Colorado School of Mines.

Elevli, B., Dagdelen, K., & M.D.G. Salamon 1989. Single time period production scheduling of open pit mines. For Presentation at the SME Annual Meeting, Las Vegas, Nevada, Feb. 27–Mar. 2, 1989. Preprint No. 89–157.

Ellis, T. 2011. A review of the many cost approach methods for minerals evaluation. Paper 11-156. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Ellis, T.R. 2011. Sales comparison valuation of development and operating stage mineral properties. Mining Engineering. 63(4): 89–104. April.

Erdem, O., Guyaguler, T. & N. Demirel. 2012. Uncertainty assessment for the evaluation of net present value: a mining industry perspective. JSAIMM. 112 (5): 405–412.

Erickson, J.D. 1968. Long range open-pit planning. *Mining Engineering* 20(4): 75–77.

Evatt, G.W., Johnson, P.V., Moriarty, J., Duck, P.W., Howell, S.D., & C. Tonkin. 2011. The resource valuation and optimisation model – Real impact from real options. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 535–545.

Everett, J.E., Howard, T.J., and K. Jupp. 2009. Simulation modelling of grade variability for iron ore mining, crushing, stockpiling and ship loading operations. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 351–358.

Ewanchuk, H.G. 1970. Open-pit mining at Bethlehem Copper Corporation, Ltd., Highland Valley, British Columbia, Canada. *Proceedings of the 9th Commonwealth Mining and Metallurgical Congress.* (M.J. Jones, editor): 415–441.

Faust, W.A. 1989. Mine scheduling at the Chimney Creek Mine. Presented at the SME-AIME Annual Meeting, Las Vegas, Feb. 27–Mar. 2, 1989. Preprint No. 89–194.

Fiscor, S. 2007. Miners turn to integrated technology: To offset turnover and share data in a more timely fashion, mining companies are looking at new software tools. E/MJ. 208(9): 49–50. November.

Fitzgerald, D., Chilès, J.P., and A. Guillen. 2009. Delineate three-dimensional iron ore geology and resource models using the potential field method. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 83–94.

Flinn, J.A. 2000. Building an intelligent mining system – Morenci's approach. Paper 00-21. SME Annual Meeting. Feb 28–Mar 1. SLC, UT.

Franklin, P.J. 1985. Computer-aided short-term mine planning. *CIM Bulletin* (879)78: 49–52.

Fraser, S. 1971. Computer-aided scheduling of open-pit mining operations. *Decision-Making in the Mineral Industry* CIM Special Volume No. 12: 379–382.

Fytas, K. & P.N. Calder. 1986. Chapter 11 – A computerized model of open pit short and long range production scheduling. In: *19th APCOM Symposium* (R.V. Ramani, editor): 109–119. SME.

Garcia, J.M. & J.P. Camus. 2011. Value creation in the resource business. JSAIMM. 111(11): 801–808.

Gershon, M. 1986. Ablending-based approach to mine planning and production scheduling. In: *Proceedings 19th APCOM* (R.V. Ramani, editor): 120–126. Littleton, CO: SME.

Gershon, M.E. 1987. An open-pit production scheduler: algorithm and implementation. *Mining Engineering* 39(8): 793–795.

Gholamnejad, J. & E. Moosavi. 2012. A new mathematical programming model for long-term production scheduling considering geological uncertainty. JSAIMM. 112(2): 77–81.

Gholamnejad, J., & M. Osanloo. 2007. Incorporation of ore grade uncertainty into the push back design process. JSAIMM. 107(3): 177–185.

Gholamnejad, J., & M. Osanloo. 2007. Uncertainty based push-back design algorithm in open pit mining. Paper 07-095. SME Annual Meeting. Feb 25–28. Denver, CO.

Gibbs, B. & S.A. Krajewski. 1991. Workshop attendees compare ore modeling and mine planning software systems. *Mining Engineering* 43(7): 732–737.
Gill, D.K. 1966. Open pit mine planning. *Mining Congress Journal* 52(7): 48–51.
Golamnejad, J., Osanloo, M., & B. Karimi. 2006. A chance-constrained programming approach for open pit long-term production scheduling in stochastic environments. JSAIMM. 106(2): 105–114.
Golder Associates 1981. Open pit design program – user guide. Personal communication.
Gomes, O.F.M., and S. Paciomik. 2008. Iron ore quantitative characterisation through reflected light-scanning electron co-site microscopy. Ninth International Congress for Applied Mineralogy, ICAM 2008 Proceedings, AusIMM Publication Series 8/2008. Australasian Institute of Mining and Metallurgy, pp. 699–702.
Goodbody, A. 2011. Mine Scheduling – Mine planning software/scheduling: Tools of the trade. Mining Magazine. September. pp. 67–73.
Goodbody, A. 2012. Mine design – A deeper design. Mining Magazine. March. pp. 25–35.
Grobler, F., Elkington, T., & J-M. Rendu. 2011. Robust Decision Making û Application to Mine Planning Under Price Uncertainty. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 371–380.
Gurtler, D.J. & L.R. Grobl. 1990. Development and implementation of economic material splits at Chino Mines Company. Presentation at the SME Annual Meeting, Salt Lake City, Utah, Feb. 26–Mar. 1, 1990. Preprint No. 90–124.
Gurtler, D.J. 1990. Development and implementation of economic material splits at Chino Mines. *Mining Engineering* 42(4): 343–344.
Hack, D.R. 2002. Comparison of commercial mine planning packages available to the aggregate industry. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 311–320. Alaska: SME.
Halatchev, R. 2002. The time aspect of the optimum long-term open pit production sequencing. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 133–146. Alaska: SME.
Halatchev, R. 2011. Contemporary criteria of open pit long-term production scheduling. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 265–285.
Halatchev, R.A. 1996. The factor of compromise in the assessment of open pit long-term production plans. In Surface Mining 1996 (H.W. Glen, editor): 31–36. SAIMM.
Halatchev, R.A. 2003. Open pit production sequencing: Pseudo-economic discretization versus technological discretization. Pre-print Paper 03-52. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.
Hall, A., & S. Williams. 2009. Benchmarking – Making Best Practice Common Practice. Project Evaluation Conference. Melbourne, Vic. April 21–22. pp. 27–33.
Hall, C.J. 1987. Discussion of 'An open-pit production scheduler: algorithm and implementation'. *Mining Engineering* (12): 1107–1108.
Hall, J., Middlemis, H., Waters, P., and K. Rozlapa, 2006. Integration of Mine Planning with Groundwater Modelling to Optimise Mine Closure Plans at the Marillana Creek (Yandi) Mine, Water in Mining Conference Brisbane, QLD, November 2006, pp. 259–265.
Halls, J.L. 1970. The basic economics of open pit mining. In: *Planning Open Pit Mines* (P.W.J. Van Rensburg, editor). Johannesburg: SAIMM.
Halls, J.L., D.P. Bellum & C.K. Lewis. 1969. Determination of optimum ore reserves and plant size by incremental financial analysis. *Instn Min Metall* (*Min. Sect A*) (Bulletin No. 746) 78(1): A20–A26.
Hansson, K., & D. Tutton. 2011. The pitfalls of benchmarking and how to avoid them. Paper 11-028. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.
Harcus, M. 2011. Mine design – design for life. Mining Magazine. March. pp. 9–18.
Heath, K.C.G. 1980. Relationship between ore reserve, mill feed and real tonnages and grades. *Instn. of Mining & Met.* 89(7): A128–130.
Helvillin, F. 2009. In the future maintaining and proving reserves will be more difficult. Paper 09-034. SME Annual Meeting. Feb 22–25. Denver, CO.
Herfst, U. 1989. Evaluation of stacking alternatives in gold heap leach operations. MS Thesis T-3711. Colorado School of Mines.
Hewlett, R.F. 1961. Application of computers to open pit ore reserve estimation. In: *Short Course on Computers and Computer Applications in the Mineral Industry* (J.C. Dotson, editor) 1: I-1–I-19. Tucson: The University of Arizona.

Hewlett, R.F. 1962. Use of high-speed data reduction and processing in the mineral industry. IC 8099: 82 pp.

Hewlett, R.F. 1963. *A Basic Computer Program for Computing Grade and Tonnage of Ore Using Statistical and Polygonal Methods.* USBM RI 6292: 20.

Hickson, R.J. 1991. Grasberg open-pit: Ertsberg's big brother comes onstream in Indonesia. *Mining Engineering* 43(4): 385–391.

Hodson, D., & C. Rich. 1986. *Open-pit production control at Rössing. The Planning and Operation of Open-Pit and Strip Mines* (J.P. Deetlefs, editor): 225–233. Johannesburg: SAIMM.

Hoerger, S., Bachmann, J., Criss, K., and E. Shortridge. 1999. Long term mine and process scheduling at newmont's Nevada operations. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 739–748. Colorado School of Mines: CSM.

Hoerger, S., Hoffman, L., and F. Seymour. 1999. Mine planning at Newmont's Nevada operations. *Mining Engineering*. 51(10): 26–30.

Hoerger, S., Hoffman, L., and F. Seymour. 1999. Nevada planning at Newmont Gold Company. Pre-print Paper 99-036. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Huang, Z., Cai, W., & A. Banfield. 2009. A new short- and medium-term production planning tool – MINESIGHT schedule optimizer (MSSO). Paper 09-079. SME Annual Meeting. Feb 22–25. Denver, CO.

Hugo, I.E, and R. Bunduwongse. 2007. Mobile Crushing/Sizing Systems for Modern Open Cast Mining – A Case Study of Mae Moh Phase 5 Project, Thailand, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 65–75.

Humphreys, M. 2011. Learning from simulation at Boddington Gold Mine. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 149–155.

Humphries, J. 2011. Maximizing return on capital projects. Paper 11-155. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Iles, C.D., & G.A. Perry. 1983. Sierrita incremental pit design system. *Mining Engineering* 35(2): 152–155.

Innes, C., Nettleton, E., & A. Melkumyan. 2011. A system for ore tracking in autonomous mining. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 719–733.

Jerez, R., and S.V. Tivy. 1992. Developments in linear programming applications at Cyprus Minerals Company. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 519–534. Tucson: SME.

Jewbali, A. 2011. Multivariate block co-simulation of an iron ore deposit. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 157–176.

Ji, C.S., and Y.D. Zhang. 2001. A criteria to optimize mine production rate. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 299–302. Beijing: Balkema.

Johnson, T.B. 1968. Optimum pit mine production scheduling. PhD Thesis. University of California, Berkeley.

Johnson, T.B. 1969. *Optimum open pit mine production scheduling. A Decade of Digital Computing in the Mineral Industry* (A. Weiss, editor): 539–562. New York: Society of Mining Engineers of AIME.

Johnson, T.B., Dagdelen, K., and S. Ramazan. 2002. Open pit mine scheduling based on the fundamental tree algorithm. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 147–162. Alaska: SME.

Joughin, N.C. 2011. Engineering considerations in the tolerability of risk JSAIMM. 111(8): 535–540.

Kahraman, M.M., & S.D. Dessureault. 2011. Development of a real-time adherence to mine plan tools as part of an integrated remote mine control centre. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30.

Kear, R.M. 2006. Strategic and tactical mine planning components. JSAIMM. 106(2): 93–96.

Kelsey, R.D. 1979. Cutoff grade economics. Chapter 28. In: *Proceedings 16th APCOM* (T.J. O'Neil, editor): 286–292. Littleton, CO: SME.

Kim, Y.C. 1979. Production scheduling – technical overview. *Computer Methods for the 80's* (A. Weiss, editor): 610–614. SME of AIME: New York.

King, B. 1999. Schedule optimization of large complex mining operations. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 749–762. Colorado School of Mines: CSM.

King, B.M. 1998. Impact of rehabilitation and closure costs on production rate and cutoff grade strategy. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 617–630. London: IMM.

Kluge, P., & D.F. Malan. 2011. The application of the analytical hierarchical process in complex mining engineering design problems. JSAIMM. 111(12): 847–855.

Knowles, A. 1996. Computer-aided mine design, production and cost management. In Surface Mining 1996 (H.W. Glen, editor): 263–272. SAIMM.

Konak, G., Onur, A.H., & D. Karakus. 2007. Selection of the optimum in-pit crusher location for an aggregate producer. JSAIMM. 107(3): 177–185.

Koski, A.E. 1994. Computerized mine planning in Michigan iron ore. *Mining Engineering*. 46(12): 1338–1342.

Koski, A.E. 1999. Mine design, planning and grade control with specialized software at the Cleveland-Cliffs Iron Company managed Empire mine. Paper 99-053. SME Annual Meeting. Mar 1–3. Denver, CO.

Koski, A.E. 2000. Mine planning software models Empire Mine ore. *Mining Engineering*. 52(9): 47–52.

Krzyzanowska, J. 2007. The impact of mixed fleet hauling on mining operations at Venetia mine. JSAIMM. 107(4): 215–224.

Kvapil, R., McMorran, J., & M. Dotson. 1982. Rapid open pit design system. *World Mining* 35(3): 66–70.

Lane, K.F. 1964. Choosing the optimum cutoff grade. *Colorado School of Mines Quarterly* 59: 811–829.

Lane, K.F. 1979. Chapter 27. Commercial aspects of choosing cutoff grades. In: *Proceedings 16th APCOM* (T.J. O'Neil, editor): 280–285. SME-AIME.

Lane, K.F. 1988. *The Economic Definition of Ore Cutoff Grades in Theory and Practice.* London: Mining Journal Books, Ltd.

Lane, K.F., Hamilton, D.J., & J.J.B. Parker. 1984. Cutoff grades for two minerals. In: *18th APCOM*: 485–492. London: Inst. Min. Met.

Lawrence, M.J. 2000. DCF/NPV modeling: Valuation practice or financial engineering. Paper 00-58. SME Annual Meeting. Feb 28–Mar 1. SLC, UT.

Lawrence, R.D. 2000. Should discounted cash flow projections for the determination of fair market value be based solely on proven and probable reserves? Paper 00-64. SME Annual Meeting. Feb 28–Mar 1. SLC, UT.

Leigh, R.W. & R.L. Blake. 1971. An iterative approach to the optimal design of open pits. *Decision-Making in the Minerals Industry*, CIM Special Volume No. 12: 254–260.

Lerchs, H. & I.F. Grossman. 1965. Optimum design of open-pit mines. *CIM Bulletin*: 47–54.

Lestage, P., Mottola, L., Scherrer, R., and F. Soumis. 1993. A computerized tool for short range production planning in use at the Mont Wright operation. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(67–74). Montreal: CIMM.

Lewis, C.K. 1969. An 'economic life' for property evaluation. *E/MJ* 170(10): 78–79.

Li, Z. 1989. A theoretical approach to determination of mine life and design capacity. *Int. J. of Surface Mining* (1)3: 49–50.

Lien, L. 2011. The next decade of change – Mining's new future management, metrics, methods, machines, maintenance, money. Paper 11-068. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Lill, J.W. 1986. *Operating features and control aspects of the Palabora copper open pit. The Planning and Operation of Open-Pit and Strip Mines* (J.P. Deetlefs, editor): 161–174. Johannesburg: SAIMM.

Lizotte, Y. & J. Elbrond. 1982. Choice of mine-mill capacities and production schedules using open-ended dynamic programming. *CIM Bulletin* (832)75(3): 154–163.

Lizotte, Y. 1988. Economic and technical relations between open-pit design and equipment selection. In: *Mine Planning and Equipment Selection* (R.K. Singhal, editor): 3–13. Rotterdam: A.A. Balkema.

Lovejoy, C. 2009. Surface communication – Clever connections. Mining Magazine. November. pp. 21–25.

Mann, C. & F.L. Wilke. 1992. Open pit short term mine planning for grade control – A combination of CAD-techniques and linear programming. In: *Proceedings 23rd APCOM* (Y.C. Kim, editor): 487–497. Littleton, CO: SME.

Mann, C., and F.L. Wilke. 1992. Open pit short term mine planning for grade control – A combination of CAD techniques and linear programming. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 487–498. Tucson:SME.

Martinez, L.A. 2011. Why accounting for uncertainty and risk can improve final decision-making in strategic open pit mine evaluation. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 113–124.

Mascini, J. 2006. Analysing and Validating Production Performance – A Reconciliation Case Study from George Fisher Mine, Northwest Queensland, AUSIMM Conference Proceedings, 6th International Mining Geology Conference, Darwin, NT, August 2006, pp. 299–306.

Mathieson, G.A. 1982. *Open pit sequencing and scheduling.* Presented at the First International SME-AIME Fall Meeting, Honolulu, Hawaii, Sept. 4–9 1982: Preprint No. 82-368.

McCormack, R., and T. Berryman. 2006. Reconciliation at KCGM Fimiston Operation, AUSIMM Conference Proceedings, 6th International Mining Geology Conference, Darwin, NT, August 2006, pp. 307–312.

McNab, B., Jankovic, A., David, D., and P. Payne. 2009. Processing of magnetite iron ores – comparing grinding options. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 277–288.

Melamud, A., and D.S. Young. 1993. A system for optimizing the interdependence of operating cost and three dimensional pit development. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(75–82). Montreal: CIMM.

Mellish, M., Preller, A.H., & D.A. Tutton. 1987. Integrated open-pit planning approach and systems. *Trans. Instn. Min. Metall* (*Sect A*: *Mine Industry*) 96(10): A162–A170.

Menabode, M., Stone, P., Law, B., & B. Baird. 2007. BLASOR – A generalized strategic mine planning optimization tool. Paper 07-053. SME Annual Meeting. Feb 25–28. Denver, CO.

Milner, T.E. 1977. Long range mine planning at Gibraltar Mines Limited. *CIM Bulletin* (786)70: 89–94.

Mims, C., McClure, S., & C. Rollo. 2012. Mine design – Ready or not? Mining Magazine. March. pp. 40–42.

Montoya, C., Emery, X., Rubio, E., & J. Wiertz. 2012 Multivariate resource modeling for assessing uncertainty in mine design and mine planning. JSAIMM. 112(5): 353–363.

Moore, P. 2010. Mining GPS – Precise control. Mining Magazine. September. pp. 8–14.

Morey, P.G. & K.J. Ashley. 1984. Open pit mine design. 1984 NWMA Short Course. Mine Feasibility – Concept to Completion, Dec. 3–5, 1984, Spokane, WA.

Morley, C., and R. Moller. 2005. Iron Ore Mine Reconciliation – A Case Study From Sishen Iron Ore Mine, South Africa, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 311–318

Morley, C., and K. Thompson. 2006. Extreme Reconciliation – A Case Study from Diavik Diamond Mine, Canada, AUSIMM Conference Proceedings, 6th International Mining Geology Conference, Darwin, NT, August 2006, pp. 313–321.

Murakami, S., Tamanishi, Y., Miwa, S., & J. Yamatomi. 2011. Four-dimensional network relaxation method for long-term open pit mine scheduling. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 311–318.

Mutmansky, J.M. 1979. Computing and operations research techniques for production scheduling. In: *Computer Methods for the 80's* (A. Weiss, editor): 615–625. New York: SME-AIME.

Myburgh, C. 2010. Evolutionary algorithms in large scale open pit mine scheduling. AusIMM Journal. Issue 3. June.

Nel, A., & J. Pienaar. 2006. Establishing a new metric for mineral resource management. JSAIMM. 106(3): 187–198.

Nelson, L.R. & M.R. Goldstein. 1979. Interactive short-range mine planning – A case study. *Computer methods for the 80's* (A. Weiss, editor): 592–599. New York: Society of Mining Engineers.

Newton, M., & A. Baje. 2006. Modeling with limited data. Mining Magazine. March. pp. 24–27.

Nilsson, D. & K. Burgher. 1982. Determining the optimal depth of an open pit. *E&MJ* 183(7): 73–80.

O'Hara, T.A. 1980. Quick guides to the evaluation of orebodies. *CIM Bulletin* (814)73: 87–99.

Ortiz, J.M., Magri, E.J., & R. Libano. 2012. Improving financial returns from mining through geostatistical simulation and the optimized advance drilling grid at El Tesoro Copper Mine. SAIMM. 112(1): 15–22.

Osae, W.K, Fauconnier, C.J., & R.C.W. Webber-Youngman. 2011. A value assessment of mergers and acquisitions in the South African mining industry. JSAIMM. 111(12): 857–869.

Osborne, W.G. MINPAC – computerized aid to pit design and extraction scheduling. Paper presented at the Spring Meeting, Arizona Section, AIME, May 20, 1976.

Pana, M.T. & R.K. Davey. 1973. *Open-pit mine design. SME Mining Engineering Handbook* (I.A. Given, editor). New York: Society of Mining Engineers of the AIME.

Parker, H.M. 2011. Reconciliation principles in the mining industry. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 47–67.

Pax, R., Germain, M., and D. Henderson. 2009. Some new concepts in dry beneficiation possibilities for iron ores. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 379–384.

Peralta, A., and P.N. Calder. 1993. Analysis of the blending problem in open pit production scheduling. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(98–105). Montreal:CIMM.

Perez, A.S., Oreja, E.C., and J.M. Perez. 1993. Short term planning and production schedule of open pit mines. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(106–110). Montreal:CIMM.

Phillis, R.C.D., & H. Gumede. 2011. Annual mine planning and execution buffering – a safety imperative. JSAIMM. 111(8): 565–572.

Pincomb, A. 2003. Aggregate reserves planning & value. Pre-print Paper 03-7. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.

Pitard, F.F. 2008. Blasthole Sampling for Grade Control – The Many Problems and Solutions, AUSIMM Conference Proceedings, Sampling Conference Perth, WA, May 2008, pp. 15–21.

Recny, C.J. 1982. *The influence of geologic characteristics on production capacity and their relation to costs and profitability of mining projects. Mineral Industry Costs* (J.R. Hoskins, compiler): 45–62. Northwest Mining Association.

Rezaee, M., Osanloo, M., & R. Honaker. 2012. Optimizing the number of open pit mine push backs. Paper 12-142. SME Annual Meeting. Feb 19–22. Seattle, WA.

Richmond, A.J. 2001. Maximum profitability with minimum risk and effort. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 45–50. Beijing: Balkema.

Roberts, B., Elkington, P.T., van Olden, K., & M. Maulen. 2009. Optimising a combined open pit and underground strategic plan. Project Evaluation 2009 Conference. Melbourne, Vic. April 21–22. pp. 85–91.

Roman, R.J. 1973. *The use of dynamic programming for determining mine-mill production schedules. 10th APCOM Symposium*: 165–170. South African Inst. Mining Met., Johannesburg.

Roman, R.J. 1974. The role of time value of money in determining an open pit mining sequence and pit limits. In: *12th Int. Symp. on the Applications of Computers and Mathematics in the Minerals Industry* (T.B. Johnson and D.W. Gentry, editors) CSM, vol. I: C-72–C-85.

Rudenno, V. 1979. Chapter 24 – Determination of optimum cutoff grades. In: *Proceedings, 16th APCOM* (T.J. O'Neil, editor): 261–268. SME.

Sabour, S.A., & R. Dimitrakopoulos. 2009. A multi-criteria decision support system for open pit mine planning under geological and market uncertainties. Paper 09-008. SME Annual Meeting. Feb 22–25. Denver, CO.

Sagdan, S. 2009. Gaining an insight into future mining scenarios: Conference report. The First International Future Mining Conference and Exhibition. 2008. AusIMM Journal. Issue 1. Feb.

Sattarvand, J., & C. Niemann-Delius. 2011. A new metaheuristic. algorithm for long-term open pit production planning. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 320–328.

Schellman, M.G. 1989. Determination of an optimum cutoff grade policy considering the stockpile alternative. MS Thesis T-3741. Colorado School of Mines.

Semeniuk, S., & T.R. Ellis. 2012. Discussion and response to the paper of T.R. Ellis "The sales comparison method works best for preliminary-stage mineral projects, but make adjustments carefully". Mining Engineering. 64(2): 51–53. February.

Shahnar, K., & F. Samimi Namin. 2007. A new approach to waste dump site selection according to the fuzzy decision-making process. CIM Magazine. September/October. pp. 240–245.

Sims, D.E. 1979. Open-pit long-range mine planning using O.R.E. In: *Computer Methods for the 80's* (A. Weiss, editor): 359–370. New York: SME.

Smith, M.L. 2001. Integrating conditional simulation and stochastic programming: An application in production scheduling. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 203–208. Beijing:Balkema.

Smith, M.L., Horacek, D., and I.K. Sheppard. 2007. Optimisation of Ok Tedi's Medium-Term Production Schedules Using the Life of Business Optimisation System (LOBOS), AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 145–152.

Soderberg, A. 1959. Elements of long range open pit planning. *Mining Congress Journal* 45(4): 54–57, 58, 62.

Stuart, N.J. 1992. Pit optimization using solid modelling and the Lerchs-Grossmann algorithm. *Int. J. of Surface Mining and Reclamation* 6(1).

Sturgul, J.R. 1985. Optimum life of a mine: Declining production case. *Int. Journal of Mining Engineering* 3: 161–166.

Suriel, J.R. 1984. A procedure for preliminary economic evaluation of open pit mines. ME Thesis ER-2905. Colorado School of Mines.

Taylor, H.K. 1972. General background theory of cutoff grades. *Institution of mining and metallurgy transactions*, Section A 81: A160–170.

Taylor, H.K. 1977. Mine valuation and feasibility studies. *Mineral Industry Costs* (J.R. Hoskins and W.R. Green, editors). 1–17. Spokane, WA: Northwest Mining Association.

Taylor, H.K. 1985. Cutoff grades – some further reflections. *Trans. Instn. Min. Metall.* (Sect. A: Min. Industry) 94(10): A204–216.

Taylor, H.K. 1986. Rates of working of mines – a simple rule of thumb. *Trans. Instn. Min. Metall.* (Sect A: Min. Industry) 95: A203–204.

Taylor, H.K. 1988. Discussion of the paper 'Examination of Empirical formulae for predicting optimum mine output' by E. Arioglu. *Trans. Instn. Min. Metall.* (Sect A: Min. Industry) 97(7): A160.

Taylor, H.K. 1991. Ore-reserves – the mining aspects. *Trans. Instn. Min. Metall.* (Sect A: Min. Industry) 100(9–12): A146–A158.

Taylor, H.K. 1992. Personal communication.

Tenorio, V.O., & S.D. Dessureault. 2011. Supervision of multiple vehicles in autonomous surface mines through a control room. Mining Engineering. 63(11): 80–86. November.

Thomas, G.S. 1996. Pit optimization and mine production scheduling – The way ahead. 26th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 221–228. Penn. State University: SME.

Tolwinski, B. 1998. Scheduling production for open pit mines. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 651–662. London: IMM.

Tolwinski, B. & R. Underwood 1992. An algorithm to estimate the optimal evolution of an open pit mine. In: *Proceedings 23rd APCOM* (Y.C. Kim, editor): 399–409. Littleton, CO: SME.

Tolwinski, B., and R. Underwood 1992. An algorithm to estimate the optimum evolution of an open pit mine. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 399–410. Tucson: SME.

Tolwinski, B., and R. Underwood 1993. Scheduling mine production while maintaining constant mill feeds. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(155–162). Montreal: CIMM.

Topuz, E. 1979. An optimal determination of capital investment, capacity and cutoff grade under multiple objectives. In: *Proceedings Council on Economics*, AIME 108th Annual Meeting: 97–107.

Toutin, E. & J. Camus 1988. Aspectos economicos y de diseno en la planificacion de la mina Sur-Sur. *Minerales* (177)42: 53–60. (Translated by M. Schellman, 1988).

Vieira, J.L., F.D. Durão, F.H. Muge, and J.Q. Rogado. 1995. The integration of short-term planning with mineral processing simulation. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 303–312. Brisbane: AusIMM.

Villafuerte, G., Gumo, S., and R. Rodriguez. 2007. Mitigating Geothermal Hazards at Lihir Gold Mine, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 153–157.

Wang, L.Z., and G.J. Butler. 2007. XPAC Destination Scheduling for Waste Dumps at BHP Billiton, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007, pp. 421–425.

Wang, Q., and H. Sevim. 1992. Enhanced production planning in open pit mining through intelligent dynamic search. In: *Proceedings 23rd APCOM* (Y.C. Kim, editor): 461–471. Littleton, CO: SME.

Wang, Q., and H. Sevim. 1993. An alternative to parameterization in finding a series of maximum-metal pits for production planning. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(168–175). Montreal:CIMM.

Wang, Q., and H. Sun. 2001. A theorem on open pit planning optimization and its application. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 295–298. Beijing:Balkema.

Wells, H.M. 1978. Optimization of mining engineering design in mineral valuation. *SME Transactions* 30(12): 1676–1684. Discussion by R.C. Kirkman, p. 406, April 1979.

Wharton, C.L. 1996. Optimization of cutoff grades for increased profitability. In Surface Mining 1996 (H.W. Glen, editor): 101–104. SAIMM.

Whittle, J. 2011. Long term scheduling. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 75–80.

Whittle, J., and C.L. Wharton. 1995. Optimizing cut-offs over time. 25th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 261–266. Brisbane: AusIMM.

Whittle. J. 1999. A decade of open pit mine planning and optimization – The craft of turning algorithms into packages. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 15–24. Colorado School of Mines: CSM.

Wilke, F.L. & T. Reimer. 1979. Optimizing the short term production schedule for an open pit iron ore mining operation. In: *Computer Methods for the 80's* (A. Weiss, editor): 642–646.

Wilke, F.L., K. Mueller & E.A. Wright. 1984. Ultimate pit and production scheduling optimization. In: *Proceedings of the 18th APCOM Symposium*: 29–38. London: IMM.

Wilkinson, B. 2012. Mine design – Devising the best plan. Mining Magazine. March. pp. 37–39.

Wilkinson, W., Jr. 2012. Top five challenges in mine planning. Paper 12-077. SME Annual Meeting. Feb 19–22. Seattle, WA.

Wilkinson, W.A. 2002. Elements of graphical mine planning. Pre-print Paper 02-46. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.

Williams, C.E. 1974. Computerized year-by-year open-pit mine scheduling. *SME Trans.* 256(12): 309–317.

Winkelmann, S.P. 1990. Newmont Gold outlines its mine planning procedures. *Mining Engineering* 42(4): 333–338.

Winkler, B.M. 1998. Mine production scheduling using linear programming and virtual reality. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 663–672. London: IMM.

Winkler, B.M., and P. Griffin. 1998. Mine production scheduling with linear programming- development of a practical tool. 27th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 673–680. London: IMM.

Wright, E.A. 1986. Dynamic programming in open pit mining sequence planning – a case study. In: *21st APCOM Proceedings*: 415–422.

Wright, E.A. 1987. The use of dynamic programming for open pit mine design: Some practical implications. *Mining Science and Technology* 4(2): 97–104.

Yarmuch, J., & J. Ortiz. 2011. A novel approach to estimate the gap between the middle- and short-term plans. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 419–425.

Yun, Q. & T. Yegalalp. 1982. Optimum scheduling of overburden removal in open pit mines. *Transactions CIM Bulletin* (848)75(12): 80–83.

Yun, Q.X., G.Q. Huang, J.M. Xie, & J.X. Liu. 1988. A CAD optimization package for production scheduling in open-pit mines. In: *Mine Planning and Equipment Selection* (R.K. Singhal, editor): 77–85. Rotterdam: A.A. Balkema.

Zhang, Y., Q. Cai, L. Wu, and D. Zhang. 1992. Combined approach for surface mine short term planning optimization. 23rd International Symposium on Application of Computers and Operations Research in the Mineral Industry: 499–506. Tucson: SME.

Zhao, Y., and Y.C. Kim. 1993. Optimum mine production sequencing using Lagrangian parameterization approach. 24th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 2(176–186). Montreal: CIMM.

Zuckerberg, M., Stone, P., Pasyar, R., & E. Mader. 2007. Joint ore extraction and in-pit dumping optimization. Paper 07-035. SME Annual Meeting. Feb 25–28. Denver, CO.

REVIEW QUESTIONS AND EXERCISES

1. Summarize the basic objectives or goals of extraction planning.
2. What are the two types of production planning?
3. Summarize the five planning 'commandments.'
4. What are the assumptions that need to be made in the mine life – plant size determination?

5. In section 6.2, an example has been presented to illustrate the mine life – plant size determination. Redo the example assuming a copper price of $1.50/lb rather than $1.00.
6. Repeat the example assuming that the copper price is $1.00/lb but the overall recovery is 88%.
7. Taylor has provided some useful insights into deciding the appropriate mine life. Summarize the factors which should be considered.
8. Develop the reasoning as to why one might expect the life of a mine to be related to the cubic root of the ore tonnage.
9. Apply Taylor's rule to the example presented in section 6.2. Do the results agree? Discuss.
10. Discuss the concept behind 'nested' pits. In practice, how are they developed?
11. Apply Taylor's rule to the molybdenum example discussed in section 6.4. Comments?
12. What would be the affect on the operation using the molybdenum prices of today?
13. What are the basic line items in a pre-production cash flow table?
14. In the U.S., how are exploration costs handled?
15. What is an ad valorem tax?
16. What is meant by a pro mille levy?
17. What is meant by the term working capital?
18. What is meant by the terms
    a. Project year
    b. Production year
    c. Calendar year
19. Briefly discuss the basic line items in the production period cash flow calculations.
20. In Figure 6.14, the mining cost is seen to vary with the production rate. Based on these data, what would be the estimated cost if the rate was to be increased to 100,000 tpd? Is this reasonable? Why or why not?
21. Discuss the concept of depletion.
22. Discuss the two types of depletion.
23. Redo the example assuming a 20 year mine life rather than 15 years. What is the effect on the NPV?
24. What is the difference between profit and cash flow?
25. Summarize the steps leading up to the determination of the final pit limits and the sizing of the mine/mill plant.
26. At what point in the process are the capital costs introduced? What effect does this have? How can this problem be minimized?
27. Discuss the significance of Figure 6.17. In practice how would this be generated?
28. In mining period 3, why is there still an overall profit produced?
29. Summarize the objections to the idea that each ore block irrespective of grade should contribute an equal portion to profits and capital payback?
30. Discuss the concept behind 'Incremental Financial Analysis'? What are the steps involved?
31. In section 6.6.2 an example is given for the pit generated with an 0.7% Cu mining cutoff. Two mill cutoffs, 0.7% and 0.6%, were considered and it was decided that the 0.6% cutoff was an improvement. Should the cutoff be further reduced to 0.5%? To 0.4%?
32. Repeat the calculations in the plant sizing example given in section 6.6.3.
33. Summarize the basic concepts behind Lane's algorithm.

34. In section 6.7.4 an illustrative example concerning Lane's algorithm has been presented. Redo the calculation assuming a concentrator cutoff of 0.4% Cu and no constraints are violated.
35. Show the development of equations (6.19), (6.22), and (6.25).
36. Repeat the calculations in section 6.7.5 regarding the cutoff grade for maximum profits.
37. Develop the three curves shown in Figure 6.22.
38. If instead of seeking the cutoff grade that maximizes profits, suppose that one was interested in maximizing the NPV. What approach would be taken?
39. In the example a fixed cutoff grade was assumed throughout the life of the mine. Could the NPV be improved by varying the cutoff grade throughout the life? How could this question be addressed?
40. Repeat the calculations for the example in section 6.7.6 for year 1.
41. What is the difference in the results obtained using the maximum profit and maximum NPV cutoff values?
42. What is the practical meaning of the cutoff grade? How many cutoff grades are there? Briefly discuss each one.
43. What is the difference between a leach dump and a leach pad? What is vat leaching?
44. For the leach dump alternative presented in section 6.8.2, assume that the grade is 0.35% Cu. What is the expected recovery in lbs/ton?
45. Repeat the example in Table 6.35 assuming leaching of material with a grade of 0.45% Cu.
46. Repeat the example in Table 6.36 assuming that material of grade 0.45% Cu is sent to the mill.
47. How should the value of the potential leach material be included when deciding final pit limits?
48. How should the differences in capital cost be taken into account when making leaching versus mill decisions?
49. Discuss the advantages and disadvantages of the stockpile alternative.
50. What were the three stockpile alternatives studied by Schellman?
51. How did the three alternatives compare with respect to the non-stockpiling option?
52. When should the stockpile alternative be considered in open pit mine planning?
53. Repeat the production scheduling example in section 6.9.1 but assume a production rate of 4 blocks per year.
54. Summarize the step-by-step approach to phase scheduling.
55. Phase scheduling is discussed in section 6.9.2. It has been done assuming an annual milling rate of 2,500,000 tons. Repeat the example assuming a milling rate of 2,000,000 tons.
56. What is meant by contingency ore? What are the advantages and disadvantages?
57. Apply the floating cone approach to the block model shown in Figure 6.41. Do you get the same result as shown in Figure 6.48?
58. What is the goal when applying the dynamic programming approach?
59. List the constraints that have been applied in the example in section 6.9.3?
60. Summarize in a simple way the procedure used in the dynamic programming approach as described in section 6.9.3.
61. In the example of section 6.9.3, how has the 'value' of the section been changed by including the fact that all of the blocks cannot be mined simultaneously?

62. In Figure 6.49, how does the choice of discount rate affect the pit limits? The mining sequence?
63. Consider the simple economic block model shown below:

1	2	3	2	1
	2	2	4	
		3		

    Using the approach of Roman, in which order should the blocks be mined to yield the maximum NPV. Assume a discount rate of 10% and the same constraints that were used in his example.
64. Redo problem 63 with constraint 2 removed.
65. Discuss the approach being applied in section 6.9.4. What constraints were applied? How does this approach compare to that introduced in section 6.9.3?
66. What is the first step in the practical planning process?
67. What are some of the terms given to the practical pit planning units?
68. What is the simple objective of the planning process?
69. What is meant by the term 'average profit ratio'? How is it applied?
70. Summarize the steps involved in phase planning as indicated by Mathieson.
71. Summarize the steps in manual pushback design offered by Crawford?
72. In section 6.10.3, an example of a pushback design has been provided. The width of the pushback is 320 ft. Redo the example for a width of 200 ft. Assume all of the other values to be the same.
73. Summarize the concepts dealing with time period plans.
74. With regard to planning, what are the different responsibilities with regard to the engineering and operating groups?
75. In reviewing figures 6.76, 6.77 and 6.78, where did the mining take place over this 2 year period from the end of year 1 to the end of year 3? Provide an estimate of the amount of material removed.
76. At what point in the planning process do you evaluate the equipment fleet requirements? On what basis is it done?
77. What might be included on a production scheduling graph?
78. What are some of the other duties of the mine planner?
79. What is meant by the term 'green field open pit ore deposit'?
80. Summarize the steps in a logical sequenced approach for evaluating a 'green field open pit ore deposit.'
81. In practical terms, what is meant by 'circular analysis'? For an open pit mine evaluation, what are the logical components and sub-components?
82. To a miner, what is meant by the German expression 'Glückauf'?

CHAPTER 7

Reporting of mineral resources and ore reserves

7.1 INTRODUCTION

When describing and classifying mineral resources, it is important that the terms being used are precisely defined and accurately applied. Otherwise, there is the real possibility that potentially very costly misrepresentations and misunderstandings will arise. When considering a property for purchase, or lease, or simply considering the purchase of mining shares, the nature and potential of the assets being represented must be properly and clearly described. Today, the popular expression for this is ‘transparency’. For the unscrupulous promoter of mineral properties (or mining shares), the lack of rules, precise definitions and specified procedures are essential for success. Smoke, mirrors, and sometimes a pinch of salt are the ‘rules’ of the game. Scams involving mineral properties have been around forever and the stories surrounding them provide interesting table discussions at mining meetings. There is, however, enough honest risk in mining without the active participation of scoundrels. The inclusion of this chapter is intended to acquaint the reader with some of the current guidelines regarding the public reporting of exploration results, mineral resources and ore reserves.

During the period of 1969–1970, Australia was home to a series of stock scams involving nickel, primarily, but other metals as well (Sykes (1978), Sykes (1988)). In response, the Australian Mining Industry Council established a committee to examine the issue. The effort was quickly joined by the Australasian Institute of Mining and Metallurgy (AusIMM) and in 1971 the Australasian Joint Ore Reserves Committee (JORC) was formed. Between 1972 and 1989, JORC issued a number of reports which made recommendations on public reporting and ore reserve classification. In 1989 the first version of the JORC Code was released. Since that time, it has gone through a number of revisions. With the gracious permission of the AusIMM (2005), the metal/nonmetal related portion of the most recent document (JORC 2004) has been included as section 7.2 of this chapter.

Over the years, the JORC Code has served as the model for the codes of a number of countries including those of Canada, the United States, South Africa, the United Kingdom/ Europe, Chile and Peru. The authors have found the Canadian ‘Estimation of Mineral Resources and Mineral Reserves – Best Practice Guidelines’ to be quite comprehensive and it is included with the kind permission of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2004) as section 7.3.

It might be argued that even with a rigorous Code in place abuses will occur. A case in point is the recent scam/scandal involving the Canadian company Bre-X Minerals Ltd. This prospect located in the headwaters of the Busang River on the Island of Borneo in Indonesia was obtained by the company in 1993. A drilling program was begun and in February of 1997, the company geologist was talking of 200 million ounces of gold. The stock which was traded on the Alberta, Toronto and Montreal stock exchanges in Canada as well as the NASDAQ National Market in the United States went along for the ride rising from pennies to a high of $250 (Can). In late March 1997 after due-diligence drilling by another company revealed insignificant amounts of gold, the share price plunged to near zero. It is a fascinating story and the interested reader is referred to one of the several books that have been written about it (Francis (1997), Goold and Willis (1997)) or to the numerous articles appearing on the Internet (see for example, www.brexclass.com/graphics/docs/complaint.pdf).

Given the world wide nature of the minerals business, there is considerable interest in the development of a uniform reserve/resource reporting system. The interested reader is referred to the extensive reference list included at the end of the chapter.

7.2 THE JORC CODE – 2004 EDITION

7.2.1 *Preamble*

The authors wish to express their sincere thanks to the Australasian Institute of Mining and Metallurgy (AusIMM) for permission to include the JORC Code – 2004 Edition as part of this book. The original format has been somewhat modified to conform to that used in the rest of the book. The interested reader is encouraged to refer to the original which is available on the JORC website (www.jorc.org).

7.2.2 *Foreword*

The Australasian Code for Reporting of Mineral Resources and Ore Reserves (the 'JORC Code' or 'the Code') sets out minimum standards, recommendations and guidelines for Public Reporting of Exploration Results, Mineral Resources and Ore Reserves in Australasia. It has been drawn up by the Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and the Minerals Council of Australia. The Joint Ore Reserves Committee was established in 1971 and published a number of reports which made recommendations on the classification and Public Reporting of Ore Reserves prior to the first release of the JORC Code in 1989. Revised and updated editions of the Code were issued in 1992, 1996 and 1999. The 2004 edition of the Code (effective December 2004) included in this section supercedes all previous editions.

7.2.3 *Introduction*

The Sections which belong to the Code are printed in ordinary text. The guidelines, placed after the respective Code clauses to provide improved assistance and guidance to readers, are printed in italics. They do not form part of the Code but should be considered persuasive when interpreting the Code. The same formatting has been applied to Table 7.1 – 'Check List of Assessment and Reporting Criteria' to emphasize that it is not a mandatory list of assessment and reporting criteria.

The Code has been adopted by The Australasian Institute of Mining and Metallurgy and the Australian Institute of Geoscientists and is therefore binding on members of those

Table 7.1. Checklist of assessment and reporting criteria (AusIMM, 2005).

CRITERIA	EXPLANATION
	SAMPLING TECHNIQUES AND DATA (criteria in this group apply to all succeeding groups)
Drilling techniques	Drill type (eg. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka etc.) and details (eg. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, etc.). Measures taken to maximise sample recovery and ensure representative nature of the samples.
Logging	Whether core and chip samples have been logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. Whether logging is qualitative or quantitative in nature. Core (or costean, channel etc.) photography.
Drill sample recovery	Whether core and chip sample recoveries have been properly recorded and results assessed. In particular whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.
Other sampling techniques	Nature and quality of sampling (eg. cut channels, random chips etc.) and measures taken to ensure sample representivity.
Sub-sampling techniques and sample preparation	If core, whether cut or sawn and whether quarter, half or all core taken. If non-core, whether riffled, tube sampled, rotary split etc. and whether sampled wet or dry. For all sample types, the nature, quality and appropriateness of the sample preparation technique. Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. Measures taken to ensure that the sampling is representative of the in situ material collected. Whether sample sizes are appropriate to the grainsize of the material being sampled.
Quality of assay data and laboratory tests	The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. Nature of quality control procedures adopted (eg. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie. lack of bias) and precision have been established.
Verification of sampling and assaying	The verification of significant intersections by either independent or alternative company personnel. The use of twinned holes.
Location of data points	Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. Quality and adequacy of topographic control.
Data density and distribution	Data density for reporting of exploration results. Whether the data density and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. Whether sample compositing has been applied.
Audits or reviews	The results of any audits or reviews of sampling techniques and data.
	REPORTING OF EXPLORATION RESULTS (criteria listed in the preceding group apply also to this group)
Mineral tenement and land tenure status	Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. In particular the security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.

(Continued)

Table 7.1. (Continued).

CRITERIA	EXPLANATION
	REPORTING OF EXPLORATION RESULTS (criteria listed in the preceding group apply also to this group)
Exploration done by other parties	Acknowledgement and appraisal of exploration by other parties.
Geology	Deposit type, geological setting and style of mineralisation.
Data aggregation methods	In reporting exploration results, weighting averaging techniques, maximum and/or minimum grade truncations (eg. cutting of high grades) and cut-off grades are usually material and should be stated. Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. The assumptions used for any reporting of metal equivalent values should be clearly stated.
Relationship between mineralisation widths and intercept lengths	These relationships are particularly important in the reporting of exploration results. If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. If it is not known and only the down-hole lengths are reported, there should be a clear statement to this effect (eg. 'downhole length, true width not known').
Diagrams	Where possible, maps and sections (with scales) and tabulations of intercepts should be included for any material discovery being reported if such diagrams significantly clarify the report.
Balanced reporting	Where comprehensive reporting of all exploration results is not practicable, representative reporting of both low and high grades and/or widths should be practised to avoid misleading reporting of exploration results.
Other substantive exploration data	Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.
Further work	The nature and scale of planned further work (eg. tests for lateral extensions or depth extensions or large-scale step-out drilling).
ESTIMATION AND REPORTING OF MINERAL RESOURCES (criteria listed in the first group, and where relevant in the second group, apply also to this group)	
Database integrity	Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. Data validation procedures used.
Geological interpretation	Nature of the data used and of any assumptions made. The effect, if any, of alternative interpretations on Mineral Resource estimation. The use of geology in guiding and controlling Mineral Resource estimation. The factors affecting continuity both of grade and geology.
Estimation and modelling techniques	The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters, maximum distance of extrapolation from data points. The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. The assumptions made regarding recovery of by-products. In the case of block model interpolation, the block size in relation to the average sample

(Continued)

Table 7.1. (Continued).

CRITERIA	EXPLANATION
ESTIMATION AND REPORTING OF MINERAL RESOURCES (criteria listed in the first group, and where relevant in the second group, apply also to this group)	
	spacing and the search employed. Any assumptions behind modelling of selective mining units (eg. non-linear kriging). The process of validation, the checking process used, the comparison of model data to drillhole data, and use of reconciliation data if available.
Cut-off grades or parameters	The basis of the cut-off grade(s) or quality parameters applied, including the basis, if appropriate, of equivalent metal formulae.
Mining factors or assumptions	Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It may not always be possible to make assumptions regarding mining methods and parameters when estimating Mineral Resources. Where no assumptions have been made, this should be reported.
Metallurgical factors or assumptions	The basis for assumptions or predictions regarding metallurgical amenability. It may not always be possible to make assumptions regarding metallurgical treatment processes and parameters when reporting Mineral Resources. Where no assumptions have been made, this should be reported.
Tonnage factors (in situ bulk densities)	Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, the frequency of the measurements, the nature, size and representativeness of the samples.
Classification	The basis for the classification of the Mineral Resources into varying confidence categories. Whether appropriate account has been taken of all relevant factors. ie. relative confidence in tonnage/grade computations, confidence in continuity of geology and metal values, quality, quantity and distribution of the data. Whether the result appropriately reflects the Competent Person(s)' view of the deposit.
Audits or reviews	The results of any audits or reviews of Mineral Resource estimates.
ESTIMATION AND REPORTING OF ORE RESERVES (criteria listed in the first group, and where relevant in other preceding group, apply also to this group)	
Mineral Resource estimate for conversion to Ore Reserves	Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.
Cut-off grades or parameters	The basis of the cut-off grade(s) or quality parameters applied, including the basis, if appropriate, of equivalent metal formulae. The cut-off grade parameter may be economic value per block rather than metal grade.
Mining factors or assumptions	The method and assumptions used to convert the Mineral Resource to an Ore Reserve (ie either by application of appropriate factors by optimisation or by preliminary or detailed design). The choice of, the nature and the appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. The assumptions made regarding geotechnical parameters (eg. pit slopes, stope sizes, etc.), grade control and pre-production drilling. The major assumptions made and Mineral Resource model used for pit optimisation (if appropriate). The mining dilution factors, mining recovery factors, and minimum mining widths used and the infrastructure requirements of the selected mining methods.

(Continued)

Table 7.1. (Continued).

CRITERIA	**EXPLANATION**
	ESTIMATION AND REPORTING OF ORE RESERVES (criteria listed in the first group, and where relevant in other preceding group, apply also to this group)
Metallurgical factors or assumptions	The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. Whether the metallurgical process is well-tested technology or novel in nature. The nature, amount and representativeness of metallurgical testwork undertaken and the metallurgical recovery factors applied. Any assumptions or allowances made for deleterious elements. The existence of any bulk sample or pilot scale testwork and the degree to which such samples are representative of the orebody as a whole.
Cost and revenue factors	The derivation of, or assumptions made, regarding projected capital and operating costs. The assumptions made regarding revenue including head grade, metal or commodity price(s), exchange rates, transportation and treatment charges, penalties, etc. The allowances made for royalties payable, both Government and private.
Market assessment	The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future. A customer and competitor analysis along with the identification of likely market windows for the product. Price and volume forecasts and the basis for these forecasts. For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.
Others	The effect, if any, of natural risk, infrastructure, environmental, legal, marketing, social or governmental factors on the likely viability of a project and/or on the estimation and classification of the Ore Reserves. The status of titles and approvals critical to the viability of the project, such as mining leases, discharge permits, government and statutory approvals.
Classification	The basis for the classification of the Ore Reserves into varying confidence categories. Whether the result appropriately reflects the Competent Person(s)' view of the deposit. The proportion of Probable Ore Reserves which have been derived from Measured Mineral Resources (if any).
Audits or reviews	The results of any audits or reviews of Ore Reserve estimates.

organizations. It is endorsed by the Minerals Council of Australia and the Securities Institute of Australia as a contribution to best practice.

The JORC Code requires the Competent Person(s) on whose work the Public Report of Exploration Results, Mineral Resources or Ore Resources is based, to be named in the report. The report or attached statement must say that the person consents to the inclusion in the report of the matters based on their information in the form and context in which it appears, and must include the name of the person's firm or employer.

7.2.4 *Scope*

The main principles governing the operation and application of the JORC Code are transparency, materiality and competence. 'Transparency' requires that the reader of a Public Report is provided with sufficient information, the presentation of which is clear and unambiguous, to understand the report and is not misled. 'Materiality' requires that a Public Report contains all the relevant information which investors and their professional advisers would reasonably require, and reasonably expect to find in the report, for the purpose of making a reasoned and balanced judgment regarding the Exploration Results, Mineral Resources or Ore Reserves being reported. 'Competence' requires that the Public Report is based on work which is the responsibility of suitably qualified and experienced persons who are subject to an enforceable professional code of ethics.

Reference in the Code to a Public Report or Public Reporting is to a report or reporting on Exploration Results, Mineral Resources or Ore Reserves, prepared for the purpose of informing investors or potential investors and their advisors. This includes a report or reporting to satisfy regulatory requirements.

The Code is a required minimum standard for Public Reporting. JORC also recommends its adoption as a minimum standard for other reporting. Companies are encouraged to provide information in their Public Reports which is as comprehensive as possible.

Public Reports include, but are not limited to: company annual reports, quarterly reports and other reports to the Australian or New Zealand Stock Exchanges or required by law. The Code applies to other publicly released company information in the form of postings on company web sites and briefings for shareholders, stockbrokers and investment analysts. The Code also applies to the following: environmental statements; Information Memoranda; Expert Reports, and technical papers referring to Exploration Results, Mineral Resources or Ore Reserves.

The term 'regulatory requirements' is not intended to cover reports provided to State and Government agencies for statuary purposes, where providing information to the investing public is not the primary intent. If such reports become available to the public, they would not normally be regarded as Public Reports under the JORC Code.

It is recognized that situations may arise where documentation prepared by Competent Persons for internal company purposes or similar non-public purposes does not comply with the JORC Code. In such situations, it is recommended that the documentation includes a prominent statement to this effect.

While every effort has been made within the Code and Guidelines to cover most situations likely to be encountered in the Public Reporting of exploration results, Mineral Resources and Ore Reserves, there may be occasions when doubt exists as to the appropriate form of disclosure. On such occasions, users of the Code and those compiling reports to comply with the Code should be guided by its intent, which is to provide a minimum standard for Public Reporting and to ensure that such reporting contains all information which investors and their professional advisers would reasonably require, and reasonably expect to find in the report, for the purpose of making a reasoned and balanced judgment regarding the Exploration Results, Mineral Resources or Ore Reserves being reported.

The Code is applicable to all solid minerals, including diamonds, other gemstones, industrial minerals and coal, for which Public Reporting of Exploration Results, Mineral Resources and Ore Reserves is required by the Australian and New Zealand Stock Exchanges.

JORC recognizes that further review of the Code and Guidelines will be required from time to time.

7.2.5 *Competence and responsibility*

A Public Report concerning a company's Exploration Results, Mineral Resources or Ore Reserves is the responsibility of the company acting through its Board of Directors. Any such report must be based on, and fairly reflect the information and supporting documentation prepared by a Competent Person or Persons. A company issuing a Public Report shall disclose the name(s) of the Competent Person or Persons, state whether the Competent Person is a full-time employee of the company, and, if not, name the Competent Person's employer. The report shall be issued with the written consent of the Competent Person or persons as to the form and context in which it appears. Documentation detailing Exploration Results, Mineral Resource and Ore Reserve estimates, on which a Public report on Exploration Results, Mineral Resources and Ore Reserves is based, must be prepared by, or under the direction of, and signed by, a Competent Person or Persons. The documentation must provide a fair representation of the Exploration Results, Mineral Resources or Ore reserves being reported.

A 'Competent Person' is a person who is a Member or Fellow of The Australasian Institute of Mining and Metallurgy or of the Australian Institute of Geoscientists, or of a 'Recognized Overseas Professional Organization' ('ROPO') included in a list promulgated from time to time. A 'Competent Person' must have a minimum of five years experience which is relevant to the style of mineralization and type of deposit under consideration and to the activity which that person is undertaking. If the Competent Person is preparing a report on Exploration results, the relevant experience must be in exploration. If the Competent Person is supervising the estimation of Mineral Resources, the relevant experience must be in the estimation, assessment and evaluation of Mineral Resources. If the Competent Person is estimating, or supervising the estimation of Ore Reserves, the relevant experience must be in the estimation, assessment, evaluation and economic extraction of Ore Reserves.

The key qualifier in the definition of a Competent Person is the word 'relevant'. Determination of what constitutes relevant experience can be a difficult area and common sense has to be exercised. For example, in estimating Mineral Resources for vein gold mineralization, experience in a high-nugget, vein-type mineralization such as tin, uranium etc. will probably be relevant whereas experience in (say) massive base metal deposits may not be. As a second example, for a person to qualify as a Competent Person in the estimation of Ore Reserves for alluvial gold deposits, considerable (probably at least five years) experience in the evaluation and economic extraction of this type of mineralization would be needed. This is due to the characteristics of gold in alluvial systems, the particle sizing of the host sediment, and the low grades involved. Experience with placer deposits containing minerals other than gold may not necessarily provide appropriate relevant experience.

The key word 'relevant' also means that it is not always necessary for a person to have five years experience in each and every type of deposit in order to act as a Competent Person if that person has relevant experience in other deposit types. For example, a person with (say) 20 years experience in estimating Mineral Resources for a variety of metalliferous hard-rock deposit types may not require five years specific experience in (say) porphyry copper deposits in order to act as a Competent Person. Relevant experience in the other deposit types could count towards the required experience in relation to porphyry copper deposits.

In addition to experience in the style of mineralization, a Competent Person taking responsibility for the compilation of Exploration Results or Mineral Resource estimates should have sufficient experience in the sampling and analytical techniques relevant to the deposit under consideration to be aware of problems which could affect the reliability of the data. Some appreciation of extraction and processing techniques applicable to that deposit type would also be important.

As a general guide, persons being called upon to act as Competent Persons should be clearly satisfied in their own minds that they could face their peers and demonstrate competence in the commodity, type of deposit and situation under consideration. If doubt exists, the person should either seek opinions from appropriately experienced colleagues or should decline to act as a Competent Person.

Estimation of Mineral Resources may be a team effort (for example, involving one person or team collecting the data and another person or team preparing the estimate). Estimation of Ore Reserves is very commonly a team effort involving several technical disciplines It is recommended that, where there is a clear division of responsibility within a team, each Competent Person and his or her contribution should be identified, and responsibility accepted for that particular contribution. If only one Competent Person signs the Mineral Resource or Ore Reserve documentation, that person is responsible and accountable for the whole of the documentation under the Code. It is important in this situation that the Competent Person accepting overall responsibility for a Mineral Resource or Ore Reserve estimate and supporting documentation prepared in whole or in part by others, is satisfied that the work of the other contributors is acceptable.

Complaints made in respect of the professional work of a Competent Person will be dealt with under the disciplinary procedures of the professional organization to which the Competent Person belongs.

When an Australian listed or New Zealand listed company with overseas interests wishes to report an overseas Exploration Results, Mineral Resource or Ore Reserve estimates prepared by a person who is not a member of The AuslMM, the AIG or a ROPO, it is necessary for the company to nominate a Competent Person or Persons to take responsibility for the Exploration Results, Mineral Resource or Ore Reserve estimate. The Competent Person or Persons undertaking this activity should appreciate that they are accepting full responsibility for the estimate and supporting documentation under Stock Exchange listing rules and should not treat the procedure merely as a 'rubber-stamping' exercise.

7.2.6 *Reporting terminology*

Public Reports dealing with Exploration Results, Mineral Resources and/or Ore Reserves must only use the terms set out in Figure 7.1.

The term 'Modifying Factors' is defined to include mining, metallurgical, economic, marketing, legal, environmental, social and governmental considerations.

Figure 7.1 sets out the framework for classifying tonnage and grade estimates to reflect different levels of geological confidence and different degrees of technical and economic evaluation. Mineral Resources can be estimated mainly by a geologist on the basis of geo-scientific information with some input from other disciplines. Ore Reserves, which are a modified sub-set of the Indicated and Measured Mineral Resources (shown within the dashed outline in Figure 7.1), require consideration of the Modifying Factors affecting extraction, and should in most instances be estimated with input from a range of disciplines.

Measured Mineral Resources may convert to either Proved Ore Reserves or Probable Ore Reserves. The Competent Person may convert Measured Mineral Resources to Probable Ore Reserves because of uncertainties associated with some or all of the Modifying Factors which are taken into account in the conversion from Mineral Resources to Ore Reserves. This relationship is shown by the broken arrow in Figure 7.1. Although the trend of the broken arrow includes a vertical component, it does not, in this instance, imply a reduction

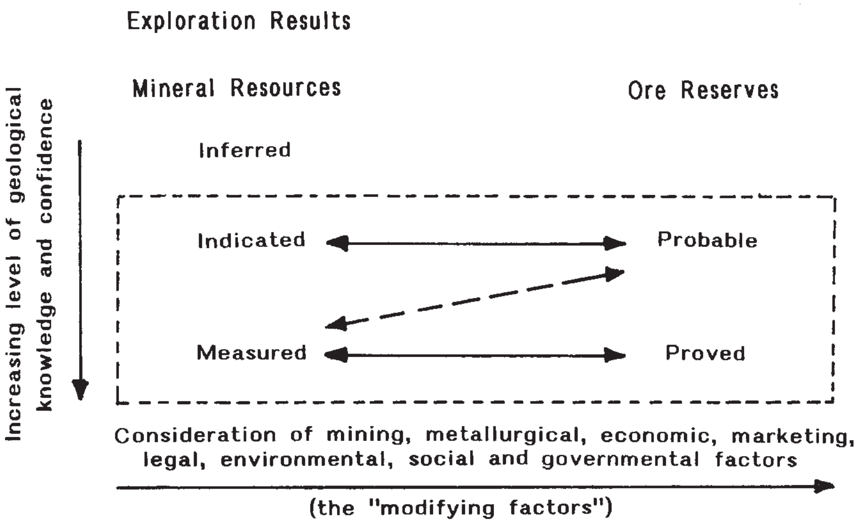


Figure 7.1. General relationship between Exploration Results, Mineral Resources and Ore Reserves. AusIMM. 2005.

in the level of geological knowledge or confidence. In such a situation these Modifying Factors should be fully explained.

7.2.7 *Reporting – General*

Public Reports concerning a company's Exploration Results, Mineral Resources or Ore Reserves should include a description of the style and nature of mineralization.

A company must disclose relevant information concerning a mineral deposit which could materially influence the economic value of that deposit to the company. A company must promptly report any material changes in its Mineral Resources or Ore Reserves.

Companies must review and publicly report on their Mineral Resources and Ore Reserves at least annually.

Throughout the Code, if appropriate, 'quality' may be substituted for 'grade' and 'volume' may be substituted for 'tonnage'.

7.2.8 *Reporting of Exploration Results*

Exploration Results include data and information generated by exploration programs that may be of use to investors. The Exploration Results may or may not be part of a formal declaration of Mineral Resources or Ore reserves.

The reporting of such information is common in the early stages of exploration when the quantity of data available is generally not sufficient to allow any reasonable estimates of Mineral resources.

If a company reports Exploration Results in relation to mineralization not classified as a Mineral Resource or an Ore Reserve, then estimates of tonnage and average grade must not be assigned to the mineralization except under very strict conditions.

Examples of Exploration Results include results of outcrop sampling, assays of drill hole intercepts, geochemical results and geophysical survey results.

Public Reports of Exploration Results must contain sufficient information to allow a considered and balanced judgment of their significance. Reports must include relevant information such as exploration context, type and method of sampling, sampling intervals and methods, relevant sample locations, distribution, dimensions and relative location of all relevant assay data, data aggregation methods, land tenure status plus information on any of the other criteria listed in Table 7.1 that are material to an assessment.

Public Reports of Exploration Results must not be presented so as to unreasonably imply that potentially economic mineralization has been discovered. If true widths of mineralization are not reported, an appropriate qualification must be included in the Public report.

Where assay and analytical results are reported, they must be reported using one of the following methods, selected as the most appropriate by the Competent Person:

- either by listing all results, along with sample intervals (or size, in the case of bulk samples), or
- by reporting weighted average grades of mineralized zones, indicating clearly how grades were calculated.

Reporting of selected information such as isolated assays, isolated drill holes, assays of panned concentrates or supergene enriched soils or surface samples, without placing them in perspective is unacceptable.

Table 7.1 is a checklist and guideline to which those preparing reports on Exploration Results, Mineral Resources and Ore Reserves should refer. The check list is not prescriptive and, as always, relevance and materiality are overriding principles which determine what information should be publicly reported.

It is recognized that it is common practice for a company to comment on and discuss its exploration in terms of target size and type. Any such information relating to exploration targets must be expressed so that it cannot be misrepresented or misconstrued as an estimate of Mineral Resources or Ore reserves. The terms Resource(s) or Reserve(s) must not be used in this context. Any statement referring to potential quantity and grade of the target must be expressed as ranges and must include (1) a detailed explanation of the basis for the statement, and (2) a proximate statement that the potential quantity and grade is conceptual in nature, that there has been insufficient exploration to define a Mineral Resource and that it is uncertain if further exploration will result in the determination of a Mineral Resource.

7.2.9 *Reporting of Mineral Resources*

A 'Mineral Resource' is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.

Portions of a deposit that do not have reasonable prospects for eventual economic extraction must not be included in a Mineral Resource. If the judgment as to 'eventual economic extraction' relies on untested practices or assumptions, this is a material matter which must be disclosed in a public report.

The term 'Mineral Resource' covers mineralization, including dumps and tailings, which has been identified and estimated through exploration and sampling and within which Ore Reserves may be defined by the consideration and application of the Modifying Factors.

The term 'reasonable prospects for eventual economic extraction' implies a judgment (albeit preliminary) by the Competent Person in respect of the technical and economic factors likely to influence the prospect of economic extraction, including the approximate mining parameters. In other words, a Mineral Resource is not an inventory of all mineralization drilled or sampled, regardless of cut-off grade, likely mining dimensions, location or continuity. It is a realistic inventory of mineralization which, under assumed and justifiable technical and economic conditions, might, in whole or in part, become economically extractable.

Where considered appropriate by the Competent Person, Mineral Resource estimates may include material below selected cut-off grade to ensure that the Mineral resources comprise bodies of mineralization of adequate size and continuity to properly consider the most appropriate approach to mining. Documentation of Mineral Resource estimates should clearly identify any diluting material included, and Public reports should include commentary on the matter if considered material.

Any material assumptions made in determining the 'reasonable prospects for eventual economic extraction' should be clearly stated in the Public Report.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron ore, bauxite and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However for the majority of gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Any adjustment made to the data for the purpose of making the Mineral resource estimate, for example by cutting or factoring grades, should be clearly stated and described in the Public Report.

Certain reports (e.g. coal inventory reports, exploration reports to government and other similar reports not intended primarily for providing information for investment purposes) may require full disclosure of all mineralization, including some material that does not have reasonable prospects for eventual economic extraction. Such estimates of mineralization would not qualify as Mineral Resources or Ore Reserves in terms of the JORC.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource.

The inferred category is intended to cover situations where a mineral concentration or occurrence has been identified and limited measurements and sampling completed, but where the data are insufficient to allow the geological and/or grade continuity to be confidently interpreted. Commonly, it would be reasonable to expect that the majority of Inferred Mineral Resources would upgrade to Indicated Mineral Resources with continued exploration. However, due to the uncertainty of Inferred Mineral Resources, should not be assumed that such upgrading will always occur.

Confidence in the estimate of Inferred Mineral resources is usually not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning. For this reason, there is not direct link from an Inferred Resource to any category of Ore Reserves (see Figure 1). Caution should be exercised if this category is considered in technical and economic studies.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource, but has a higher level of confidence than that applying to an Inferred Mineral Resource.

Mineralization may be classified as an Indicated Mineral Resource when the nature, quality, amount and distribution of data are such as to allow confident interpretation of the geological framework and to assume continuity of mineralization. Confidence in the estimate is sufficient to allow the appropriate application of technical and economic parameters and to enable an evaluation of economic viability.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and/or grade continuity.

Mineralization may be classified as a Measured Mineral Resource when the nature, quality, amount and distribution of data are such as to leave no reasonable doubt, in the opinion of the Competent Person determining the Mineral Resource, that the tonnage and grade of the mineralization can be estimated to within close limits and that any variation from the estimate would not significantly affect potential economic viability.

This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Confidence in the estimate is sufficient to allow the application of technical and economic parameters and to enable an evaluation of economic viability that has a greater degree of certainty than an evaluation based on an Indicated Mineral Resource.

The choice of the appropriate category of Mineral Resource depends upon the quantity, distribution and quality of data available and the level of confidence that attaches to those data. The appropriate Mineral Resource category must be determined by a Competent Person or Persons.

Mineral Resource classification is a matter for skilled judgment and Competent Persons should take into account those items in Table 7.1 which relate to confidence in Mineral Resource estimation.

In deciding between Measured Mineral Resources and Indicated Mineral Resources, Competent Persons may find it useful to consider, in addition to the phrases in the two definitions relating to geological and grade continuity, the phrase in the guideline to the *definition for Measured Mineral Resources: '.... any variation from the estimate would be unlikely to significantly affect potential economic viability'.*

In deciding between Indicated Mineral Resources and Inferred Mineral Resources, Competent Persons may wish to take into account, in addition to the phrases in the two definitions relating to geological and grade continuity, the guideline to the definition for Indicated Mineral Resources: 'Confidence in the estimate is sufficient to allow the application of technical and economic parameters and to enable an evaluation of economic viability', which contrasts with the guideline to the definition for Inferred Mineral Resources: 'Confidence in the estimate of Inferred Resources is usually not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning' and 'Caution should be exercised if this category is considered in technical and economic studies'.

The Competent Person should take into consideration issues of the style of mineralization and cutoff grade when assessing geological and grade continuity.

Cutoff grades chosen for the estimation should be realistic in relation to the style of mineralization.

Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. Reporting of tonnage and grade figures should reflect the order of accuracy of the estimate by rounding off to appropriately significant figures and, in the case of Inferred Mineral Resources, by qualification with terms such as 'approximately'.

In most situations, rounding to the second significant figure should be sufficient. For example 10,863,000 tonnes at 8.23 per cent should be stated as 11 million tonnes at 8.2 per cent. There will be occasions, however, where rounding to the first significant figure may be necessary in order to convey properly the uncertainties in estimation. This would usually be the case with Inferred Mineral Resources.

To emphasize the imprecise nature of a Mineral Resource estimate, the final result should always be referred to as an estimate not a calculation.

Competent Persons are encouraged, where appropriate, to discuss the relative accuracy and/or confidence of the Mineral resource estimates. The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnage or volume. Where a statement of the relative accuracy and/or confidence is not possible, a qualitative discussion of the uncertainties should be provided (refer to Table 7.1).

Public Reports of Mineral Resources must specify one or more of the categories of 'Inferred', 'Indicated' and 'Measured'. Categories must not be reported in a combined form unless details for the individual categories are also provided. Mineral Resources must not be reported in terms of contained metal or mineral content unless corresponding tonnages and grades are also presented. Mineral Resources must not be aggregated with Ore Reserves.

Public Reporting of tonnages and grades outside the categories covered by the Code is generally not permitted.

Estimates of tonnage and grade outside of the categories covered by the Code may be useful for a company in its internal calculations and evaluation processes, but their inclusion in Public Reports could cause confusion.

Table 7.1 provides, in a summary form, a list of the main criteria which should be considered when preparing reports on Exploration Results, Mineral Resources and Ore Reserves. These criteria need not be discussed in a Public Report unless they materially affect estimation or classification of the Mineral Resources.

It is not necessary, when publicly reporting, to comment on each item in Table 7.1, but it is essential to discuss any matters which might materially affect the reader's understanding or interpretation of the results or estimates being reported. This is particularly important where inadequate or uncertain data affect the reliability of, or confidence in, a statement of Exploration Results or an estimate of Mineral Resources or Ore Reserves; for example, poor sample recovery, poor repeatability of assay or laboratory results, limited information on bulk densities, etc.

If there is doubt about what should be reported, it is better to err on the side of providing too much information rather than too little.

Uncertainties in any of the criteria listed in Table 7.1 that could lead to under- or over-statement of resources should be disclosed.

Mineral Resource estimates are sometimes reported after adjustment from reconciliation with production data. Such adjustments should be clearly stated in a Public Report of Mineral Resources and the nature of the adjustment or modification described.

The words 'ore' and 'reserves' must not be used in describing Mineral Resource estimates as the terms imply technical feasibility and economic viability and are only appropriate when all relevant Modifying Factors have been considered. Reports and statements should continue to refer to the appropriate category or categories of Mineral Resources until technical feasibility and economic viability have been established. If re-evaluation indicates that the Ore Reserves are no longer viable, the Ore Reserves must be reclassified as Mineral Resources or removed from Mineral Resource/Ore Reserve statements.

It is not intended that re-classification from Ore Reserves to Mineral Resources or vice-versa should be applied as a result of changes expected to be of a short term or temporary nature, or where company management has made a deliberate decision to operate on a non-economic basis. Examples of such situations might be a commodity price fluctuations expected to be of short duration, mine emergency of a non-permanent nature, transport strike etc.

7.2.10 *Reporting of Ore Reserves*

An 'Ore Reserve' is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified. Ore Reserves are sub-divided in order of increasing confidence into Probable Ore Reserves and Proved Ore Reserves.

In reporting Ore Reserves, information on estimated mineral processing recovery factors is very important, and should always be included in Public Reports.

Ore Reserves are those portions of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Competent Person making the estimates, can be the basis of a viable project after taking account of all relevant Modifying Factors.

Ore Reserves are reported as inclusive of marginally economic material and diluting material delivered for treatment or dispatched from the mine without treatment.

The term ‘economically mineable’ implies that extraction of the Ore Reserve has been demonstrated to be viable under reasonable financial assumptions. What constitutes the term ‘realistically assumed’ will vary with the type of deposit, the level of study that has been carried out and the financial criteria of the individual company. For this reason, there can be no fixed definition for the term ‘economically mineable’.

In order to achieve the required level of confidence in the Modifying Factors, appropriate studies will have been carried out prior to the determination of Ore Reserves. The studies will have determined a mine plan that is technically achievable and economically viable and from which the Ore Reserves can be derived. It may not be necessary for these studies to be at the level of a final feasibility study.

The term ‘Ore Reserve’ need not necessarily signify that extraction facilities are in place or operative, or that all necessary approvals have been received. It does signify that there are reasonable expectations of such approvals or contracts. The Competent Person should consider the materiality of any unresolved matter that is dependent on a third party on which extraction is contingent. If there is doubt about what should be reported, it is better to err on the side of providing too much information rather than too little.

Any adjustment made to the data for the purpose of making the Ore Reserve estimate, for example by cutting of factoring grades, should be clearly stated and described in the Public Report.

Where companies prefer to use the term ‘Mineral Reserves’ in their Public Reports, e.g. for reporting industrial minerals or for reporting outside Australasia, they should state clearly that this is being used with the same meaning as ‘Ore Reserves’, defined in this Code. If preferred by the reporting company, ‘Ore Reserve’ and ‘Mineral Resource’ estimates for coal may be reported as ‘Coal Reserve’ and ‘Coal Resource’ estimates.

JORC prefers the term ‘Ore Reserve’ because it assists in maintaining a clear distinction between a ‘Mineral Resource’ and an ‘Ore Reserve’.

A ‘Probable Ore Reserve’ is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified.

A Probable Ore Reserve has a lower level of confidence than a Proved Ore Reserve but is of sufficient quality to serve as the basis for a decision on the development of the deposit.

A ‘Proved Ore Reserve’ is the economically mineable part of a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified.

A Proved Ore Reserve represents the highest confidence category of reserve estimate. The style of mineralization or other factors could mean that Proved Ore Reserves are not achievable in some deposits.

The choice of the appropriate category of Ore Reserve is determined primarily by the relevant level of confidence in the Mineral Resource and after considering any uncertainties in the Modifying Factors. Allocation of the appropriate category must be made by a Competent Person or Persons.

The Code provides for a direct two-way relationship between Indicated Mineral Resources and Probable Ore Reserves and between Measured Mineral Resources and Proved Ore Reserves. In other words, the level of geological confidence for Probable Ore Reserves is similar to that required for the determination of Indicated Mineral Resources, and the level of geological confidence for Proved Ore Reserves is similar to that required for the determination of Measured Mineral Resources.

The Code also provides for a two-way relationship between Measured Mineral Resources and Probable Ore Reserves. This is to cover a situation where uncertainties associated with any of the Modifying Factors considered when converting Mineral Resources to Ore Reserves may result in there being a lower degree of confidence in the Ore Reserves than in the corresponding Mineral Resources. Such a conversion would not imply a reduction in the level of geological knowledge or confidence.

A Probable Ore Reserve derived from a Measured Mineral Resource may be converted to a Proved Ore Reserve if the uncertainties in the Modifying Factors are removed. No amount of confidence in the Modifying Factors for conversion of a Mineral Resource to an Ore Reserve can override the upper level of confidence that exists in the Mineral Resource. Under no circumstances can an Indicated Mineral Resource be converted directly to a Proved Ore Reserve (see Figure 7.1).

Application of the category of Proved Ore Reserve implies the highest degree of confidence in the estimate, with consequent expectations in the minds of readers of the report. These expectations should be borne in mind when categorizing a Mineral Resource as Measured.

Ore Reserve estimates are not precise calculations. Reporting of tonnage and grade figures should reflect the relative uncertainty of the estimate by rounding off to appropriately significant figures.

To emphasize the imprecise nature of an Ore Reserve, the final result should always be referred to as an estimate not a calculation.

Competent Persons are encouraged, where appropriate, to discuss the relative accuracy and/or confidence of the Ore Reserve estimates. The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnage or volume. Where a statement of the relative accuracy and/or confidence is not possible, a qualitative discussion of the uncertainties should be provided (refer to Table 7.1).

Public Reports of Ore Reserves must specify one or other or both of the categories of ‘Proved’ and ‘Probable’. Reports must not contain combined Proved and Probable Ore Reserve figures unless the relevant figures for each of the individual categories are also provided. Reports must not present metal or mineral content figures unless corresponding tonnage and grade figures are also given.

Public Reporting of tonnage and grade outside the categories covered by the Code is generally not permitted.

Estimates of tonnage and grade outside of the categories covered by the Code may be useful for a company in its internal calculations and evaluation processes, but their inclusion in Public Reports could cause confusion.

Ore Reserves may incorporate material (dilution) which is not part of the original Mineral Resource. It is essential that this fundamental difference between Mineral Resources and Ore Reserves is borne in mind and caution exercised if attempting to draw conclusions from a comparison of the two.

When revised Ore Reserve and Mineral Resource statements are publicly reported they should be accompanied by reconciliation with previous statements. A detailed account of differences between the figures is not essential, but sufficient comment should be made to enable significant changes to be understood by the reader.

In situations where figures for both Mineral Resources and Ore Reserves are reported, a statement must be included in the report which clearly indicates whether the Mineral Resources are inclusive of, or additional to the Ore Reserves.

Ore Reserve estimates must not be aggregated with Mineral Resource estimates to report a single figure.

In some situations there are reasons for reporting Mineral Resources inclusive of Ore Reserves and in other situations for reporting Mineral Resources additional to Ore Reserves. It must be made clear which form of reporting has been adopted. Appropriate forms of clarifying statements may be:

'The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Ore Reserves.' or 'The Measured and Indicated Mineral Resources are additional to the Ore Reserves'.

In the former case, if any Measured and Indicated Mineral Resources have not been modified to produce Ore Reserves for economic or other reasons, the relevant details of these unmodified Mineral Resources should be included in the report. This is to assist the reader of the report in making a judgment of the likelihood of the unmodified Measured and Indicated Mineral Resources eventually being converted to Ore Reserves.

Inferred Mineral Resources are by definition always additional to Ore Reserves.

The reported Ore Reserve figures must not be aggregated with the reported Mineral Resource figures. The resulting total is misleading and is capable of being misunderstood or of being misused to give a false impression of a company's prospects.

Table 7.1 provides, in a summary form, a list of the main criteria which should be considered when preparing reports on Exploration Results, Mineral Resources and Ore Reserves. These criteria need not be discussed in a Public Report unless they materially affect estimation or classification of the Ore Reserves. Changes in economic or political factors alone may be the basis for significant changes in Ore Reserves and should be reported accordingly.

Ore Reserve estimates are sometimes reported after adjustment from reconciliation with production data. Such adjustments should be clearly stated in a Public Report of Ore Reserves and the nature of the adjustment or modification described.

7.2.11 *Reporting of mineralized stope fill, stockpiles, remnants, pillars, low grade mineralization and tailings*

The Code applies to the reporting of all potentially economic mineralized material. This can include mineralized fill, remnants, pillars, low grade mineralization, stockpiles, dumps and tailings (remnant materials) where there are reasonable prospects for eventual economic extraction in the case of Mineral Resources, and where extraction is reasonably justifiable in the case of Ore Reserves. Unless otherwise stated, all other parts of the Code (including Figure 7.1) apply.

Any mineralized material as described here can be considered to be similar to in situ mineralization for the purposes of reporting Mineral Resources and Ore Reserves. Judgments about the mineability of such mineralized material should be made by professionals with relevant experience.

If there are no reasonable prospects for the eventual economic extraction of all or part of the mineralized material as described here, then this material cannot be classified as either Mineral Resources or Ore Reserves. If some portion of the mineralized material is sub-economic, but there is a reasonable expectation that it will become economic, then this material may be classified as a Mineral Resource. If technical and economic studies have demonstrated that economic extraction could reasonably be justified under realistically assumed conditions, then the material may be classified as an Ore Reserve.

The above guidelines apply equally to low grade in situ mineralization, sometimes referred to as 'mineralized waste' or 'marginal grade material', and often intended for stockpiling and treatment towards the end of mine life. For clarity of understanding, it is recommended that tonnage and grade estimates of such material be itemized separately in Public Reports, although they may be aggregated with total Mineral Resource and Ore Reserve figures.

Stockpiles are defined to include both surface and underground stockpiles, including broken ore in stopes, and can include ore currently in the ore storage system. Mineralized material in the course of being processed (including leaching), if reported, should be reported separately.

7.3 THE CIM BEST PRACTICE GUIDELINES FOR THE ESTIMATION OF MINERAL RESOURCES AND MINERAL RESERVES – GENERAL GUIDELINES

7.3.1 *Preamble*

The authors wish to express their sincere thanks to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) for permission to include the 'Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines' as part of this book. These Guidelines were adopted by the CIM Council on November 23, 2003. The format has been somewhat modified to conform to that used in the rest of the book. The interested reader is encouraged to refer to the original which is available on the CIM website (www.cim.org/committees/estimation.cfm).

7.3.2 *Foreword*

These guidelines have been prepared by the Canadian Institute of Mining and Metallurgy and Petroleum (CIM) led 'Estimation Best Practices Committee'. They are intended to assist the Qualified Person(s) (QP) in the planning, supervision, preparation and reporting of Mineral Resource and Mineral Reserve (MRMR) estimates. All MRMR estimation work from which public reporting will ensue must be designed and carried out under the direction of a QP. A QP is defined as 'an individual who is an engineer or geoscientist with at least five (5) years of experience in mineral exploration, mine development, mine operation, project assessment or any combination of these; has experience relevant to the subject matter of the mineral project and technical report; and is a member in good standing of a professional association'.

The 'General Guidelines' section of the document (that included in this section) deals primarily with the description of best practice as it applies to metalliferous deposits.

In planning, implementing and directing any estimation work, the QP should ensure that practices followed are based on methodology that is generally accepted in the industry and that the provisions of the Exploration Best Practices Guidelines have been adhered to during the exploration phase that led to the delineation of the resource.

In addition to assisting the QP in the preparation of MRMR estimates, these 'Best Practice Guidelines' are intended to ensure a consistently high quality of work and foster greater standardization of reporting in publicly disclosed documents.

Qualified Person

The Qualified Person will base the MRMR estimation work on geological premises, interpretation and other technical information as the QP deems appropriate. In addition, the QP will select an estimation method, parameters and criteria appropriate for the deposit under consideration. In planning, implementing and supervising any estimation work, the QP will ensure that the methods employed and the practices followed can be justified on technical merit and/or are generally accepted in the industry.

Because a MRMR model is based fundamentally on accurate geological interpretation and economic understanding, the persons responsible for the Mineral Resource and subsequent Mineral Reserve estimation should have a firm understanding of geology, mining, and other issues affecting the estimate. This level of understanding would normally be developed through acquiring appropriate geological, mining and Mineral Reserve preparation experience in a relevant operating mine.

While the reporting QP ultimately will have responsibility for the resulting estimate, he or she should have access to other QP, in the compilation of the estimate, who have suitable training or experience in disciplines that may fall outside the expertise of the reporting QP. This will allow appropriate consideration of all factors affecting the estimate including, for example, geology and geological interpretation, metallurgy, mining and social, legal and environmental matters.

Definitions

- Mineralization: 'material of potential interest. Mineral Resources and Mineral Reserves are economic subsets of such mineralization'.

- Quality Assurance/Quality Control (QA/QC): for the purpose of this document;
Quality Assurance means:
'All of those planned or systematic actions necessary to provide adequate confidence in the data collection and estimation process', and
Quality Control means:
'The systems and mechanisms put in place to provide the Quality Assurance. The four steps of quality control include; setting standards; appraising conformance; acting when necessary and planning for improvements'.

- Mineral Resource: a concentration or occurrence of natural, solid, inorganic, or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics, and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge'.

- Mineral Reserve: 'the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined'.

• Estimate: (verb) 'to judge or approximate the value, worth, or significance of; to determine the size, extent, or nature of'. (noun) 'an approximate calculation; a numerical value obtained from a statistical sample and assigned to a population parameter'.

• Preliminary Feasibility Study: 'a comprehensive study of the viability of a mineral project that has advanced to the stage where the mining method, in the case of underground mining, or the open pit configuration, in the case of an open pit, has been established and which, if an effective method of mineral processing has been determined includes a financial analysis based on reasonable assumptions of technical, engineering, operating, and economic factors and evaluation of other relevant factors which are sufficient for a QP, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve'.

• Deposit: 'a natural occurrence of mineral or mineral aggregate, in such quantity and quality to invite exploitation'.

• Classification and Categorization: 'a mineral deposit may be subdivided into two Classes, Mineral Resources and Mineral Reserves. Each of these Classes may be subdivided into Categories: Measured, Indicated and Inferred in the case of Mineral Resources and Proven and Probable in the case of Mineral Reserves'.

7.3.3 *The Resource Database*

The Resource Database is established by the collection, verification, recording, storing and processing of the data and forms the foundation necessary for the estimation of MRMR. The establishment of a QA/QC program of all data is essential during this process.

Components of the Resource Database typically will include geological data (e.g. lithology, mineralization, alteration, and structure), survey data, geophysical data, geochemical data, assay data, rock quality and bulk density information and activity dates.

As stated in the CIM Standards and as noted above, a Mineral Resource must have reasonable prospects of economic extraction. Consequently, preliminary data and information concerning a number of factors (e.g. mining, metallurgy, economics and social and environmental sensitivity) will be collected and assessed during the estimation of a Mineral Resource.

General comments

• A database consists of two types of data, primary data and interpreted data. Primary data are parameters amenable to direct physical measurement. Examples include assays, survey data, and geological observations. Interpreted data sets are derivations or interpretations of primary information. Examples are geological projections and block models.

• Bulk density is an important parameter that should be measured and recorded at appropriate intervals, and in an appropriate manner, for the deposit. The choice of methods for determining the bulk density of a particular deposit will depend on the physical characteristics of the mineralization and the available sampling medium.

• The QP should be diligent in ensuring that the final database fairly represents the primary information. Data verification is an essential part of finalizing the resource database.

• The Resource Database provides a permanent record of all the data collected from the work carried out, the date of the work, observations and comments from the results obtained. It should be readily available for future reference. The database provides all of the information necessary to enable current and future geological interpretations and modeling.

• Although most databases are generally maintained in an electronically-stored digital format, hand-printed tables with well-organized information may also form a database. It is recommended that data be stored digitally, using a documented, standard format and a reliable medium that allows for easy and complete future retrieval of the data.

Primary data visualization

• It is essential that the systematic recording of geological observations from mapping and drill hole logging be entered into an organized database.

• Data collection and display must foster a good geological understanding of a deposit as a prerequisite for the Mineral Resource estimation process (see Section 7.3.4).

• The important primary data must be identified and accurately presented in three dimensions, typically on a set of plans and sections. Examples are lithologies, structural measurements, assays, etc.

• Where local mine coordinates are used on geological maps and sections, a mechanism for conversion to universal coordinates must be provided. Maps and sections must include appropriate coordinates, elevation, scale, date, author(s) and appropriate directional information.

• Data positioning information should be relative to a common property co-ordinate system and should include the methodology and accuracy used to obtain that information. Accurate location of data points is essential. If data points are referred to a particular map or grid, those reference data should be included, the map properly identified and the coordinate system clearly stated.

• If primary data have been intentionally omitted from the presentation, they should be identified with an explanatory note for their exclusion.

Interpreted data visualization

• The geological interpretation including mineralization and its controls (e.g. structure, alteration, and lithology) is essential for MRMR estimation. The primary data (i.e. from outcrops, trenches and drill holes) should be clearly identifiable and be distinct from the interpreted data so that it may be utilized in subsequent interpretations and Mineral Resource estimates.

• The relevant geophysical/geochemical/topographic data used to support the interpretation of faults or boundaries must be included or referenced appropriately.

• Since the mineralizing episode(s) and related features of the geology are critical aspects in the MRMR estimations, they must be clearly represented. Examples are controlling features, style(s) and age(s) of mineralization, boundaries of the mineralization, and zonation of the mineralization.

Data collection, recording, storing and processing

• Primary data collected must be recorded even if not used in the MRMR estimation.

• Original assay data should be stored in the units of measure as received by the laboratory (e.g. large ppm values should not be reported as percentages). The analytical method used must be described.

• Analytical data should be converted into common units of measure provided the analytical technique supports the conversion. The original and converted assay should be reported, including the conversion factor(s).

• Data that have been acquired over multiple periods and by various workers should be verified and checked prior to entry into the database. In addition, data records should possess unique identifiers (e.g. unique drill hole, zone and sample numbers, etc.). A distinction must be made between data collected by different methodologies (e.g. reverse circulation holes versus diamond drill holes, etc.) and an explanation of how these data sets are integrated, should be provided.

• Upon the reporting of MRMR estimates, all the tabulations and defining parameters become part of the database. Summations, tabulations, maps, assumptions and related parameters, for example cut-off grade(s), commodity price(s), dilution, losses, plant recovery(ies) become interpreted data and must be enumerated.

• Mine production data are primary and must be incorporated into the MRMR database. Best practice includes routine reporting of reconciliations and monitoring systems implemented during the operational phases of the project. These results will be used for revisions in the MRMR estimation.

• Periodic review of data to ensure its integrity is recommended.

• Duplicate, secure off site storage of data is recommended.

QA/QC

• QA/QC must be addressed during the collection, recording and storage of any of the data ultimately used in the MRMR estimation. This program should be concerned with, but not limited to: data verification, drill sample recovery, sample size, sample preparation, analytical methods, the use of duplicates/blanks/standards, effects of multiple periods of data acquisition and consistency of interpretation in three dimensions. The sample preparation description should include aliquot weight used in the laboratory. The results of the QA/QC program form part of the database and must be recorded.

7.3.4 *Geological interpretation and modeling*

The purpose of this section is to give guidance to the QP responsible for estimating MRMR. These Guidelines outline requirements for interpretation of geological data, the consideration of economic and mining criteria and the linkage of that information to the grade distribution of the MRMR model as described in Section 7.3.5.

Geological data

• Comprehensive geology and reliable sample information remain the foundation of MRMR estimates.

• Information used for MRMR estimation should include surface geology at suitable scales (lithologies, mineralogical zones, structural regimes, alteration, etc.), topographical data, density information, a complete set of all available sample results and surveyed locations of all sample sites (chips, drill samples, etc.).

• All geological information within the deposit should be transposed from plan onto sections (or vice versa) to confirm reliability and continuity using all available data (drill holes, mine workings, etc.). Two directions of vertical sections (usually orthogonal) and plans should be used to ensure manual interpretations are internally consistent.

• Geological interpretation is frequently completed in a three dimensional (3-D) computer environment. Computer assisted interpretations should be validated on plan and orthogonal section to evaluate the reliability of the geological interpretation.

• Understanding the relationship between the mineralization and the geological processes that govern its geometry is essential. Mineralized limits (whether sharp or gradational) within which the MRMR are to be determined must be interpreted and depicted on maps, plans and sections.

Geological interpretations

• MRMR modeling should be developed within a regional context. Accordingly, the regional geology and property geology are important parts of the geological database.

• The interpretation of geological field data (lithology, structure, alteration and mineralized zones, etc.) should include direct input from individuals with mapping or core logging experience on the deposit.

• Field data should be presented in their entirety, in an unmodified form. Every effort must be made to analyze these data in an unbiased, scientific fashion to develop a 'Geological Concept' which forms the underlying premise on which the geologic interpretation is developed. The concept should include, among others, geological setting, deposit type, styles of mineralization, mineralogical characteristics and genesis.

• The styles of the mineralization under investigation must be identified to allow the modeler to establish geological controls for mineralization and permit more accurate interpolation of grades within the model.

• The geological interpretation and ideas regarding genesis of the deposit should be reviewed in the context of the resultant MRMR model. Aspects and assumptions, for which field data are incomplete, should be clearly identified.

Controls of mineralization

• Once the geological framework of the deposit has been reasonably established, geological controls for mineralization and the limits of those controls are determined. Attention to detail is vital for early recognition of important features that control the spatial distribution, variability and continuity of economic mineralization.

• Mineralization may be defined or limited by some combination of features such as structure, lithology and the alteration envelope. These limits or boundaries should be used to constrain the interpolation of grade or quality within the MRMR model.

• When determining limits of mineralization, the estimator must recognize that many mineral deposits comprise more than one type of mineralization. The characteristics of each type will likely require different modeling techniques and/or parameters.

Mining and economic requirements

• By definition, a Mineral Resource must have 'reasonable prospects of economic extraction'.

• Factors significant to project economics must be considered for both Mineral Resource and Mineral Reserve estimates. These will include the extraction characteristics for both the mining and processing method selected as affected by geotechnical, grade control, and metallurgical, environmental and economic attributes.

• For a Mineral Resource, factors significant to project economics should be current, reasonably developed and based on generally accepted industry practice and experience. Assumptions should be clearly defined. For Mineral Reserves, parameters must be detailed with engineering complete to Preliminary Feasibility standards as defined in the CIM Standards.

• Mining assumptions for a Mineral Reserve include: continuity of mineralization, methods of extraction, geotechnical considerations, selectivity, minimum mining width, dilution and percent mine extraction.

• Cut-off grade or cut-off net smelter return (NSR) used for MRMR reporting are largely determined by reasonable long term metal price(s), mill recovery and capital and operating costs relating to mining, processing, administration and smelter terms, among others. All assumptions and sensitivities must be clearly identified.

• Cut-off grade must be relevant to the grade distribution. The mineralization must exhibit sufficient continuity for economic extraction under the cut-off applied.

Three dimensional computer modeling

• MRMR models can be generated with or without the use of 3-D computer software. However, it is likely that any MRMR estimate that is included in a feasibility study will be in the form of a 3-D computer model. This section refers only to those MRMR models generated using such techniques.

• The modeling technique(s) adopted for a project should be appropriate for the size, distribution and geometry of the mineralized zones. The technique should also be compatible with the anticipated mining method(s) and size and type of equipment.

• The QP must analyze the grade distribution to determine if grade compositing is required. Where necessary, assay data should be composited to normalize the grade distribution and to adequately reflect the block size and production units.

• The size of the blocks in the model will be chosen to best match mining selectivity, drill hole and sample density, sample statistics and anticipated grade control method. A change in cut-off grade or economic limit and selectivity of the mining method(s) frequently requires the development of new models and perhaps increased drill definition to properly evaluate the mineral deposit in question.

• An aspect of block modeling is the loss of critical geological and assay information through smoothing of details inherent in the modeling technique. General validation of the block model against raw data is required to ensure reliability.

Selection of software

• This section refers only to those MRMR models generated using software. It is recognized that MRMR can be estimated using other methods without the use of computers and software.

• The software and the version used should be clearly stated.

• There is a number of adequate, commercially available data handling and modeling software packages currently in use. The person responsible for the development of the MRMR model should have appropriate knowledge of the software, methodology, limitations and underlying assumptions utilized during the modeling process.

7.3.5 *Mineral Resource estimation*

This section considers important factors in estimating a Mineral Resource and documenting the estimation process. It provides guidelines to the QP responsible for the Mineral Resource estimate with respect to data analysis, sample support, model setup and interpolation. Critical elements to the Mineral Resource estimate are the consideration of the appropriate geological interpretation and the application of reasonably developed economic parameters, based on generally accepted industry practice and experience. While innovation is encouraged, comparisons with other tested methods are essential, prior to publicizing or reporting estimates. Optimization of the Mineral Resource interpretation in consideration of economic parameters is an iterative process.

Data density

• A key initial step prior to the commencement of estimating a Mineral Resource is the assessment of data adequacy and representativeness of the mineralization to be modeled. If the number and spatial distribution of data are inadequate, an estimation is required of how much additional data are needed before a Mineral Resource calculation can meaningfully be done. The QP responsible for modeling must ensure that the available information and sample density allow a reliable estimate to be made of the size, tonnage and grade of the mineralization in accordance with the level of confidence established by the Mineral Resource categories in the CIM Standards.

Integration of geological information

• The deposit geology forms the fundamental basis of the Mineral Resource estimation. The data must be integrated into, and reconciled with, the geological interpretation as part of the estimation process. The interpretation should include the consideration and use of reasonable assumptions on the limits and geometry of the mineralization, mineralization controls and internal unmineralized or 'waste' areas (i.e. dikes or sills). Interpretive information should be continuously reassessed as knowledge of the geological characteristics of a deposit improves.

Listing/recording the data set

• All data and information used in the Mineral Resource estimation must be identified, catalogued and stored for future reference and audits. Any portion of the pertinent data acquired during the exploration and development of the property that is not used in the Mineral Resource estimation must be identified and an explanation provided for its exclusion.

• Sampling, sample preparation, assaying practice and methodologies must be clearly described and an explanation given for the choice of the particular methods used. A comment as to their effectiveness should also be provided.

• Particular care should be taken in recording, analyzing and storing data and results from QA/QC programs related to the Mineral Resource estimation.

Data analysis

• The principal purpose of data analysis is to improve the quality of estimation through a comprehensive understanding of the statistical and spatial character of variables on which the estimate depends. This would include establishment of any interrelationships among the variables of interest, recognition of any systematic spatial variation of the variables (e.g. grade, thickness), definition of distinctive domains that must be evaluated independently for the estimate, and identification and understanding of outliers. In particular, it will be necessary to understand the extent to which 'nugget effect' affects the mineralized sample population. This is often a major concern for precious metal deposits and may be important in other types of deposits.

• Data analysis should be comprehensive and be conducted using appropriate univariate, bivariate, and/or multivariate procedures. Univariate procedures include statistical summaries (mean, standard deviation, etc.), histograms and probability plots. Bivariate procedures include correlation studies, evaluation of scatter plots and regression analysis whereas multivariate analysis might involve procedures such as multiple regression (e.g. bulk density – metal relationships) and multiple variable plots (e.g. triangular diagrams).

• Variography is an aspect of data analysis that assists in defining the correlation and range of influence of a grade variable in various directions in three dimensions.

• Outlier recognition and treatment of outliers is an important part of the data analysis. An outlier is an observation that appears to be inconsistent with the majority of the data and attention for the purposes of Mineral Resource estimation usually is directed to those that are high relative to most data. The modeler must state how an outlier is defined and how it is treated during the resource estimation process (i.e. grade cutting strategy, restricted search philosophy).

Sample support

• Sample or data support (size, shape and orientation of samples) must be considered. Data for the Mineral Resource estimate generally are obtained from a variety of supports and statistical parameters can vary substantially from one support to another. If composites are used as a basis for estimation, the data must be combined in a manner to produce composites of approximately uniform support prior to grade estimation.

• Selection of a composite length should be appropriate for the data and deposit (e.g. bench or half bench height, dominant assay interval length, vein thickness). Commonly compositing is specific to a geological domain.

Economic parameters

• The cut-off grade or economic limit used to define a Mineral Resource must provide 'reasonable prospects for economic extraction'. In establishing the cutoff grade, it must realistically reflect the location, deposit scale, continuity, assumed mining method, metallurgical processes, costs and reasonable long-term metal prices appropriate for the deposit. Assumptions should be clearly defined.

• Variations within the resource model (rock characteristics, metallurgy, mining methods, etc.) that may necessitate more than one cut-off grade or economic limit in different parts of the deposit model must be an ongoing consideration.

Mineral Resource model

• The Mineral Resource estimation techniques employed are dependent to a degree on the size and geometry of the deposit and the quantity of available data. Currently, most resource models are computer models constructed using one of several specialized commercially available software packages. Simple geometric methods may be acceptable in some cases (e.g. early stage deposit definition) but three-dimensional modeling techniques may be more appropriate for advanced projects.

• Model parameters (e.g. block size, model orientation) should be developed based on mining method (e.g. open pit versus underground, blast hole versus cut and fill mining), deposit geometry and grade distribution (e.g. polymetallic zoning in a sulphide deposit).

Estimation techniques

• The QP responsible for the Mineral Resource model must select a technique to estimate grades for the model. Methods range from polygonal or nearest neighbor estimates, inverse distance to a power, various kriging approaches (e.g. ordinary kriging, multiple indicator kriging) through to more complex conditional simulations. The choice of method will be based on the geology and complexity of grade distribution within the deposit and the degree to which high-grade outliers are present.

• In some complex models, it may be necessary to use different estimation techniques for different parts of the deposit.

• The QP should ensure that the selected estimation method is adequately documented and should not rely solely on the computer software to produce a comprehensive document or report 'trail' of the interpolation process.

Mineral Resource model validation

• The QP must ensure the Mineral Resource model is consistent with the primary data. The validation steps should include visual inspection of interpolated results on suitable plans and sections and compared with the composited data, checks for global and local bias (comparison of interpolated and nearest neighbor or declustered composite statistics), and a change of support check (degree of grade smoothing in the interpolation). It is recommended that manual validation of all or part of a computer-based Mineral Resource estimate be completed.

• For Mineral Resource models of deposits that have had mine production or are currently being mined, the validation mist include a reconciliation of production to the Mineral Resource model.

• A final step, best practice includes the re-evaluation of the economic parameters to confirm their suitability.

• As per the CIM Standards, Mineral Resource estimation involves the classification of resources into three classes. The criteria used for classification should be described in sufficient detail so that the classification is reproducible by others.

7.3.6 *Quantifying elements to convert a Mineral Resource to a Mineral Reserve*

This section forms the logical extension of the topics discussed in Section 7.3.5, 'Mineral Resource Estimation', and addresses factors required for the conversion of a Mineral Resource into a Mineral Reserve. These factors are provided, in the form of a checklist, for assembling information that should be considered prior to the process of estimating Mineral Reserves. This checklist referred to as quantifying elements or modifying factors, is not intended to be exhaustive. The QP should ensure that these elements/factors have been considered in adequate detail to demonstrate that economic extraction can be justified. The appropriate level of detail for each of these elements/factors is left to the discretion of the QP. However, in aggregate, the levels of detail and engineering must meet or exceed the criteria contained in the definition of a Preliminary Feasibility Study.

Quantifying Element or Modifying Factor Check List:

a) Mining:
- data to determine appropriate mine parameters, (e.g. test mining, RQD)
- open pit and/or underground
- production rate scenarios
- cut-off grade (single element, multiple element, dollar item)
- dilution: included in the Mineral Resource model or external factor(s) recovery with respect to the Mineral Resource model
- waste rock handling
- fill management (underground mining)
- grade control method
- operating cost
- capital cost
- sustaining capital cost

b) Processing
- sample and sizing selection: representative of planned mill feed, measurement of variability, is a bulk sample appropriate
- product recoveries
- hardness (grindability)
- bulk density

- presence and distribution of deleterious elements
- process selection
- operating cost
- capital cost
- sustaining capital cost

c) Geotechnical/Hydrological
- slope stability (open pit)
- ground support strategy (underground), test mining
- water balance
- area hydrology
- seismic risk

d) Environmental
- baseline studies
- tailings management
- waste rock management
- acid rock drainage issues
- closure and reclamation plan
- permitting schedule

e) Location and infrastructure
- climate
- supply logistics
- power source(s)
- existing infrastructure
- labor supply and skill level

f) Marketing elements or factors
- product specification and demand
- off-site treatment terms and costs
- transportation costs

g) Legal elements or factors
- security of tenure
- ownership rights and interests
- environmental liability
- political risk (e.g. land claims, sovereign risk)
- negotiated fiscal regime

h) General costs and revenue elements or factors
- general and administrative costs
- commodity price forecasts
- foreign exchange forecasts
- inflation
- royalty commitments
- taxes
- corporate investment criteria

i) Social issues
- sustainable development strategy
- impact assessment and mitigation
- negotiated cost/benefit agreement
- cultural and social influences

7.3.7 *Mineral Reserve estimation*

This section considers important factors in estimating a Mineral Reserve and documenting the estimation process. As a Mineral Reserve estimate represents the collation of work carried out by numerous professional disciplines, the QP producing the Mineral Reserve estimate must understand the significance of each discipline's work in order to assess economic viability. In addition, the QP should recognize that the time from discovery, to production, through to closure, of a mine is often measured in years and this timeframe makes good documentation an important aspect of the estimation process.

Preparation

The QP should document and use a methodology in estimating Mineral Reserves to ensure no significant factor is ignored. Pre-planning is important to identify the factors affecting the Mineral Reserve estimate. Utilizing a checklist to ensure all aspects are considered is good practice.

Mineral Reserve definition and classification is covered by the CIM Standards. Definitions do change from time-to-time and in the compilation of a Mineral Reserve estimate the QP should ensure the current definitions are being used. Of significance are the requirements that the material forms the basis of an economically viable project.

The test of economic viability should be well documented as part of the Mineral Reserve estimation process. The requirement for economic viability implies determination of annual cash flows and inclusion of all the parameters that have an economic impact.

Classification

The CIM Standards provide two categories for the definition of the Mineral Reserve, Proven Mineral Reserve and Probable Mineral Reserve and the QP must ensure that the minimum criteria are met prior to assigning these categories. The QP should be mindful of all the inputs used in establishing the Mineral Reserve that affect the confidence in the categories. The methodology of establishing the classification should be well documented and easily understood. Best practice includes providing a narrative description of the qualitative reasons behind the classification selection.

Where practical, empirical evidence (e.g. production data) should be used to calibrate and justify the classification.

Verification of inputs

It is the responsibility of the QP to ensure the verification of all inputs to the Mineral Reserve estimate. As the Mineral Reserve estimate is based on many data inputs, including the Mineral Resource model, it is important that the inputs and the consistency of the inputs be validated as part of the Mineral Reserve estimation process. A defined methodology to achieve this is considered best practice and the use of a protocol such as the checklist contained in Section 7.3.6 is recommended. Identification of critical aspects of the Mineral Reserve estimate is an important part of the input verification.

Application of Cut-off grade

Cut-off grade is a unit of measure that represents a fixed reference point for the differentiation of two or more types of material. Owing to the complexity of Mineral Reserve estimates, numerous cut-off grades may be required to estimate a Mineral Reserve, (e.g. the set point defining waste from heap leach ore and the set point defining heap leach ore from milled ore).

The cut-off grade(s) (the economic limit or pay limit) should be clearly stated, unambiguous and easily understood. Complex ores may require complicated procedures to determine cut-off grades and to define the Mineral Reserve. The procedures used to establish the cut-off strategies should be well documented, easily available for review, and clearly stated in disclosure statements.

Cut-off grade must be relevant to the grade distribution modeled for the Mineral Resource. If cut-off grades are outside the specified range, the QP must review model-reliability and a new model might be necessary.

A key objective of Mineral Reserve estimation is the successful extraction and delivery of a Mineral Resource for processing at the grade estimated. Due consideration should be given to the problems associated with selective mining where the cut-off grade is set high relative to the average grade of the Mineral Resource.

Practicality of mining

The practicalities of the mining/processing rates and methods for a deposit are important considerations in the estimation of a Mineral Reserve. The QP must assess the various proposals when estimating a Mineral Reserve. Care should also be taken to ensure that the mining equipment selected is appropriate for the deposit. Inappropriate equipment selection may have an effect on both dilution and extraction. The QP must have a high level of confidence in the viability of the mining and processing methods considered in determining the Mineral Reserves.

A QP should, when appropriate, consider of alternative mine/plant configurations. Selecting the appropriate mining and processing methods and rates may involve several iterations and will involve input from members of other disciplines. Trial evaluations, referred to as ‘trade-off’ or ‘scoping’ studies, may be required as a prelude to the completion of a Preliminary Feasibility Study.

Project risk assessment

While the classification of the Mineral Reserve allows the QP to identify technical risk in broad terms, best practice includes the establishment of a methodology to identify and rank risks associated with each input of the Mineral Reserve estimate. This will assist the QP in establishing the Mineral Reserve categorization, thus providing an understanding of the technical risk associated with the Mineral Reserve estimate. This methodology, ranking and analysis should be well documented.

Peer reviews

Best practice includes the use of an internal peer review of the Mineral Reserve estimate including inputs, methodology, underlying assumptions, the results of the estimate itself, and test for economic viability.

Audits/Governance

Upon completion of a Preliminary Feasibility Study, or in the case of significant changes to a Mineral Reserve estimate, best practice includes completion of a properly scoped audit carried out by an impartial QP. The audit should consider the methodology used, test the reasonableness of underlying assumptions, and review conformity to Mineral Reserve definitions and classification. The methodology for Mineral Reserve risk identification, assessment and management should also be included in the Mineral Reserve audit. The audit should be documented, distributed and responded to in a manner that recognizes good corporate governance.

Documentation

There are often several iterations of evaluations carried out over a protracted period of time prior to completion of a Preliminary Feasibility Study. Best practice includes appropriate documentation of the inputs/methodology/risks/assumptions used in these valuations so these will be available for future Mineral Reserve estimates. Information should be easily retrievable, readily available and catalogued in a manner that allows easy assessment of the history of the evaluations carried out and records the location of all relevant information/reports/etc. It is important to ensure that the information used in an evaluation, and understanding gained of a mineral deposit, is available for future work. Care should be taken in storage and consideration given to the continuous evolution of computer file formats and the impact this may have on previous work. File conversion of historic work into formats that allow continued access is recommended.

Mineral Reserve statements

Mineral Reserve statements should be unambiguous and sufficiently detailed for a knowledgeable person to understand the significance of, for example, cut-off grade and its relationship to the Mineral Resource. In the case of open pit Mineral Reserve estimates, the waste:ore ratio (the strip ratio) should be unambiguously stated. There should be an obvious linkage of the Mineral Reserve estimate to the Mineral Resource estimate provided in disclosure documents. Best practice includes documentation of those linkages (e.g. dilution and mining recovery) that were used in preparing the Mineral Reserve estimation.

7.3.8 *Reporting*

This section is primarily a compilation of references regarding reporting standards that should be considered when preparing reports on MRMR estimates. Although these standards are intended for public disclosure, they also represent the minimum requirement for best practice for all reporting.

National Instrument 43-101, Form 43-l01Fl and Companion Policy 43-101CP, establish standards for all oral and written disclosure made by an issuer concerning mineral projects that are reasonably likely to be made available to the public. All disclosure concerning mineral projects including oral statements and written disclosure in, for example, news releases, prospectuses and annual reports is to be based on information supplied by or under the supervision of a QP. Disclosure of information pertaining to MRMR estimation is to be made in accordance with industry standard definitions contained in the CIM Standards which have been incorporated by reference into the NI 43-101.

One of the objectives of the Estimation Best Practice Guidelines is to foster greater standardization of reporting in publicly disclosed documents. The recommendations included below represent further guidance and attempt to develop a reporting template, which should help reporting Canadian companies achieve greater standardization.

The QP should familiarize themselves with current disclosure regulations as part of preparing a MRMR estimate.

Reporting units

In the preparation of all reports and press releases, either metric or imperial units may be used. However, the following provisos apply:

- Reports must maintain internal consistency – metric and imperial units should not be used in different parts of the same report.
- The mixing of metric and imperial units (e.g. oz/tonne) is never acceptable.

The Committee considers that reporting in metric units is preferable.

Technical reports

A technical report shall be in accordance with Form 43-101 Fl, NI 43-101. The obligation to file a technical report arises in a number of different situations. These are set out in NI 43-101 in Part 4.

The CIM Standards on Mineral Resources and Mineral Reserves referenced in NI 43-101, provide additional guidance for reporting of MRMR estimates. A listing of the main recommendations and requirements is as follows:

(a) The QP is encouraged to provide information that is as comprehensive as possible in Technical Reports on MRMR.

(b) Fundamental data such as commodity price used and cut-off grade applied must be disclosed.

(c) Problems encountered in the collection of data or with the sufficiency of data must be clearly disclosed.

(d) Modifying factors applied to MRMR estimates such as cutting of high grades or resulting from reconciliation to mill data must be identified and their derivation explained.

(e) MRMR estimates are not precise calculations and, as a result should be referred to as estimates.

(f) Tonnage and grade figures should reflect the order of accuracy of the estimate by rounding off to the appropriate number of significant figures.

(g) Technical Reports of a Mineral Resource must identify one or more categories of ‘Inferred’, ‘Indicated’ and ‘Measured’ and Technical Reports of Mineral Reserves must specify one or both categories of ‘Proven’ and ‘Probable’. Categories must not be reported in combined form unless details of the individual categories are also provided. Inferred Mineral Resources cannot be combined with other categories and must always be reported separately.

(h) Mineral Resources must never be added to Mineral Reserves and reported as total Resources and Reserves. MRMR must not be reported in terms of contained metal or mineral content unless corresponding tonnages, grades and mining, processing and metallurgical recoveries are also presented.

(i) In cases where estimates for both Mineral Resources and Mineral Reserves are reported, a clarifying statement must be included that clearly indicates whether Mineral Resources are inclusive or exclusive of Mineral Reserves.
The Estimation Best Practice Committee recommends that Mineral Resources should be reported separately and exclusive of Mineral Reserves.

(j) Mineral Reserves may incorporate material (dilution) which is not part of the original Mineral Resource and exclude material (mining losses) that is included in the original Mineral Resource. It is essential that the fundamental differences between these estimates be understood and duly noted.

(k) In preparing a Mineral Reserve report, the relevant Mineral Resource report on which it is based should be developed first. The application of mining and other criteria to the Mineral Resource can then be made to develop a Mineral Reserve statement that can also be reconciled with the previous comparable report. A detailed account of the differences between current and previous estimates is not required, but sufficient commentary should be provided to enable significant differences to be understood by the reader. Reconciliation of estimates with production whenever possible is required.

(l) Where Mineral Reserve estimates are reported, commodity price projections, operating costs and mineral processing/metallurgical recovery factors are important and must be included in Technical Reports.
The Committee considers that when reporting a Mineral Reserve mineable by open pit methods, the waste-to-ore ration must be disclosed.

(m) Reports must continue to refer to the appropriate categories of Mineral Resources until technical feasibility and economic viability have been established by the completion of at least a Preliminary Feasibility Study.

(n) Reporting of mineral or metal equivalence should be avoided unless appropriate correlation formulae including assumed metal prices, metallurgical recoveries, comparative smelter charges, likely losses, payable metals, etc. are included.

(o) Broken mineralized inventories, as an example, surface and underground stockpiles, must use the same basis of classification outlined in the CIM Standards. Mineralized material being processed (including leaching), if reported, should be reported separately.

(p) Reports of MRMR estimates for coal deposits should conform to the definitions and guidelines on Paper 88-21 of the Geological Survey of Canada. ‘A Standardized Coal Resource/Reserve Reporting System for Canada’.

(q) When reporting MRMR estimates relating to an industrial mineral site, QP must make the reader aware of certain special properties of these commodities and relevant standard industry specifications.

(r) Reports of MRMR estimates of diamonds or gemstones must conform to the definitions and guidelines found in ‘Reporting of Diamond Exploration Results, Identified Mineral Resources and Ore Reserves’ published by the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories. These definitions and guidelines remain in force until/if they are replaced by guidelines of the Diamond Exploration Best Practice Committee, the relevant sections of these guidelines, or other guidelines which may be accepted by the CSA or CIM.

Annual reports

Written disclosure (including annual reports) of MRMR is covered by Part 3.4 of NI 43-101. Further reference is made in Parts 1.3, 1.4, 2.1, and 2.2 of NI 43-101.

An issuer shall ensure that all written disclosure of MRMR on a property material to an issuer includes:

(a) the effective date of each estimate of MRMR;

(b) details of quantity and grade or quality of each category of MRMR;

(c) details of key assumptions, parameters and methods used to estimate the MRMR;

(d) a general discussion of the extent to which the estimate of MRMR may be materially affected by any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues; and

(e) a statement that Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Further to these requirements, the Committee recommends that:

(a) MRMR estimates should be reported as a tabulation.

(b) The name of the appropriate QP must be included with the estimate. The relationship of the QP to the reporting company should be stated. Note that in the estimation of Mineral Reserves, the services of a number of different QP are likely to have been employed. Under CSA guidelines a corporation may designate a reporting QP with overall responsibility for the estimates and, if so, the name must be included. In some Canadian Provinces, it may not be appropriate to designate a reporting QP.

(c) These data remain 'estimates' and should be reported as such.

(d) NI 43-101 Part 3.4 (c) requires those details of key assumptions, parameters and methods used to estimate MRMR must be included. These details could be included as a footnote in the MRMR section:

- Metal prices assumptions.
- Cut-off grades.
- Ore losses and dilution.
- Mill recoveries.
- Estimation methodologies.
- It is essential that the estimates conform to the CIM Standards on Mineral Resources and Mineral Reserves, Definitions and Guidelines, or equivalent foreign code as described in Part 7 of NI 43-101. A note stating the standard being used must be included.
- Year-to-year changes in MRMR must be included, together with the reasons for the changes.
- A statement whether the Mineral Resources are inclusive or exclusive of Mineral Reserves. In the interests of standardization, the Committee recommends that Mineral Resources should be reported exclusive of Mineral Reserves in Annual Reports.
- Date of the estimate of MRMR.

Press releases

The content of press releases discussing MRMR is covered in Section 3.0 of Appendix B of Disclosure Standard No. 1450-025 of the Toronto Stock Exchange (TSX), and Corporate Finance Manual Appendix 3F-Mining Standards Guidelines-Policy 3.3-Timely Disclosure of the TSX Venture Exchange.

Section 3.1 (Definitions) states that estimates 'must conform to the definitions contained in NI 43-101'. Section 3.2 (Use) covers a number of points regarding reporting:

- All MRMR estimations must disclose the name of the QP responsible for the estimate and the relationship of the QP the reporting company. The company must also state whether, and how, any independent verification of the data has been published.

- The statement must make a clear distinction between Mineral Resources and Mineral Reserves.

- MRMR should, wherever possible, be published in such a manner so as not to confuse the reader as to the potential of the deposit. Inferred Resources must not be aggregated with Indicated and Measured Resources. Any categories of MRMR that are aggregated must also be disclosed separately.

- When Mineral Reserves are first reported, the key economic parameters of the analysis must be provided. These will include:
  - Operating and capital cost assumptions.
  - Commodity prices (If commodity prices used differ from current prices of the commodities which could be produced, an explanation should be given, including the effect on the economics of the project if current prices were used. Sensitivity analysis may be used in this section).
  - All reported quantities of MRMR must be expressed in terms of tonnage and grade or characteristics. Contained ounces must not be disclosed out of the context of the tonnage and grade of a deposit.
  - MRMR for polymetallic deposits may not be disclosed in terms of 'metal equivalents' except in limited circumstances as set out in NI 43-101 Fl, 19(k) and in the CIM Standards on MRMR. It is also inappropriate to refer to the gross value or in situ value of MRMR.

The Committee recommends that any press release that reports initial estimates of MRMR include all of the information listed under 'Annual Reports' above. Subsequent press releases may refer back to the initial press release. It should be noted that the CSA is reviewing this requirement.

Annual Information Filing

The requirements for reporting for an AIF (Annual Information Filing) are set out in the CSA document National Instrument 44-101 and Form 44-101 Fl.

7.3.9 *Reconciliation of Mineral Reserves*

Production monitoring and reconciliation of Mineral Reserves are the ultimate activities by which the QP can continuously calibrate and refine the Mineral Reserve estimate. While this section is primarily concerned with Mineral Reserves, the only valid confirmation of both the Mineral Resource and Mineral Reserve estimate is through appropriate production monitoring and reconciliation of the estimates with mine and mill production.

The QP must take into consideration the results of any grade-tonnage reconciliation in any public disclosure of MRMR estimates.

Production monitoring is the grade control and tonnage accounting function performed at an operating mine. This function provides the information required to minimize dilution, maximize mineral recovery and supply a consistent and balanced feed to the process plant as required. Grade and tonnage control comprises representative sampling of production sources, establishing ore/waste boundaries and accurately recording production tonnes and grade.

Reconciliation is required to validate the Mineral Reserve estimate and allows a check on the effectiveness of both estimation and operating practices. Since the MRMR estimates are based on much wider spaced sampling than used for production, reconciliation identifies anomalies, the resolution of which may prompt changes to the mine/processing operating practices and/or to the estimation procedure.

Production monitoring

The following should be given consideration in effective production monitoring and forms part of an ongoing quality control program, which takes into account the closer spaced production sampling and mapping:

Minimize Dilution/Maximize Mineral Recovery

- When ore/waste contacts are visual, mine operators can classify ore and waste easily and send material to the correct destination, however, production monitoring is still required.

- Where assay boundaries are used to delineate ore, appropriately spaced representative samples are required to estimate the locations of economic margins.

- Given the negative economic consequences of misclassification, diligence is necessary to ensure that mined ore and waste types are delivered to the appropriate destinations.

Characterize the ore to ensure the requested metallurgical balance is achieved.

- This may require geological mapping and logging of blast hole chips if ore types are visually distinguishable.

- Where characterization is non-visual, other testing is required to achieve appropriate blending.

Characterize waste to allow for potential mixing (blending) or separation based on environmental requirements (e.g. acid rock drainage control).

Monitor deleterious or by-product constituents that might compromise or enhance mill recoveries and concentrate quality.

Record accurately production tonnes and grade to permit reconciliation of mine production to the processing facility and ultimately to the Mineral Reserve estimate. Mine production needs to be reconciled to mine surveys on a regular basis, commonly monthly.

Ensure that accurate measurements of in-situ bulk densities for various ore and waste types have been determined so that volumes can be converted appropriately to tonnes. Periodic checks are required of in situ bulk densities, truck and bucket factors and weightometers.

Develop appropriate sampling protocols and continuously evaluate them to ensure representative sampling in both the mine and plant.

Ensure that acceptable QA/QC procedures are being followed at all laboratories being utilized.

Maps of workings/benches at appropriate scales must be kept current to provide:

- Geological information to compare to the MRMR model and update where appropriate.
- Ore type classification information to compare to the MRMR model and provide data for blending requirements of the process plant.
- Structural information that may impact MRMR continuity or provide valuable geotechnical information.
- Current, detailed, grade distribution from production sampling which, when combined with geological information, may be used to improve the grade interpolation in the MRMR estimation process.
- Information that will assist in quantifying dilution and mining losses, which can be used for future MRMR estimations.

Volumetric surveys should be retained so they can be used for future reconciliations.

Production reconciliation

The following should be considered in undertaking production reconciliation:

Reconciliation of mine and mill

- Reconciliation between the mine and mill production should be done on a regular basis but monitored on a daily basis to ensure accuracy of sampling and record keeping. Current best practice is considered to be reconciliation on a monthly basis.
- A reconciliation provides checks for discrepancies, which may require changes to operational procedures or the MRMR model.
- Mine production is usually reconciled to the plant since measurement in the plant is generally accepted to be more accurate. Significant discrepancies and resulting adjustment factors should be explained and reported.
- On a yearly basis, mill production should be reconciled with the final concentrate, bullion or mineral shipped and resulting adjustment factors should be explained and reported.

Reconciliation of production and MRMR estimates:

- Reconciliation should be done at least annually to coincide with the corresponding MRMR statement.
- Reconcile production to estimated depletion of Mineral Reserves; any discrepancies in grade and/or tonnes should be explained and appropriate changes should be made to operating practice or the MRMR estimation process.

Annual review of remaining MRMR

- Remaining MRMR at operating mines should be reviewed at least annually and should reflect changes in the underlying criteria, including long term commodity price forecasts, increases or decrease in costs, changes in metallurgical processing performance, and changes in mining methods, dilution or mining recovery.

- Cut-off grades or economic limits should be reassessed and updated at least annually.

- Remaining MRMR should be adjusted for improved geological interpretation due to drilling or mapping.

- The rationale for any changes to operating practice or to MRMR estimation procedures must be documented.

End-of-year MRMR estimates should be reconciled with previous year's MRMR estimates by showing a balance sheet detailing the changes due to mining extraction, commodity price change, cost changes, additions or deletions due to drilling or mining losses/gains, among others.

7.3.10 *Selected References*

The Committee considers that there are several documents and publications which are essential or useful in dealing with best practice requirements for the estimation of MRMR.

CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines. Prepared by the CIM Standing Committee on Reserve Definitions, October 2000. CIM Bulletin Vol. 93, No. 1044, pp. 53–61.

Vallée, M. and Sinclair, A.J. (eds.) (1998), 'Quality Assurance, Continuous Quality Improvement and Standards in Mineral Resource Estimation' Exploration and Mining Geology (Volume 7, nos. I and 2, 1998).

Mineral Resource and Ore Reserve Estimation – The AusIMM Guide to Good Practice, Monograph 23. Editor: A.C. Edwards, The Australasian Institute of Mining and Metallurgy: Melbourne.

National Instrument 43-101 – Standards of Disclosure for Mineral Projects. Ontario Securities Commission Bulletin 7815, November 17, 2000.

Exploration Best Practice Guidelines: included in 'CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines'. Prepared by the CIM Standing Committee on Reserve Definitions, October, 2000. CJM Bulletin Vol. 93, No. 1044, pp. 53–61.

Guidelines for the Reporting of Diamond Exploration Results. Available at www.cim.org and in press.

REFERENCES AND BIBLIOGRAPHY

Anonymous. 1999. Guide for reporting exploration information, mineral resources and mineral reserves – an abstract. Mining Engineering. 51(6): 82–84.

Arick, A. 2002. Comparison of resource methodologies with a new approach. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 57–64. Alaska: SME.

Arik, A. 1999. An alternative approach to resource classification. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 45–54. Colorado School of Mines.

Askari-Nasab, H., and K. Awuah-Offei. 2008. An agent-based framework for open pit mine planning. CIM Magazine. December 2008/January 2009. pp. 145–154.

AusIMM. 1995. Code and guidelines for technical assessment and/or valuation of mineral and petroleum assets and mineral and petroleum securities for independent expert reports (The VALMIN Code). http://www.ausimm.com.au/codes/valmin/valcode0.asp.

AusIMM. 1998a. The revised and updated VALMIN Code: Code and guidelines for technical assessment and/or valuation of mineral and petroleum assets and mineral and petroleum securities for independent expert reports (The VALMIN Code). http://www.ausimm.com.au/codes/valmin.

AusIMM. 1998b. The revised VALMIN Code and guidelines: An aide memoire to assist its interpretation. http://www.ausimm.com.au/codes/valmin/eexp11.asp.

AusIMM. 2004. JORC information. www.jorc.org/main.

AusIMM. 2005. Australasian Code for the reporting of exploration results, mineral resources and ore reserves: The JORC Code – 2004 Edition. www.jorc.org/pdf/jorc2004print.pdf.

Bre-X. 2005. Bre-X/Bresea shareholder class action information web site. www.brexclass.com

Briggs, T., Dormer, D., Darvall, M., and R. Fare. 2006. Putting Geology and Engineering Back into Resource/Reserve Determination, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 137–142.

Camisani-Calzolari, F.A. 2004. National and international codes for reporting mineral resources and reserves: Their relevance, future and comparison. JSAIMM. 104(5): 297–305.

Camisani-Calzolari, F.A., and D.G. Krige. 2001. The SAMREC Code seen in a global context. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 39–44. Beijing: Balkema.

Cawood, F.T. 2004. Mineral property valuation in South Africa: A basket of assets and legal rights to consider. JSAIMM. 104(1): 45–52.

Cawood, F.T. 2004. The Mineral and Petroleum Resources Development Act of 2002: A paradigm shift in mineral policy in South Africa. JSAIMM. 104(1): 53–63.

Cawood, F.T. 2004. Towards a mineral property valuation code: Considerations for South Africa. JSAIMM. 104(1): 35–44.

Cawood, F.T., and A.S. Macfarlane. 2003. The Mineral and Petroleum Royalty Bill – Report to the National Treasury.

CIM. 2000. Exploration Best Practice Guidelines. Aug 20. www.cim.org/definitions/exploration BESTPRACTICE.pdf.

CIM. 2003a. Standards and guidelines for valuation of mineral properties. February. www.cim.org/committees/CIMVal_final_standards.pdf.

CIM. 2003b. Guidelines for the reporting of diamond exploration results. Mar 9. www.cim.org/committees/diamond_exploration_final.cfm.

CIM. 2003c. Estimation of mineral resources and mineral reserves: Best practice guidelines. Nov 23. www.cim.org/committees/estimation.cfm.

CIM. 2004. Definition Standards – On mineral resources and mineral reserves. www.cim.org/committees/StdsAppNovpdf.

CIM. 2005. Standards and guidelines. www.cim.org/committees/guidelinesStandards_main.cfm.

Dagdelen, K., and K. Kawahata. 2008. Value creation through strategic mine planning and cutoff-grade optimization. Mining Engineering. 60(1): 39–45. January.

Davis, G.A., and J.E. Tilton. 2008. Why the resource curse is a concern. Mining Engineering. 60(4): 29–32. April.

Dessureault, S. 2007. Data mining, mining data: energy consumption modeling. CIM Magazine. September/October. pp. 263–270.

Dessureault, S., Lipsett, M., and M. Scoble. 2008. Achieving the benefits of information technology in surface and underground mining processes. CIM Magazine. March/April. pp. 21–25.

Dessureault, S.D., and A.P. Schissler. 2006. Challenges to overcome in applying information technology in the mining industry. CIM Magazine. March/April. pp. 59–67.

Diatchkov, S.A. 1994. Principles of classification of reserves and resources in the CIS countries. Mining Engineering. 46(3): 214–217.

Dowd, P.A. 1999. Reserves and resources – Continuity and levels of confidence. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 93–100. Colorado School of Mines.

Ellis, T.R. 2000. The difference between a value estimate and an appraisal. Pre-print Paper 00-119. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Ellis, T.R. 2001. U.S. and international valuation standards – The future. Pre-print Paper 01-161. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.

Ellis, T.R. 2002. Recent developments in international valuation standards. 2002. Pre-print Paper 02-168. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.

Ellis, T.R. 2002. The U.S. mineral property valuation patchwork of regulations and standards. CIM Bulletin 95(1059): 110–118.
Ellis, T.R. 2003. An international perspective on U.S. minerals appraisal standards development. Paper 03-109. SME Annual Meeting. Feb 24–26. Cincinnati, OH.
Ellis, T.R. 2004. International perspective on U.S. minerals appraisal standards development. Mining Engineering. 56(3): 44–48.
Ellis, T.R. 2004. Philosophy and application of the international valuation standards for minerals and petroleum. Paper 04-133. SME Annual Meeting. Feb 23–25. Denver, CO.
Ellis, T.R. 2005. Implementation of the international valuation standards. Pre-print Paper 05-093. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.
Ellis, T.R. 2006. Implementation of the international valuation standards. Mining Engineering. 58(2): 62–64. February.
Ellis, T.R., Abbott, D.M., Jr., and H.J. Sandri. 1999. Trends in the regulation of mineral deposit valuation. Pre-print Paper 99-029. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.
Emery, X., Bertini, J.P., and J.M. Ortiz. 2005. Resource and reserve evaluation in the presence of imprecise data. CIM Magazine. September/October. pp. 366–377.
Francis, D. 1997. Bre-X: The Inside Story. Key Porter Books, Toronto. 240 pp.
Goold, D., and A. Willis. 1997. Bre-X Fraud. McClelland & Stewart. 272 pp.
Graff, R.P. 2008. Recognition of value beyond proven and probable reserves in business combinations. Mining Engineering. 60(8): 45–50. August.
Heintz, J.H. 2002. Valuation of mineral interests in condemnation cases. Pre-print Paper 02-74. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.
Kirk, W.S. 1998. Iron ore reserves and mineral resource classification systems. Skillings Mining Review. June 6. pp. 4–11.
Kral, S. 2003. Experts discuss reserves reporting at SME meeting. Mining Engineering. 55(12): 23–26.
Lampard, E., and T.G. Joseph. 2011. Towards formalizing qualitative operations input into mine planning. CIM Journal. 2(4): 178–188.
Lawrence, M.J. 2002. The VALMIN Code – The Australian experience. CIM Bulletin 95(1058): 76–81.
Lawrence, R.D. 2002. Valuation of mineral properties without mineral resources: A review of market-based approaches. CIM Bulletin 95(1060): 103–106.
Lilford, E.V., and R.C.A. Minnitt. 2002. Methodologies in the valuation of mineral rights. JSAIMM. 102(7): 369–384.
Limb, J.S. 2001. Mineral appraisal issues in court testimony. Pre-print Paper 01-90. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.
Mcfarlane, A.S. 2002. A Code for the valuation of mineral properties and projects in South Africa. JSAIMM. 102(1): 37–48.
Millar, D.L., Levesque, M., Lyle, G., and K. Bullock. 2012. Enabling progressive energy management practice in minerals operations. CIM Journal. 3(3): 176–184.
Mwasinga, P.P. 2001. Approaching resource classification: General practices and the integration of geostatistics. 29th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 97–107. Beijing: Balkema.
Nel, D. 2007. A logical approach to determination of density and porosity values for iron ore deposits. Iron Ore 2007 Conference, Proceedings. AusIMM Publication Series 6/2007. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 141–146.
Postle, J.T. 2009. Canadian system for the disclosure of mineral property information. AusIMM Journal. Issue 6. Dec.
Ramazan, S., and R. Dimitrakopoulos. 2006. Probabilistic practical optimization of production scheduling for multi-element deposits. CIM Magazine. March/April. pp. 49–58.
Rendu, J.M. 1999. International definitions of mineral resources and mineral reserves – A U.S. perspective. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 27–34. Colorado School of Mines: CSM.
Rendu, J.M. 2003. SME meets with the SEC – resources and reserves reporting discussed. Mining Engineering. 55(7): 35.

Rendu, J.M. 2005. SME submits reserves, resources reporting recommendations to SEC. Mining Engineering. 57(8): 8–9.

Roscoe, W.E. 2002. Valuation of mineral exploration properties using the cost approach. CIM Bulletin 95(1059): 105–109.

SAIMM. 2000. South African Code for reporting of mineral resources and mineral reserves (the SAMREC Code). www.saimm.co.za/pages/comppages/samrec_version.pdf.

SEC. 2005. Description of property by users engaged or to be engaged in significant mining operations. Guide 7. www.sec.gov/divisions/corpfin/forms/industry.htm#secguide7/.

Sides, E.J. 2010. A time-value framework for reporting mineral assets. CIM Journal. 1(1): 34–43.

Sinclair, A. 2001. High-quality, axiomatic to high-quality resource/reserve estimates. CIM Bulletin 94(1049): 37–41.

SME. 1991. A guide for reporting exploration information, resources and reserves. Mining Engineering. 53(4): 379–384.

SME. 1999. A guide for reporting exploration information, mineral resources, and mineral reserves. www.smenet.org.digital_library/index.cfm.

SME. 2003. Reporting mineral resources and reserves: Regulatory, financial, legal, accounting, managerial and other aspects. Conference October 1–3, 2003. SME.

Spence, K. 2002. An overview of valuation practices and the development of a Canadian code for the valuation of mineral properties. CIM Bulletin. 95(1057): 77–82.

Spence, K.N., and W.E. Roscoe. 2001. Development of Canadian standards and guidelines for valuation of mineral properties. Pre-print Paper 01-138. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.

Stephenson, P.R., Allman, A., Carville, D.P., Stoker, P.T., Mokos, P., Tyrrell, J., and T. Burrows. 2006. Mineral Resource Classification – It's Time to Shoot the 'Spotted Dog'!, AUSIMM Conference Proceedings, 6th International Mining Geology Conference, Darwin, NT, August 2006, pp. 91–95.

Stephenson, P.R., and P.T. Stoker. 1999. Classification of mineral resources and ore reserves. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 55–68. Colorado School of Mines: CSM.

Stoker, P., and C. Moorhead. 2009. Confidence in Resource Estimates – Beyond Classification. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 269–272.

Subelj, A. 2002. Why 'World Code' has to cover all mineral resources, not only 'Reserves'. 30th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 3–12. Alaska: SME.

Sykes, T. 1978. The Money Miners: Australia's Mining Boom 1969–1970. Wildcat Press, Sydney. 388 pages.

Sykes, T. 1988. Two Centuries of Panic: A History of Corporate Collapses in Australia. Allen & Unwin. London. 593 pages.

Sykes, T. 1996. The Official History of Blue Sky Mines. Australian Financial Review. 165 pages.

USGS. 1976. Principles of the mineral resource classification system of the U.S. Bureau of Mines and U.S. Geological Survey. Bulletin 1450-A.

USGS. 1980. Principles of a resource/reserve classification for minerals. USGS Circular 831. http://pubs.er.usgs.gov/pubs/cir/cir831/ or http://imcg.wr.usgs.gov/usbmak/c831.html.

Valleá, M. 1999. Resource/reserve inventories: What are the objectives? CIM Bulletin 92(1031): 151–155.

Valleé, M. 1999. Toward resource/reserve estimation, inventory and reporting standards. 28th International Symposium on Application of Computers and Operations Research in the Mineral Industry: 69–76. Colorado School of Mines.

Vaughan, W.S., and S. Felderhof. 2005. International mineral resource and mineral reserve classification and reporting systems. CIM Magazine. March/April. pp. 229–241.

Wilkinson, W.A. 2010. Benefits of building an efficient mine planning process. Mining Engineering. 62(10): 30–42. September.

Yeates G., and D. Hodson. 2006. Resource Classification – Keeping the End in Sight, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 97–104.

Yennamani, A.L., Aquirre, S., and P. Mousset-Jones. 2011. JORC and mineral resource classification systems. Mining Engineering. 63(2): 34–39. February.

REVIEW QUESTIONS AND EXERCISES

1. Why is it important that the terms used to describe and classify mineral resources be precisely defined and accurately applied?
2. What is meant by the popularly-used term ‘transparency’?
3. Who benefits from ‘smoke-and-mirrors’ in the mining business?
4. What does JORC denote? Why was it formed? What did it produce? Why was it important to the worldwide mining business?
5. Briefly describe the Bre-X scandal. Who were the main participants? Whose resource/reserve definitions were used? What was the outcome?
6. What are the main principles governing the operation and application of the JORC code? Briefly describe each.
7. What is meant by a Public Report or Public Reporting? What is the relationship to the JORC code?
8. What is meant by a ‘Competent Person’?
9. What is the role of a ‘Competent Person’?
10. What are the qualifications of a ‘Competent Person’?
11. What is meant by relevant experience?
12. Are there penalties for unprofessional work? What is the enforcement mechanism?
13. Figure 7.1 sets out the framework for classifying tonnage and grade estimates to reflect different levels of geological confidence and different degrees of technical and economic evaluation. How is the figure to be applied in practice?
14. What are meant by modifying factors?
15. What are the rules regarding the reporting of exploration results?
16. How can Table 7.1 be profitably used?
17. What is meant by the term ‘prescriptive’? What is meant by the term ‘guideline’? How do the two terms differ?
18. What are the rules regarding the reporting of mineral resources?
19. What is covered by the term ‘Mineral Resource’?
20. Why is the term ‘reasonable prospects for eventual extraction’ important?
21. What is meant by the term ‘Inferred Mineral Resource’?
22. What is meant by the term ‘Indicated Mineral Resource’?
23. What is meant by the term ‘Measured Mineral Resource’?
24. How do the three categories listed in problems 21, 22 and 23 differ? On what basis do you choose the appropriate category?
25. In reporting resources, what is the guidance concerning significant figures?
26. What is covered by the term ‘Ore Reserve’?
27. What is meant by the term ‘economically mineable’?
28. What is meant by the term ‘Probable Ore Reserve’?
29. What is meant by the term ‘Proved Ore Reserve’?
30. How do the two categories listed in problems 28 and 29 differ? On what basis do you choose the appropriate category?
31. Discuss in detail the conversion process between resources and reserves and within categories.

32. How is the relative accuracy and/or confidence in the estimates expressed?
33. How should Ore Reserve and Mineral Resource data be reported?
34. What is meant by 'reconciliation with production data'?
35. How does the JORC Code deal with stockpiles, low grade mineralization and tailings?
36. What is the purpose of the 'CIM Best Practice Guidelines for the Estimation of Mineral Resources and Mineral Reserves – General Guidelines'?
37. What is meant by the term 'Qualified Person (QP)'? Is it the same as a 'Competent Person' as used in the JORC code?
38. Define the following terms based on the CIM code:
    - Quality Assurance
    - Quality Control
    - Mineral Resource
    - Mineral Reserve
    - Preliminary Feasibility Study
39. Describe the establishment, content and importance of the Resource Database.
40. What is the difference between primary data and interpreted data? Give an example of each.
41. Describe the guidelines for visualization of the primary data.
42. What are the guidelines concerning the visualization of interpreted data?
43. What are the guidelines for data collection, recording, storing and processing?
44. Discuss the required data QA/QC program.
45. In section 7.3.4 'Geological Interpretation and Modeling' the following appears:

    > 'These guidelines outline requirements for interpretation of geological data, the consideration of economic and mining criteria and the linkage of that information to the grade distribution of the MRMR model'.

    Are these 'guidelines' or 'requirements'? What is the difference? Are these prescriptive?
46. Summarize the material in section 7.3.4 'Geological Interpretation and Modeling' using a step by-step approach. Does it make good sense? Is it a good checklist for the first part of a senior thesis project? Why or why not?
47. Summarize the material in section 7.3.5 'Mineral Resource Estimation' using a step-by-step approach. Does it make good sense? Is it a good checklist for the second part of a senior thesis project? Why or why not?
48. Review the checklist included in section 7.3.6 'Quantifying Elements to Convert a Mineral Resource to a Mineral Reserve.' How would this be used as part of a senior thesis project? Is it complete? Are there factors missing?
49. Summarize the material in section 7.3.7 'Mineral Reserve Estimation' using a step-by-step approach. Does it make good sense? Does this form a good checklist for a senior thesis project? Why or why not?
50. Summarize the discussion on the selection and application of cutoff grade.
51. How does the 'practicality' of mining enter into the process?
52. What is the value of performing a risk assessment? How should it be performed?
53. What is a Mineral Reserve audit?

54. How should mineral reserves be stated?
55. The senior thesis is presented as a comprehensive technical report. Which items under the heading Technical Report in section 7.3.8 ‘Reporting’ would be useful in that regard? Summarize them.
56. Summarize the discussion of ‘Mineral Reserve Reconciliation’.
57. Summarize the discussion on ‘Production Monitoring’.
58. Summarize the discussion on ‘Production Reconciliation’.

CHAPTER 8

Responsible mining

8.1 INTRODUCTION

At the beginning of Chapter 1, 'ore' was defined as:

Ore: A natural aggregation of one or more solid minerals that can be mined, processed and sold at a profit.

The meaning of 'profit' was defined very simply in Equation (1.1) as

$$\text{Profits} = \text{Revenues} - \text{Costs} \tag{1.1}$$

Replacing 'revenues' and 'costs' by their equivalent expressions in Equation (1.1), one obtains Equation (1.4).

$$\text{Profits} = \text{Material sold(units)} \times (\text{Price/unit} - \text{Cost/unit}) \tag{1.4}$$

The profit (or loss) incurred is often referred to as the 'bottom line' since it is the final calculation made in a financial analysis and appears at the bottom of a financial ledger sheet. In simple terms, it is the amount of money remaining after all of the bills have been paid. At the conclusion of a presentation on a potential mining project, the first focus is on the 'bottom line.' Other financial indicators such as the NPV, ROR, DCFROR and the payback period as discussed in Chapter 2 will also be closely examined. For a project to be considered a 'Go,' the ROR, for example, must be greater than the 'hurdle rate' (the minimum ROR), required by the company. If the projected rate of return is below the 'hurdle', then the project may be re-evaluated using other parameters/assumptions or put aside. The potential 'ore reserve' is no more but remains a mineralized occurrence or mineral concentration.

The requirement to make a profit commensurate with the risk being taken is a key aspect of any business but particularly in mining where the risks can be very high. Because of this, it is of value to consider the notion of 'profit' in today's world in greater detail.

In this respect, Equation (1.4) can be re-written as

$$\text{Profit/unit} = \text{Price/unit} - \text{Cost/unit} \tag{8.1}$$

For metals and many other minerals, the price received per unit is set on the international market and will be considered fixed for the purposes of this discussion. The Cost/unit is something that is project dependent. For reasons that will become obvious as the reader traverses this chapter, it will be expressed as

$$\text{Cost/unit} = \text{Extraction cost/unit} + \text{Environmental cost/unit} + \text{Social cost/unit} \tag{8.2}$$

For any project there will be environmental and social impacts. It is clear that neither the natural nor the societal setting of the orebody will be the same pre- and post-project. Measures will be taken to address these changes. Each level of mitigation will have an associated cost that will directly impact the bottom line. When simply considering profits, one should set the environmental and social cost components in Equation (8.2) to zero. This has, unfortunately, been the approach taken by many mining operations in the past and even by some today. However, positive changes have occurred. In much of the world today, this approach is unacceptable and the project would not go forth. Hence the profits would be zero. The other extreme would be to allocate a very high level of funding to mitigate all potential adverse environmental and social impacts. When these costs are entered into Equation (8.2), and subsequently into Equation (8.1), the resulting profits become unacceptably low and the project would not go forth. This forms the other extreme.

Assuming that the project on the whole is a potentially 'good' one for all of the stakeholders, significant efforts must be made to establish appropriate levels of environmental and social mitigation lying between these two extreme cases that will still lead to an acceptable economic result. This approach and the eventual results achieved are encompassed by the phrase 'Responsible Mining.' This phrase has a very clear meaning and directly includes the 'Economic Responsibility', the 'Environmental Responsibility', and the 'Social Responsibility' components that all responsible mining companies must satisfactorily address when embarking on a new project and when continuing with a current project.

Today, unfortunately, it is not the phrase 'Responsible Mining' which is on the tongues and in the minds of mining executives but rather the phrase 'Sustainable Development.' Although a connection between mining and sustainable development can and has been made, the connection must be described as rather weak. 'Responsible Mining' is one important *response* to 'Sustainable Development.' As such, the continuing world-wide focus on 'Sustainable Development' is important to maintaining and even accelerating the pace toward the universal adoption of 'Responsible Mining' practices.

This chapter will review some of the milestones in the 'Sustainable Development' movement and the actions being taken by the mining industry in moving toward the universal adoption of 'Responsible Mining' practices.

8.2 THE 1972 UNITED NATIONS CONFERENCE ON THE HUMAN ENVIRONMENT

In June of 1972, the United Nations Conference on the Human Environment (UNCHE) met in Stockholm to consider the need for a common outlook and for common principles to inspire and guide the peoples of the world in the preservation and enhancement of the human environment. At the conclusion of the conference it was proclaimed that (UNEP (1972a, 1972b)):

1. Man is both creature and molder of his environment, which gives him physical sustenance and affords him the opportunity for intellectual, moral, social and spiritual growth. In the long and tortuous evolution of the human race on this planet a stage has been reached when, through the rapid acceleration of science and technology, man has acquired the power to

transform his environment in countless ways and on an unprecedented scale. Both aspects of man's environment, the natural and the man-made, are essential to his well-being and to the enjoyment of basic human rights the right to life itself.

2. The protection and improvement of the human environment is a major issue which affects the well-being of peoples and economic development throughout the world; it is the urgent desire of the peoples of the whole world and the duty of all Governments.

3. Man has constantly to sum up experience and go on discovering, inventing, creating and advancing. In our time, man's capability to transform his surroundings, if used wisely, can bring to all peoples the benefits of development and the opportunity to enhance the quality of life. Wrongly or heedlessly applied, the same power can do incalculable harm to human beings and the human environment. We see around us growing evidence of man-made harm in many regions of the earth: dangerous levels of pollution in water, air, earth and living beings; major and undesirable disturbances to the ecological balance of the biosphere; destruction and depletion of irreplaceable resources; and gross deficiencies, harmful to the physical, mental and social health of man, in the man-made environment, particularly in the living and working environment.

4. In the developing countries most of the environmental problems are caused by under development. Millions continue to live far below the minimum levels required for a decent human existence, deprived of adequate food and clothing, shelter and education, health and sanitation. Therefore, the developing countries must direct their efforts to development, bearing in mind their priorities and the need to safeguard and improve the environment. For the same purpose, the industrialized countries should make efforts to reduce the gap between themselves and the developing countries. In the industrialized countries, environmental problems are generally related to industrialization and technological development.

5. The natural growth of population continuously presents problems for the preservation of the environment, and adequate policies and measures should be adopted, as appropriate, to face these problems. Of all things in the world, people are the most precious. It is the people that propel social progress, create social wealth, develop science and technology and, through their hard work, continuously transform the human environment. Along with social progress and the advance of production, science and technology, the capability of man to improve the environment increases with each passing day.

6. A point has been reached in history when we must shape our actions throughout the world with a more prudent care for their environmental consequences. Through ignorance or indifference we can do massive and irreversible harm to the earthly environment on which our life and well being depend. Conversely, through fuller knowledge and wiser action, we can achieve for ourselves and our posterity a better life in an environment more in keeping with human needs and hopes. There are broad vistas for the enhancement of environmental quality and the creation of a good life. What is needed is an enthusiastic but calm state of mind and intense but orderly work. For the purpose of attaining freedom in the world of nature, man must use knowledge to build, in collaboration with nature, a better environment. To defend and improve the human environment for present and future generations has become an imperative goal for mankind – a goal to be pursued together with, and in harmony with, the established and fundamental goals of peace and of worldwide economic and social development.

7. To achieve this environmental goal will demand the acceptance of responsibility by citizens and communities and by enterprises and institutions at every level, all sharing equitably in common efforts. Individuals in all walks of life as well as organizations in many fields, by their values and the sum of their actions, will shape the world environment of the future.

The twenty-six principles included in the Stockholm Declaration (UNEP (1972a, 1972b)) are listed below:

Principle 1: Man has the fundamental right to freedom, equality and adequate conditions of life, in an environment of a quality that permits a life of dignity and well-being, and he bears a solemn responsibility to protect and improve the environment for present and future generations. In this respect, policies promoting or perpetuating apartheid, racial segregation, discrimination, colonial and other forms of oppression and foreign domination stand condemned and must be eliminated.

Principle 2: The natural resources of the earth, including the air, water, land, flora and fauna and especially representative samples of natural ecosystems, must be safeguarded for the benefit of present and future generations through careful planning or management, as appropriate.

Principle 3: The capacity of the earth to produce vital renewable resources must be maintained and, wherever practicable, restored or improved.

Principle 4: Man has a special responsibility to safeguard and wisely manage the heritage of wildlife and its habitat, which are now gravely imperiled by a combination of adverse factors. Nature conservation, including wildlife, must therefore receive importance in planning for economic development.

Principle 5: The non-renewable resources of the earth must be employed in such a way as to guard against the danger of their future exhaustion and to ensure that benefits from such employment are shared by all mankind.

Principle 6: The discharge of toxic substances or of other substances and the release of heat, in such quantities or concentrations as to exceed the capacity of the environment to render them harmless, must be halted in order to ensure that serious or irreversible damage is not inflicted upon ecosystems. The just struggle of the peoples of all countries against pollution should be supported.

Principle 7: States shall take all possible steps to prevent pollution of the seas by substances that are liable to create hazards to human health, to harm living resources and marine life, to damage amenities or to interfere with other legitimate uses of the sea.

Principle 8: Economic and social development is essential for ensuring a favorable living and working environment for man and for creating conditions on earth that are necessary for the improvement of the quality of life.

Principle 9: Environmental deficiencies generated by the conditions of under-development and natural disasters pose grave problems and can best be remedied by accelerated development through the transfer of substantial quantities of financial and technological assistance as a supplement to the domestic effort of the developing countries and such timely assistance as may be required.

Principle 10: For the developing countries, stability of prices and adequate earnings for primary commodities and raw materials are essential to environmental management, since economic factors as well as ecological processes must be taken into account.

Principle 11: The environmental policies of all States should enhance and not adversely affect the present or future development potential of developing countries, nor should they hamper the attainment of better living conditions for all, and appropriate steps should be taken by States and international organizations with a view to reaching agreement on meeting the possible national and international economic consequences resulting from the application of environmental measures.

Principle 12: Resources should be made available to preserve and improve the environment, taking into account the circumstances and particular requirements of developing countries and any costs which may emanate from their incorporating environmental safeguards into their development planning and the need for making available to them, upon their request, additional international technical and financial assistance for this purpose.

Principle 13: In order to achieve a more rational management of resources and thus to improve the environment, States should adopt an integrated and coordinated approach to their development planning so as to ensure that development is compatible with the need to protect and improve environment for the benefit of their population.

Principle 14: Rational planning constitutes an essential tool for reconciling any conflict between the needs of development and the need to protect and improve the environment.

Principle 15: Planning must be applied to human settlements and urbanization with a view to avoiding adverse effects on the environment and obtaining maximum social, economic and environmental benefits for all. In this respect projects which are designed for colonialist and racist domination must be abandoned.

Principle 16: Demographic policies which are without prejudice to basic human rights and which are deemed appropriate by the Governments concerned should be applied in those regions where the rate of population growth or excessive population concentrations are likely to have adverse effects on the environment of the human environment and impede development.

Principle 17: Appropriate national institutions must be entrusted with the task of planning, managing or controlling the environmental resources of States with a view to enhancing environmental quality.

Principle 18: Science and technology, as part of their contribution to economic and social development, must be applied to the identification, avoidance and control of environmental risks and the solution of environmental problems and for the common good of mankind.

Principle 19: Education in environmental matters, for the younger generation as well as adults, giving due consideration to the underprivileged, is essential in order to broaden the basis for an enlightened opinion and responsible conduct by individuals, enterprises and communities in protecting and improving the environment in its full human dimension. It is also essential that mass media of communications avoid contributing to the deterioration of the environment, but, on the contrary, disseminates information of an educational nature on the need to protect and improve the environment in order to enable all to develop in every respect.

Principle 20: Scientific research and development in the context of environmental problems, both national and multinational, must be promoted in all countries, especially the developing countries. In this connection, the free flow of up-to-date scientific information and transfer of experience must be supported and assisted, to facilitate the solution of environmental problems; environmental technologies should be made available to developing countries on terms which would encourage their wide dissemination without constituting an economic burden on the developing countries.

Principle 21: States have, in accordance with the Charter of the United Nations and the principles of international law, the sovereign right to exploit their own resources pursuant to their own environmental policies, and the responsibility to ensure that activities within their jurisdiction or control do not cause damage to the environment of other States or of areas beyond the limits of national jurisdiction.

Principle 22: States shall cooperate to develop further the international law regarding liability and compensation for the victims of pollution and other environmental damage caused by activities within the jurisdiction or control of such States to areas beyond their jurisdiction.

Principle 23: Without prejudice to such criteria as may be agreed upon by the international community, or to standards which will have to be determined nationally, it will be essential in all cases to consider the systems of values prevailing in each country, and the extent of the applicability of standards which are valid for the most advanced countries but which may be inappropriate and of unwarranted social cost for the developing countries.

Principle 24: International matters concerning the protection and improvement of the environment should be handled in a cooperative spirit by all countries, big and small, on an equal footing. Cooperation through multilateral or bilateral arrangements or other appropriate means is essential to effectively control, prevent, reduce and eliminate adverse environmental effects resulting from activities conducted in all spheres, in such a way that due account is taken of the sovereignty and interests of all States.

Principle 25: States shall ensure that international organizations play a coordinated, efficient and dynamic role for the protection and improvement of the environment.

Principle 26: Man and his environment must be spared the effects of nuclear weapons and all other means of mass destruction. States must strive to reach prompt agreement, in the relevant international organs, on the elimination and complete destruction of such weapons.

8.3 THE WORLD CONSERVATION STRATEGY (WCS) – 1980

One of the products of the United Nations Conference on the Human Environment (UNCHE) was the establishment of the United Nations Environment Program (UNEP). In 1980, the document 'World Conservation Strategy: Living Resource Conservation for Sustainable Development' was produced. It was commissioned by UNEP, prepared by the International Union for Conservation of Nature and Natural Resources (IUCN), and jointly financed by UNEP and the World Wildlife Fund (WWF).

Although IUCN spear-headed the overall preparation of the 'Strategy,' many governments, non-government organizations and individuals from both developed and developing countries participated in its development. The draft reports were circulated for comment to a large number of specialists on ecology, threatened species, protected areas, environmental planning, environmental policy, law and administration, and environmental education (Munro, 1980).

In 1980, the 'World Conservation Strategy' report was published in pack format intended for decision-makers and, as such, had a very limited distribution. An unofficial version of the 'Strategy' is the publication 'How to Save the World: Strategy for World Conservation' prepared by Robert Allen (Allen, 1980). It was based on the material contained in the report but oriented toward the general reader. The material contained in this section is extracted directly from that publication (Allen, 1980).

The intention of the 'The World Conservation Strategy' was to stimulate a more focused approach to living resource conservation and to provide policy guidance on how this can be carried out. The 'Strategy,' according to Scott (1980), is the first time that:(1) it has been clearly shown how conservation can contribute to the development objectives of governments, industry, commerce, organized labor, and the professions, and (2) development has been suggested as a major means of achieving conservation instead of being viewed as an obstruction to it.

The present authors consider the following points from the book to be especially relevant to mining readers (Allen, 1980).

1. The biosphere is like a self-regenerating cake, and conservation is the conduct of our affairs so that we can have our cake and eat it too. As long as certain bits of cake are not consumed and consumption of the rest is kept within certain limits, the cake will renew itself and provide for continuing consumption. For people to gain a decent livelihood from the earth without undermining its capacity to go on supporting them, they must conserve the biosphere. This means doing three things:

1. Maintaining essential ecological processes and life-support systems
2. Preserving genetic diversity
3. Utilizing species and ecosystems sustainably.

2. Although environmental modification is natural and a necessary part of development, this does not mean that all modification leads to development (nor that preservation impedes development). While it is inevitable that most of the planet will be modified by people and that much of it will be transformed, it is not at all inevitable that such alterations will achieve the social and economic objectives of development. *Unless it is based on conservation, much development will continue to have unacceptably harmful side effects, provide reduced benefits or even fail altogether; and it will become impossible to meet the needs of today without foreclosing the achievement of tomorrow* (emphasis added by current authors).

3. A world strategy for the conservation of the earth's living resources is needed for three reasons:

a. The need for conservation is so pressing that it should be in the forefront of human endeavor. Yet for most people and their governments, conservation is an obscure peripheral activity perpetrated by birdwatchers. One consequence of this view is that development, which should be the main means of solving human problems, is so little affected by conservation that too often it adds to human problems by destroying or degrading living resources essential for human welfare. A world strategy is needed to focus the attention of the world on conservation.

b. National and international organizations concerned with conservation, whether governmental or non-governmental, are ill-organized and fragmented, split up among different interests such as agriculture, forestry, fisheries and wildlife. As a result, there is duplication of effort, gaps in coverage, competition for money and influence, and conflict. A world strategy is needed to promote that effort and define the areas where cooperation is most needed.

c. The action required to cure the most serious current conservation problems and to prevent still worse ones takes time: time for planning, education, training, better organization and research. When such action is undertaken, it takes time for the biosphere to respond: reforestation, the restoration of degraded land, the recovery of depleted fisheries, and so on are not instantaneous processes. Because time is running out a world strategy is necessary: (1) to determine the priorities, (2) to indicate the main obstacles to achieving them and (3) to propose ways of overcoming the obstacles.

4. For development practitioners, the 'Strategy' proposes ways of improving the prospects of *sustainable development – development that is likely to achieve lasting satisfaction of human needs and improvement of the quality of human life – by integrating conservation into the development process* (emphasis added by current authors). It also attempts to identify those areas where the interests of conservation and of development are most likely to coincide, and therefore where a closer partnership between the two processes would be particularly advantageous.

Although the focus of the 'World Conservation Strategy' is on living things, the main obstacles cited have relevance to mining. The obstacles are (Allen, 1980):

1. The belief that the conservation of living resources is a specialized activity rather than a process that cuts across and must be considered by all sectors of activity.
2. The consequent of failure to integrate conservation with development.
3. A development process that is generally inflexible and needlessly destructive, because of inadequate environmental planning and a lack of rational allocation of land and water uses.
4. The lack of capacity, because of inadequate legislation, to conserve; poor organization (notably government agencies with insufficient mandates and a lack of coordination); lack of trained personnel; and a lack of basic information on priorities, on the productive and regenerative capacities of the living resources concerned, and on trade-offs between one management option and another.
5. The lack of support for conservation, because of a lack of awareness (other than at the most superficial level) of the need for conservation and of the responsibility to conserve amongst those who use or have an impact on living resources, including in many cases, governments.
6. Failure to deliver conservation–based development where it is most needed, notably the rural areas of developing countries.

The publication 'How to Save the World: Strategy for World Conservation' is still available today and its reading is strongly encouraged. The 'World Conservation Strategy' is credited as being the source of the phrase 'sustainable development' which is in such wide use today.

In this document it is very correctly applied since it concerns living, renewable objects and things.

8.4 WORLD COMMISSION ON ENVIRONMENT AND DEVELOPMENT (1987)

In 1983, the 'World Commission on Environment and Development' was established under the auspices of the United Nations Environment Program (UNEP). The Commission's directive was to re-examine the critical environment and development problems on the planet and to formulate realistic proposals to solve them. A global agenda for change was to be developed working within the principle of Environmentally Sustainable Development (Alcor, 2005).

Some of the specific goals were (Brundtland, 1987):

(1) to propose long-term environmental strategies for achieving sustainable development by the year 2000 and beyond;

(2) to recommend ways in which concern for the environment may be translated into greater co-operation among developing countries and between countries at different stages of economic and social development and lead to the achievement of common and mutually supportive objectives that take account of the inter-relationships between people, resources, environment, and development;

(3) to consider ways and means by which the international community can deal more effectively with environmental concerns;

(4) to help define shared perceptions of long-term environmental issues and the appropriate efforts needed to deal successfully with the problems of protecting and enhancing the environment, a long-term agenda for action during the coming decades, and aspirational goals for the world community.

Mrs. Gro Harlem Brundtland, then Prime Minister of Norway, chaired the commission.

The Commission's report 'Our Common Future: The World Commission on Environment and Development' (often referred to as simply 'Our Common Future' or 'The Brundtland Report') was delivered in 1987. In the report, population and human resources, food security, species and ecosystems, energy, and the 'urban challenge' of humans in their built environment are examined. Recommendations are made for institutional and legal changes in order to confront common global problems. These include the development and expansion of international institutions for cooperation and legal mechanisms to confront common concerns. It called for increased cooperation with industry (Alcor, 2005).

As part of their work, the Commission (Our Common Future, 1987) defined sustainable development as:

> 'development that meets the needs of the present without compromising the ability of future generations to meet their own needs.'

The words are very similar to the definition provided in 1980 as part of the 'World Conservation Strategy.' However, its meaning has been considerably broadened. Sustainable development is considered to be a *process of change* (emphasis added by current authors) in which

> 'the exploitation of resources, the direction of investments, the orientation of technological development, and institutional change are made consistent with future as well as present needs (Our Common Future, 1987).'

In Chapter 2, which is entitled 'Towards Sustainable Development,' there are some additional statements regarding what they mean by sustainable development, both in general, and also with respect to non-renewable resources. A few selected by the present authors are presented below (Our Common Future, 1987):

1. There are two key concepts in sustainable development:
   - the concept of 'needs', in particular the essential needs of the world's poor, to which overriding priority should be given.
   - the idea of limitations imposed by the state of technology and social organization on the environment's ability to meet present and future needs.

2. Sustainable development requires that societies meet human needs both by increasing productive potential and by ensuring equitable opportunities for all.

3. Settled agriculture, the diversion of water courses, the extraction of minerals, the emission of heat and noxious gases into the atmosphere, commercial forests, and genetic manipulation are all examples of human intervention in natural systems during the course of development. Until recently, such interventions were small in scale and their impact limited. Today's interventions are more drastic in scale and impact, and more threatening to life-support systems both locally and globally. This need not happen. At a minimum, sustainable development must not endanger the natural systems that support life on Earth: the atmosphere, the waters, the soils, and the living beings.

4. In general, renewable resources like forests and fish stocks need not be depleted provided the rate of use is within the limits of regeneration and natural growth.

5. For non-renewable resources, like fossil fuels and minerals, their use reduces the stock available for future generations. But this does not mean that such resources should not be used. In general, the rate of depletion should take into account the criticality of that resource, the availability of technologies for minimizing depletion, and the likelihood of substitutes being available. Thus land should not be degraded beyond reasonable recovery.

6. With minerals and fossil fuels, the rate of depletion and the emphasis on recycling and economy of use should be calibrated to ensure that the resource does not run out before acceptable substitutes are available.

7. Sustainable development requires that the rate of depletion of non-renewable resources should foreclose as few future options as possible.

8. So-called free goods like air and water are also resources. Sustainable development requires that the adverse impacts on the quality of air, water and other natural elements are minimized so as to sustain the ecosystem's overall integrity.

9. In essence, sustainable development is a process of change in which the exploitation of resources, the direction of investments, the orientation of technological development, and institutional change are all in harmony and enhance both current and future potential to meet human needs and aspirations.

10. Many problems of resource depletion and environmental stress arise from disparities in economic and political power. An industry may get away with unacceptable levels of air and water pollution because the people who bear the brunt of it are poor and unable to complain

effectively. A forest may be destroyed by excessive felling because the people living there have no alternatives or because timber contractors generally have more influence than forest dwellers.

For mining, the above should be kept firmly in mind in addition to the simple and often quoted 'sustainable development' definition statement.

This section will close with a listing of what the authors of the report felt to be required in the pursuit for sustainable development (Our Common Future, 1987):

- a political system that secures effective citizen participation in decision making,
- an economic system that is able to generate surpluses and technical knowledge on a self-reliant and sustained basis,
- a social system that provides for solutions for the tensions arising from disharmonious development,
- a production system that respects the obligation to preserve the ecological base for development,
- a technological system that can search continuously for new solutions,
- an international system that fosters sustainable patterns of trade and finance, and
- an administrative system that is flexible and has the capacity for self-correction.

As the authors of the document indicate, these requirements are more in the nature of goals that should underlie national and international action on development. They stress that what matters is the sincerity with which these goals are pursued and the effectiveness with which departures from them are corrected.

8.5 THE 'EARTH SUMMIT'

The United Nations Conference on Environment and Development (UNCED) was held in Rio de Janeiro, Brazil in 1992 (UNCED, 1992a). It is often referred to as the 'Earth Summit' and that name has been used here. It had the following goals: (1) building upon the United Nations Conference on the Human Environment held in Stockholm in 1972; (2) establishing a new and equitable global partnership through the creation of new levels of cooperation among States, key sectors of societies and people; and (3) working towards international agreements which respect the interests of all and protect the integrity of the global environmental and developmental system (UNCED, 1992b).

Several important documents were produced, two of which, 'The Rio Declaration' (UNCED, 1992b) and 'Agenda 21' (UNCED, 1992c) will be reproduced in this section.

8.5.1 *The Rio Declaration*

Recognizing the integral and interdependent nature of the Earth, our home, the following principles were proclaimed (UNCED, 1992b):

Principle 1: Human beings are at the centre of concerns for sustainable development. They are entitled to a healthy and productive life in harmony with nature.

Principle 2: States have, in accordance with the Charter of the United Nations and the principles of international law, the sovereign right to exploit their own resources pursuant to their own environmental and developmental policies, and the responsibility to ensure that activities within their jurisdiction or control do not cause damage to the environment of other States or of areas beyond the limits of national jurisdiction.

Principle 3: The right to development must be fulfilled so as to equitably meet developmental and environmental needs of present and future generations.

Principle 4: In order to achieve sustainable development, environmental protection shall constitute an integral part of the development process and cannot be considered in isolation from it.

Principle 5: All States and all people shall cooperate in the essential task of eradicating poverty as an indispensable requirement for sustainable development, in order to decrease the disparities in standards of living and better meet the needs of the majority of the people of the world.

Principle 6: The special situation and needs of developing countries, particularly the least developed and those most environmentally vulnerable, shall be given special priority. International actions in the field of environment and development should also address the interests and needs of all countries.

Principle 7: States shall cooperate in a spirit of global partnership to conserve, protect and restore the health and integrity of the Earth's ecosystem. In view of the different contributions to global environmental degradation, States have common but differentiated responsibilities. The developed countries acknowledge the responsibility that they bear in the international pursuit of sustainable development in view of the pressures their societies place on the global environment and of the technologies and financial resources they command.

Principle 8: To achieve sustainable development and a higher quality of life for all people, States should reduce and eliminate unsustainable patterns of production and consumption and promote appropriate demographic policies.

Principle 9: States should cooperate to strengthen endogenous capacity-building for sustainable development by improving scientific understanding through exchanges of scientific and technological knowledge, and by enhancing the development, adaptation, diffusion and transfer of technologies, including new and innovative technologies.

Principle 10: Environmental issues are best handled with the participation of all concerned citizens, at the relevant level. At the national level, each individual shall have appropriate access to information concerning the environment that is held by public authorities, including information on hazardous materials and activities in their communities, and the opportunity to participate in decision-making processes. States shall facilitate and encourage public awareness and participation by making information widely available. Effective access to judicial and administrative proceedings, including redress and remedy, shall be provided.

Principle 11: States shall enact effective environmental legislation. Environmental standards, management objectives and priorities should reflect the environmental and developmental context to which they apply. Standards applied by some countries may be inappropriate and of unwarranted economic and social cost to other countries, in particular developing countries.

Principle 12: States should cooperate to promote a supportive and open international economic system that would lead to economic growth and sustainable development in all countries, to better address the problems of environmental degradation. Trade policy measures for environmental purposes should not constitute a means of arbitrary or unjustifiable discrimination or a disguised restriction on international trade. Unilateral actions to deal with environmental challenges outside the jurisdiction of the importing country should be avoided. Environmental measures addressing trans-boundary or global environmental problems should, as far as possible, be based on an international consensus.

Principle 13: States shall develop national law regarding liability and compensation for the victims of pollution and other environmental damage. States shall also cooperate in an expeditious and more determined manner to develop further international law regarding liability and compensation for adverse effects of environmental damage caused by activities within their jurisdiction or control to areas beyond their jurisdiction.

Principle 14: States should effectively cooperate to discourage or prevent the relocation and transfer to other States of any activities and substances that cause severe environmental degradation or are found to be harmful to human health.

Principle 15: In order to protect the environment, the precautionary approach shall be widely applied by States according to their capabilities. Where there are threats of serious or irreversible damage, lack of full scientific certainty shall not be used as a reason for postponing cost-effective measures to prevent environmental degradation.

Principle 16: National authorities should endeavor to promote the internalization of environmental costs and the use of economic instruments, taking into account the approach that the polluter should, in principle, bear the cost of pollution, with due regard to the public interest and without distorting international trade and investment.

Principle 17: Environmental impact assessment, as a national instrument, shall be undertaken for proposed activities that are likely to have a significant adverse impact on the environment and are subject to a decision of a competent national authority.

Principle 18: States shall immediately notify other States of any natural disasters or other emergencies that are likely to produce sudden harmful effects on the environment of those States. Every effort shall be made by the international community to help States so afflicted.

Principle 19: States shall provide prior and timely notification and relevant information to potentially affected States on activities that may have a significant adverse trans-boundary environmental effect and shall consult with those States at an early stage and in good faith.

Principle 20: Women have a vital role in environmental management and development. Their full participation is therefore essential to achieve sustainable development.

Principle 21: The creativity, ideals and courage of the youth of the world should be mobilized to forge a global partnership in order to achieve sustainable development and ensure a better future for all.

Principle 22: Indigenous people and their communities and other local communities have a vital role in environmental management and development because of their knowledge and traditional practices. States should recognize and duly support their identity, culture and interests and enable their effective participation in the achievement of sustainable development.

Principle 23: The environment and natural resources of people under oppression, domination and occupation shall be protected.

Principle 24: Warfare is inherently destructive of sustainable development. States shall therefore respect international law providing protection for the environment in times of armed conflict and cooperate in its further development, as necessary.

Principle 25: Peace, development and environmental protection are interdependent and indivisible.

Principle 26: States shall resolve all their environmental disputes peacefully and by appropriate means in accordance with the Charter of the United Nations.

Principle 27: States and people shall cooperate in good faith and in a spirit of partnership in the fulfillment of the principles embodied in this Declaration and in the further development of international law in the field of sustainable development.

8.5.2 *Agenda 21*

The climax of the Earth Summit was the adoption of Agenda 21. Although weakened by compromise and negotiation during the generation process, it offered a wide-ranging blueprint for action to achieve sustainable development worldwide (UNCED, 1992a). The contents are indicated below (UNCED, 1992c).

Chapter 1. Preamble

SECTION I. SOCIAL AND ECONOMIC DIMENSIONS

Chapter 2. International cooperation to accelerate sustainable development in developing countries and related domestic policies
Chapter 3. Combating poverty
Chapter 4. Changing consumption patterns
Chapter 5. Demographic dynamics and sustainability
Chapter 6. Protecting and promoting human health conditions
Chapter 7. Promoting sustainable human settlement development
Chapter 8. Integrating environment and development in decision-making

SECTION II. CONSERVATION AND MANAGEMENT OF RESOURCES FOR DEVELOPMENT

Chapter 9. Protection of the atmosphere
Chapter 10. Integrated approach to the planning and management of land resources

Chapter 11. Combating deforestation
Chapter 12. Managing fragile ecosystems; combating desertification and drought
Chapter 13. Managing fragile ecosystems: sustainable mountain development
Chapter 14. Promoting sustainable agriculture and rural development
Chapter 15. Conservation of biological diversity
Chapter 16. Environmentally sound management of biotechnology
Chapter 17. Protection of the oceans, all kinds of seas, including enclosed and semi-enclosed seas, and coastal areas and the protection, rational use and development of their living resources
Chapter 18. Protection of the quality and supply of freshwater resources; application of integrated approaches to the development, management and use of water resources
Chapter 19. Environmentally sound management of toxic chemicals, including prevention of illegal international traffic in toxic and dangerous products
Chapter 20. Environmentally sound management of hazardous wastes, in hazardous wastes
Chapter 21. Environmentally sound management of solid wastes and sewage-related issues
Chapter 22. Safe and environmentally sound management of radioactive wastes

SECTION III. STRENGTHENING THE ROLE OF MAJOR GROUPS

Chapter 23. Preamble
Chapter 24. Global action for women towards sustainable and equitable development
Chapter 25. Children and youth in sustainable development
Chapter 26. Recognizing and strengthening the role of indigenous and their communities
Chapter 27. Strengthening the role of non-governmental organizations; partners for sustainable development
Chapter 28. Local authorities' initiatives in support of Agenda 21
Chapter 29. Strengthening the role of workers and their trade unions
Chapter 30. Strengthening the role of business and industry
Chapter 31. Scientific and technological community
Chapter 32. Strengthening the role of farmers

SECTION IV. MEANS OF IMPLEMENTATION

Chapter 33. Financial resources and mechanisms
Chapter 34. Transfer of environmentally sound technology, cooperation and capacity-building
Chapter 35. Science for sustainable development
Chapter 36. Promoting education, public awareness and training
Chapter 37. National mechanisms and international cooperation for capacity-building in developing countries

Chapter 38. International institutional arrangements
Chapter 39. International legal instruments and mechanisms
Chapter 40. Information for decision-making

8.6 WORLD SUMMIT ON SUSTAINABLE DEVELOPMENT (WSSD)

In 2002, the World Summit on Sustainable Development (WSSD) was held in Johannesburg, South Africa to review the progress made on Agenda 21 during the intervening ten-year period (WSSD, 2002a). Specific attention was placed on reviewing (1) the obstacles encountered, (2) the lessons learned during the implementation process, and (3) new factors which have emerged (WSSD, 2002b). The major output of the Summit was contained in two documents, 'Plan of Implementation for the World Summit on Sustainable Development' (WSSD, 2002c) and the 'Political Declaration' (WSSD, 2002d).

The 'Plan of Implementation' contains several direct references to the mining sector in addition to a wide range of proposals that will impact upon the activities of the sector.

The contents of the 'Plan of Implementation' document prepared at the World Summit on Sustainable Development are listed below (WSSD, 2002c).

I. Introduction
II. Poverty eradication
III. Changing unsustainable patterns of consumption and production
IV. Protecting and managing the natural resource base of economic and social development
V. Sustainable development in a globalizing world
VI. Health and sustainable development
VII. Sustainable development of small-island developing States
VIII. Sustainable development for Africa
IX. Other regional initiatives
    A. Sustainable development in Latin America and the Caribbean
    B. Sustainable development in Asia and the Pacific
    C. Sustainable development in the West Asia region
    D. Sustainable development in the Economic Commission for Europe region
X. Means of implementation
XI. Institutional framework for sustainable development
    A. Objectives
    B. Strengthening the institutional framework for sustainable development at the international level
    C. Role of the General Assembly
    D. Role of the Economic and Social Council
    E. Role and function of the Commission on Sustainable Development
    F. Role of international institutions
    G. Strengthening institutional arrangements for sustainable development at the regional level
    H. Strengthening institutional frameworks for sustainable development at the national level
    I. Participation of major groups

Paragraph 46 of this document (Section X – Means of implementation) which deals directly with mining, minerals and metals, is provided below (WSSD, 2002c):

Mining, minerals and metals are important to the economic and social development of many countries. Minerals are essential for modern living. Financing the contribution of mining, minerals and metals to sustainable development includes actions at all levels to:

a) Support efforts to address the environmental, economic, health and social impacts and benefits of mining, minerals and metals throughout their life cycle including workers health and safety, and use a range of partnerships, furthering existing activities at the national and international levels among interested Governments, intergovernmental organizations, mining companies and workers and other stakeholders to promote transparency and accountability for sustainable development;

b) Enhance the participation of stakeholders, including local and indigenous communities and women, to play an active role in minerals, metals and mining development throughout the life cycles of mining operations, including after closure for rehabilitation purposes in accordance with national regulations and taking into account significant trans-boundary impacts.

c) Foster sustainable mining practices through the provision of financial, technical and capacity-building support to developing countries and countries with economies in transition for the mining and processing of minerals, including small-scale mining, and where possible and appropriate, improve value-added processing, upgrade scientific and technological information and reclaim and rehabilitate degraded sites.

8.7 MINING INDUSTRY AND MINING INDUSTRY-RELATED INITIATIVES

8.7.1 *Introduction*

To this point in this chapter, the focus has been on some of the major sustainable development initiatives organized under the auspices of the United Nations. This section will deal with some of the sustainable development oriented activities involving various sectors of the world mining industry.

8.7.2 *The Global Mining Initiative (GMI)*

In the fall of 1998, discussions between the chairmen and chief executives of several of the world's largest mining companies were held concerning the social and environmental challenges facing the mining and metals industries. Their common concerns were: (1) the signs of increasing resistance to new mining projects from a wide range of stakeholders worldwide; (2) growing concerns over perceptions of health and environmental threats resulting from the use of metals; and (3) the effectiveness of industry associations. These discussions led to the birth of the Global Mining Initiative (GMI). This Initiative was aimed at ensuring that an industry essential to the well-being of a changing world was responsive to global needs and challenges (GMI (1998a, 1998b)).

The participating mining companies hoped to learn how they could manage these issues more effectively in order to maximize the contribution made to the wider transition to sustainable development. They hoped that non-industry participants in the initiative, such as governments, international organizations and other representatives of civil society, would share in the effort to define legitimate mutual expectations and to clarify where the boundaries lay for action by different actors (GMI, 1998c).

The GMI was concerned with the full range of issues in the mining, minerals and metals cycles, including (GMI, 1998c):

- access to land and resources
- exploration
- project development and secondary development impacts
- governance of mining projects, their place in social and economic development and issues of capacity building
- rent capture and distribution
- mining operations
- stewardship of resources such as water and biodiversity
- energy use
- management of waste
- social and environmental aspects of mine closure
- primary and subsequent stages of processing
- the trade in materials produced by mining
- how those materials are used – their consumption, recycling and disposal

It was noted that since the 'Earth Summit' at Rio de Janeiro in 1992, the relationship between the mining industry and the environmental movement could be characterized as being more positive. Both have realized that co-operation and open discussion are essential for real progress towards sustainable development. The mining industry has accepted that, alone, it cannot 'rewrite its reputation' and needs to work with a wide range of non-industry participants, such as governments, international organizations like the World Bank, and the NGO community, in order to achieve a common understanding of the challenges and priorities (GMI, 1998b).

The GMI planned three major activities the objective of which was to reach a clearer definition and understanding of the positive part the mining and minerals industry can play in making the transition to sustainable patterns of economic development. These activities were (GMI, 1998b):

1. The Mining, Minerals and Sustainable Development (MMSD) project. This activity, sponsored by 30 mining companies, had the aim of 'identifying how mining and minerals can best contribute to the global transition to sustainable development'. The project was commissioned by the mining working group of the World Business Council for Sustainable Development, managed by a committee with a majority of non-industry representatives, and coordinated by Richard Sandbrook, co-founder of Friends of the Earth. The work was undertaken with the respected think-tank, the International Institute for Environment and Development (GMI, 1998b).

2. A careful re-appraisal of all the major bodies representing mining interests around the world. The aim was to refine association activities to give a more concerted approach to sustainable development issues. In this way, the activities can be focused via a global structure which would serve as a pro-active advocate for the mineral industries. The result was the creation of a strong global organization, the International Council on Mining and Metals (ICMM), to represent and lead the industry in meeting the challenges of sustainable development (GMI, 1998b).

3. Organization of the 'Global Mining Initiative Conference.' This conference which was held in Toronto in May 2002 (just prior to the World Summit on Sustainable Development held in Johannesburg, September 2002), provided a platform for industry leaders to discuss key issues and recommendations of the MMSD report with leaders of government, international organizations and NGOs (GMI, 1998b).

Following publication of the final report of the Mining, Minerals and Sustainable Development (MMSD) project and the close of the Global Mining Initiative Conference, the Global Mining Initiative had served its function and ceased to exist as an entity.

8.7.3 *International Council on Mining and Metals (ICMM)*

As indicated in section 8.7.2, the International Council on Mining and Metals (ICMM) was created under the auspices of the Global Mining Initiative. It has as its vision 'A viable mining, minerals and metals industry that is widely recognized as essential for modern living and a key contributor to sustainable development.' The mission statement of the ICMM may be summarized as: (1) ensuring the continued access to land, capital and markets by the mining, minerals and metals industry (as well as building trust and respect) by demonstrating its ability to contribute successfully to sustainable development; (2) offering strategic industry leadership towards achieving continuous improvements in sustainable development performance; and (3) providing a common platform for the industry to share challenges and responsibilities as well as to engage with key constituencies on issues of common concern at the international level, based on science and principles of sustainable development. This mission, they believe, is best achieved by acting collectively (ICCM, 2002a).

For the mission to be accomplished it must be accompanied by an action plan with clearly stated goals and timely deliverables. The ICMM goals are (ICCM, 2002a) to:

1. Offer strategic leadership to achieve improved sustainable development performance in the mining, minerals and metals industry.

2. Represent the views and interests of its members and serve as a principal point of engagement with the industry's key constituencies in the international arena.

3. Promote science-based regulations and material-choice decisions that encourage market access and the safe production, use, reuse, and recycling of metals and minerals.

4. Identify and advocate the use of good practices to address sustainable development issues within the industry.

The ICMM consists of a Council, the Executive Working Group, and various Committees and Task Forces (ICCM, 2002b). The Council which is made up of the CEOs of all ICMM member companies and associations is ICMM'S principal governing body. The Executive Working Group comprises nominated representatives from each of the corporate and association members. This group facilitates input from members on cross-cutting issues and ensures the effective implementation of the ICMM work program. Task Force chairs are appointed from this group. Committees and Task Forces are appointed by the Council to develop policy and pursue programs as required. The current task forces (ICCM, 2002b) are:

– Integrated Materials Management
– Community and Social Development
– Health and Safety
– Environmental Stewardship

The ICMM Principles are as follows (ICCM, 2002c):

- Implement and maintain ethical business practices and sound systems of corporate governance
- Integrate sustainable development considerations within the corporate decision-making process
- Uphold fundamental human rights and respect cultures, customs and values in dealings with employees and others who are affected by our activities
- Implement risk management strategies based on valid data and sound science
- Seek continual improvement of our health and safety performance
- Seek continual improvement of our environmental performance
- Contribute to conservation of biodiversity and integrated approaches to land use planning
- Facilitate and encourage product design, use, re-use, recycling and disposal of our products
- Contribute to the social, economic and institutional development of the communities in which we operate
- Implement effective and transparent engagement, communication and independently verified reporting arrangements with our stakeholders.

At the conclusion of the Global Mining Initiative Conference held May 12–15, 2002, the ICMM Toronto Declaration was issued (ICMM, 2002d). Based on a shared desire to enhance the contribution that mining and metals can make to social and economic development, the following realizations were put forth (ICMM, 2002d):

- that successful mining and metals processing operations require the support of the communities in which they operate;
- that respect for these communities and a serious engagement with them is required to ensure that mining and metals processing are seen as beneficial for the community and the company;
- that successful companies will respect fundamental human rights, including workplace rights and the need for a healthy and safe workplace: and
- that successful companies will accept their environmental stewardship responsibilities for their facility locations.

To give expression to these values will take dedicated and focused action on our part. We cannot achieve this alone. Progress towards sustainable development will be the product of continuing engagement with government and civil society. This engagement, which will have to occur at all levels of our industry, will at times involve trade-offs and difficult choices (ICMM, 2002d).

ICMM recognizes that (ICMM, 2002d):

- The MMSD Report and the process on which it was based, including the regional programs, have elevated and informed the debate leading to a way forward for the sector.
- Decisive and principled leadership is required at this critical time.
- Accountability, transparency and credible reporting is essential.
- Its Members, in satisfying their obligations to shareholders, must do business in a manner that merits the trust and respect of key constituencies, including the communities in which they operate.
- Constructive and value-adding engagement among constituencies at the local, national, and global levels is essential.
- Its Members must move beyond a regulatory-compliance-based mind set to effectively manage the complex trade-offs of economic, environmental, and social issues.

- The industry requires additional capacity to be effective in advancing sustainable development.
- The rates and responsibilities of the diverse parties comprising governments, civil society, and business are different and must be respected.
- Artisanal, small-scale mining and orphan site legacy issues are important and complex. However, they are beyond the capacity of ICMM to resolve. Governments and international agencies should assume the lead role in addressing them.

ICMM will (ICMM, 2002d):

- Expand the current ICMM Sustainable Development Charter to include appropriate areas recommended in the MMSD Report.
- Develop best-practice protocols that encourage third-party verification and public reporting.
- Engage in constructive dialogue with key constituencies.
- Assist Members in understanding the concepts and application of sustainable development.
- Together with the World Bank and others, seek to enhance effective community development management tools and systems.
- Promote the concept of integrated materials management throughout the minerals value chain wherever relevant.
- Promote sound science-based regulations and material-choice decisions that encourage market access and the safe use, reuse and recycling of metals and minerals.
- Create an emergency response regional register for the global mining, metals and minerals industry.
- In partnership with IUCN – The World Conservation Union and others, seek to resolve the questions associated with protected areas and mining.

8.7.4 *Mining, Minerals, and Sustainable Development (MMSD)*

The Mining, Minerals and Sustainable Development (MMSD) project was commissioned as part of the Global Mining Initiative. It was an independent two-year process of consultation and research aimed at understanding how to maximize the contribution of the mining and minerals sector to sustainable development at the global, national, regional and local levels. Managed by the International Institute for Environment and Development (IIED) in London, under contract to the World Business Council for Sustainable Development, the project began in April 2000. It was designed to produce concrete results in the form of a final report and a series of working papers as well as to create a dialogue process that could be carried forward into the future (MMSD, 2000).

The general objectives of MMSD (MMSD, 2000) were:

1. To assess global mining and minerals use in terms of the transition to sustainable development. This was to cover the current contribution – both positive and negative – to economic prosperity, human well-being, ecosystem health and accountable decision-making, as well as the track record of past practice.
2. To identify how the services provided by the minerals system could be delivered in accordance with sustainable development in the future.
3. To propose key elements of an action plan for improving the minerals system.
4. To build platforms of analysis and engagement for ongoing cooperation and networking among all stakeholders.

A listing of the contents of the final report 'Breaking New Ground' released in May 2002 is presented below (MMSD, 2002a):

- Executive Summary
- Table of Contents
- Titles and Copyright
- Foreword and Statements by Assurance Group and Sponsors Group
- Introduction

Part I: A Framework for Change
1. The Minerals Sector and Sustainable Development

Part II: Current Trends and Actors
2. Producing and Selling Minerals
3. A Profile of the Minerals Sector
4. The Need for and Availability of Minerals
5. Case Studies on Minerals

Part III: Challenges
6. Viability of the Minerals Industry
7. The Control, Use, and Management of Land
8. Minerals and Economic Development
9. Local Communities and Mines
10. Mining, Minerals and the Environment
11. An Integrated Approach to Using Minerals
12. Access to Information
13. Artisanal and Small-Scale Mining
14. Sector Governance: Roles, Responsibilities and Instruments for Change

Part IV: Responses and Recommendations
15. The Regional Perspectives
16. Agenda for Change

Appendices
Appendix 1. The MMSD Project
Appendix 2. MMSD Consultation Activities
Acronyms and Abbreviations
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Bibliography

As the authors of the report indicate (MMSD, 2002b), given the limited time and resources available, they could not even begin to address all of the issues that will ever be faced by the mining and minerals industries. The report does, however, provide a starting point for identifying different concerns and getting processes under way.

8.7.5 *The U.S. Government and federal land management*

The material included in this section has been extracted directly from the excellent publication 'Sustainable Development and Its Influence on Mining Operations On Federal Lands – A Conversation In Plain Language' prepared by representatives of the U.S. Bureau of Land Management and the U.S. Forest Service (Anderson, et al., 2002). These agencies administer 262 million acres and 194 million acres of federal lands, respectively, in the United States.

The remainder of this section has been taken directly from their publication.

Sustainable development is about ensuring human well-being while respecting ecosystem well-being and the earth's environmental limits and capacities. It encompasses environmental and social issues, as well as economic activity. These are interrelated and actions in any domain may, over time, impact all aspects of life in the region where we live.

A sustainable development perspective applied to resource management puts multiple use-including conservation, production, remediation, and land stewardship into a larger, integrated picture of resource management activities. Sustainable development gives us a checklist to work from, such as: What are the environmental and social impacts of an economic proposal? What are the economic and social implications of an environmental regulation?

The continuity of supply of resources obtainable through mining, and the sound management of these resources and the environment, are essential parts of sustainable development requiring a long-term view. We remain concerned not only about current results and impacts and the well being of the present generation, but also about cumulative impacts and the wellbeing of our children and grandchildren. This approach is not new to natural resource managers, but in some cases it is a welcome change for economic production and consumption to be managed with these broader and more long-term values in mind.

Simply put, sustainable development means thinking more broadly and longer-term about our national, corporate, and individual actions and how they relate to our environment and community It also means regularly checking our progress, as well as learning from experience and studying how we can better meet human needs. Perhaps more importantly for federal agencies, a sustainable development perspective in resource management gives us the opportunity to create better relationships with and among our stakeholders, including local, state, regional, tribal, corporate, and non-government communities of interest. Each of us can contribute knowledge, information, or resources to help us accomplish together what we cannot do alone.

The question arises as to why include minerals when they are a finite resource and not sustainable? It is true that individual mineral deposits are finite, but that does not mean minerals and metals have no place in sustainable development. Rather, sustainable development can provide the foundation for a policy framework that ensures minerals and metals are produced, used, reused, recycled and, if necessary, stored for the future (landfills) in a manner that respects the economic, social, and environmental needs of the local, national, and global communities. Within this framework, the benefits provided by minerals and mining are acknowledged, as is the reality that geology dictates the location of mineral deposits. Moreover, sustainable development makes good business sense because improving the efficiency of extracting and processing mineral resources creates both economic and environmental rewards.

The basic components of sustainable development (not in order of importance) are: social well-being, environmental health, and economic prosperity. Essentially, sustainable development requires that social, environmental, and economic issues be integrated in decision making. In all decisions, the long-term effects on resources and capital and the capacity for future creation of benefits should be considered. Decision making by natural resource managers should be broad, participatory, and also interdisciplinary.

Concern for economic and technological efficiency, for local environmental quality including planning for cleanup and reclamation at the closure of a mine, and concern for the social well-being of the local mine community and nearby population have long been mineral industry issues. Sustainable development provides a context within which to integrate these concerns.

Employees with the United States Departments of Interior and Agriculture have been working with their stakeholders to show how the social, environmental, and economic components of sustainable development could apply to mining operations. Following are examples of each.

Social: This component relates to community responsibilities. It is aimed at alerting companies, governments, and others to the need for enhancing the health of people and their communities, while maintaining profitable companies. Further, it raises the need for communities to understand and agree upon the distribution of cost, benefits, and risks of any proposed project or activity. It includes concepts such as:

- Respecting the cultures, customs, and values of individuals and groups whose livelihoods may be affected by exploration, mining, and processing
- Respecting the authority of national, regional, and tribal governments; taking into account their development objectives; contributing information related to mining and metal processing activities; and supporting the sharing of economic benefits generated by operations
- Recognizing local communities and other affected organizations and engaging with them in an open, honest, and effective process of consultation and communication from exploration through production to mine closure
- Assessing the social and cultural impacts of proposed activities
- Reducing to acceptable levels, as recommended by all stakeholders, the adverse social impacts on communities of activities related to exploration, extraction, and closure of mining and processing facilities
- Promoting health and safety both on and off the project site
- Developing one-on-one programs to support the wellbeing of employees' families in mining communities, such as activities and educational opportunities for spouses and children of mine employees

Environmental: This component relates to environmental stewardship. It is aimed at alerting companies, governments, and others to the need for enhancing environmental conditions over the long term. It includes concepts such as:

- Making environmental management a high priority
- Planning for mine closure beginning with exploration and mine approval
- Establishing environmental accountability in industry and government at the highest management and policymaking levels
- Adopting best practices to minimize environmental degradation and adapting them to local conditions as necessary
- Using energy and materials that conserve resources and avoid waste and expensive cleanup
- Conducting environmental impact assessments and collecting baseline data for flora and fauna, soil, and underground and surface waters
- Determining the capacity of the land for uses other than mining
- Minimizing noise and dust during operations
- Handling hazardous materials safely
- Minimizing pollution during operations

- Developing a mine waste management plan that includes tailings dam inspections, emergency checks, and hazard prevention
- Reclaiming the land to prevent erosion and planting native species targeting the same density and diversity of plants that were there before mining

Economic: This component relates to economic and financial actions, impacts, and policies. It is aimed at recognizing that the health of the economy has to be maintained as a principal means for achieving our quality of life. It includes concepts such as:

- Assessing the economic impacts of proposed and ongoing activities and developing management policies that maximize positive and minimize negative community and household impacts
- Working with local communities to develop strategies for sustaining their economies after mine closures and encouraging the establishment of other sustainable local and regional business activities looking for continuing improvements in design and efficiency that will help both profitability and competitiveness while reducing wastes released into the environment
- Investing to optimize long term returns to investment rather than immediate returns
- Investing in programs that improve the skills and thus productivity of the workforce with the goal of creating both economic and social benefits
- Encouraging suppliers to use energy efficient materials and technologies

In closing, the authors of the report indicate that these examples reflect current thinking about how sustainable development principles could apply to mining operations on federal land and do not constitute policy or regulation for the mining industry. The Forest Service and Bureau of Land Management have embraced sustainable development because the concept is complementary to and consistent with each agency's mission to provide for many uses of federal lands. This includes developing natural resources and working with stakeholders to achieve a sustainable future for our lands and for our communities.

8.7.6 *The position of the U.S. National Mining Association (NMA)*

The National Mining Association (NMA) includes more than 325 member companies involved in all aspects of the mining industry. In September 2002, the NMA adopted the Sustainable Development Pledge and the Sustainable Development Principles Statement reproduced below (NMA, 2002):

NMA Sustainable Development Pledge

The members of the National Mining Association pledge to conduct their activities in a manner that recognizes the needs of society and the needs for economic prosperity, national security and a healthy environment. Accordingly, we are committed to integrating social, environmental, and economic principles in our mining operations from exploration through development, operation, reclamation, closure and post closure activities, and in operations associated with preparing our products for further use.

NMA Sustainable Development Principles Statement

1. From an environmental standpoint this involves:
   - Recognizing and being responsive to possible environmental impacts of exploration activities;

- Developing approaches to mine planning and development that are responsive to possible environmental impacts through every stage of the mining cycle including closure and post closure activities;
- Planning in advance for the timely reclamation of sites in accordance with site specific criteria and recognizing community priorities, needs and interests as the mine approaches and reaches closure;
- Assisting in addressing legacy issues through existing mechanisms and laws and by working with appropriate government bodies to establish responsible, balanced and cost effective solutions;
- Being a leader in developing, establishing and implementing good environmental practices;
- Promoting the safe use, recycling and disposal of products through an understanding of their life cycles;
- Developing and promoting new technologies that continue to improve efficiencies and environmental performance in our mining and processing operations and in the use of our products;
- Recognizing that the potential for climate change is a special concern of global scope that requires significant attention and a responsible approach cutting across all three of the sustainable development pillars: environmental, social and economic;
- Encouraging climate policies that promote fuel diversity, development of technology and long term actions to address climate concerns in order to ensure that technological and financial resources are available to support the needs of the future;
- Supporting additional research to improve scientific understanding of the existence, causes and effects of climate change and to enhance our understanding of carbon absorbing sinks; advancements in technology to increase efficiencies in electrical generation and capture and sequester carbon dioxide; voluntary programs to improve efficiency and reduce greenhouse gas emission intensity; and, constructive participation in climate policy formulation on bath international and national levels.

2. From a social standpoint this involves:
   - Being committed to the safety, health, development and well being of our employees;
   - Respecting human rights;
   - Treating our employees with respect, promoting diversity, and providing competitive compensation programs consistent with performance and industry practice;
   - Being a progressive and constructive partner to advance the economic, educational, and social infrastructures of the communities in which we operate;
   - Respecting the cultures, customs and values of people wherever we operate, being responsive to- and respecting - community needs and priorities and encouraging and participating in an open and on-going dialogue with constituencies; and,
   - Adhering to the highest ethical business practices in all our operations and interacting with communities in a responsible manner.

3. From an economic standpoint this involves:
   - Creating wealth and products that contribute to economic prosperity;
   - Helping eliminate poverty and providing economic opportunities;
   - Contributing to national, regional and local economic well-being and security through creation of employment opportunities, wage payments, purchase of goods and materials and payment of fair and competitive taxes and usage fees; and,

- Allowing shareholders and investors to earn a fair and equitable return commensurate with the risk they take.

8.7.7 *The view of one mining company executive*

Yearley (2003) in his 2003 D.C. Jackling Award lecture entitled ‘Sustainable Development for the Global Mining and Metals Industry’ provided some insights as to how a large mining company views ‘sustainable development. Some of the key points from his lecture are summarized below (Yearley, 2003).

1. Fifteen years ago, few in the mining industry had heard the phrase sustainable development, let alone knew what it meant. Today, it is a different story. Industry leaders have embraced the concept. They have taken action to incorporate the inherent values and principles of sustainable development into the policies and modus operandi of their companies.

2. What does sustainable development mean for the mining and metal producers in operational terms? For years, this question was extensively debated both within and outside the industry. Today, there is broad consensus that sustainable development requires three things:
- Integrated approaches to decision making on a full, life-cycle basis that satisfy obligations to shareholders and that are balanced and supported by sound science and social, environmental and economic analysis within a framework of good governance.
- Consideration of the needs of current and future generations.
- Establishment of meaningful relationships with key constituencies based on mutual trust and a desire for mutually beneficial outcomes, including those inevitable situations that require informed trade-offs.

3. There is also a consensus view that sustainable development is a journey rather than a destination. That means the concept evolves in response to changing societal values, priorities and needs. To move forward against this backdrop, the mining and minerals sector needs to adapt, to create a new way of doing business.

4. Improving performance is in our interest. There is a clear business case. Low returns and a poor social-environmental record make mining a risky investment. When mining projects are restricted, delayed or halted, revenues and profits are lost. Companies succeed by minimizing cost and risks. Investors invest in sectors that can manage their business costs and risks. Sustainable development strategies can help minimize costs as follows:

- Lower labor costs – providing clean, safe working conditions can improve productivity, result in fewer union disputes and increase retention rates.
- Lower health costs – healthy communities and workers will again be more productive.
- Lower production costs – waste reduction, recycling and energy efficiency can significantly lower production costs.
- Lower regulatory burden – a trusted company can enjoy a smoother permitting and regulatory path.
- Lower closure costs – terminal liabilities can be more accurately predicted, managed and controlled.
- Lower cost borrowing – managing risk will improve relations with lenders.
- Lower insurance costs.

- Improved investor relations.
- Ultimately, a company that implements sustainable development policies can become the mining company of choice.

5. Managing environmental and social impact is not just a cost to be minimized, but an opportunity to enhance business value and ensure the long-term success of the company is fully realized. The evidence for mining companies during the last few years is compelling. It shows that mining companies that embrace environmental issues are more efficient, have superior management and represent a lower risk profile. This improved reputation ensures a company's social license to operate. The ultimate prize is access to land and markets combined with competitive advantage through improved performance. That is why the leading companies will act on sustainable development – it is in the interest of their shareholders. The message is that change is not always easy. There will be cost implications but the rewards are also potentially high.

6. There is a strong business case for change. The evidence for mining and metal companies to engage in this process has been compelling over the past few years. It shows that those companies that embrace environmental and social issues are more efficient, have superior management and better relationships with local communities, and represent a lower risk profile. This improved reputation will ensure a company's social license to operate and makes the company a better investment prospect.

7. Change is not always easy. There will be cost implications but the rewards are also potentially high. Ultimately, those companies that implement sustainable development policies will reap the benefits and become mining companies of choice from the standpoint of the investment and financial community, governments and other key constituencies.

8. Enhancing the contribution of mining, minerals and metals to sustainable development will require a concerted effort by governments, multilateral organizations, civil society groups as well as industry. Key priorities include:

- Ensure regulatory and material choice decisions based on precise and explicit criteria, cost-effective/timely risk assessments taking into account special characteristics of minerals and metals.
- Ensure openness and transparency and views of all stakeholders incorporated in decision-making processes likely to affect them.
- Establish market incentives to encourage product design, re-cyclability and economic collection and recovery of metals.
- Ensure benefits from mineral development are more fully realized by addressing the capacity-building needs of developing countries.

9. All constituents need to work together and have fair rules of engagement. The mining and metals industry is trying to reform and put in place the transparency and accountability systems that have been demanded. It is and will remain important that governments and civil society groups are held to the same high standards as are expected by companies.

10. A new course has been set for the industry. Sustainable development offers an excellent policy framework that will enable industry to demonstrate why minerals and metals should be considered materials of choice and to highlight the contributions that mining can make to economic and social development. There is a good business case for sustainable development, one that offers the opportunity – based on improved economic, environmental and

social performance – to build trust and improve our reputation, while enhancing shareholder value.

8.8 'RESPONSIBLE MINING' – THE WAY FORWARD IS GOOD ENGINEERING

8.8.1 *Introduction*

It is hoped that the reader now has a much better idea of the world in which new mining projects will be born. The successful ones will enter this world in the form of a three-legged stool whose legs, economics, environmental, and social/society, have the same length. This is as shown in Figure 8.1a. Clearly, if one or more of the legs is somewhat short with respect to the others (or missing entirely), the stool (project) will not be stable, or at least not perform in the way intended (Figures 8.1b and 8.1c). Some stools will be short (small) while others will be tall (large) but the stability (success) requirement of equal length legs remains the same.

In all of the foregoing text, little was indicated in the way of a path forward. One of the key elements, if not the key element, in the opinion of the authors, is an emphasis on good engineering. Unfortunately, good engineering has been absent or in very short supply in many past mining operations. Today, the production/operations group is still generally 'king' of the mine and represents the 'profit' center. Engineering is relegated to one of the many 'cost' centers and therefore labeled as an item to be minimized.

Logically, one could argue that production should be labeled a cost center since this is the home to all unit operations and their easily calculated costs. The true 'profit' center lies in engineering since this is where the optimal extraction of the resource is designed and planned, while keeping firmly in mind the economic, environmental and the societal aspects of a responsible project.

The operative phrase for future mining must be (Moss, 2005)

> 'Plan the mine, mine the plan'

In this way, the engineering and production/operations team, working together, form a very strong profit center. Inadequacies exhibited by either partner can lead to very high, unexpected costs, in one or more of the three legs – economic, environmental, and social. This is not 'Responsible Mining' and the consequences may be severe.

Today, even while the demand for well-engineered mines is increasing world-wide, the supply of the required mining, minerals, metallurgical/process, and geological engineers is small and decreasing rather than growing. Furthermore, the teaching corps required to train the future engineers is also diminishing through retirements and the lack of new-hires at the mineral's universities. Increasing demand and diminishing supply is not a good recipe for meeting the 'Responsible Mining' challenges of the future.

8.8.2 *The Milos Statement*

The 'Milos Statement: Contribution of the Minerals Professional Community to Sustainable Development' is used to close this chapter. This document was formulated and endorsed by the minerals professional community during their meeting at Milos, Greece in May 2003 (Karmis, 2003). This group is comprised of engineers, scientists, technical experts, and academics who work in, consult for; study, or are in some other manner associated with the

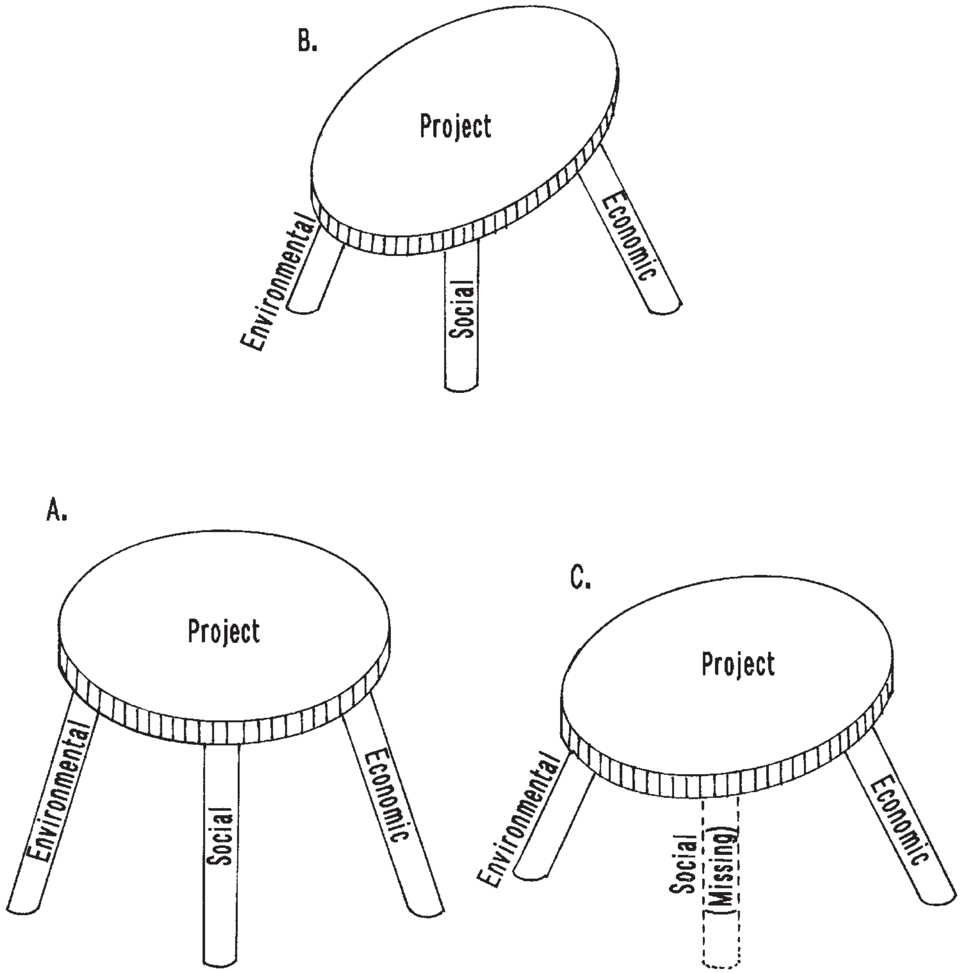

Figure 8.1. The three-legged stool analogy illustrating 'Responsible Mining' (A) and two undesirable alternatives (B and C).

minerals industry. The continuing efforts of this group will have a large impact as to how well we succeed toward the goal of 'Responsible Mining.'

The Milos Statement is:

Society's transition towards a sustainable future cannot be achieved without the application of the professional principles, scientific knowledge, technical skills, educational and research capabilities, and democratic processes practiced by our community. Our members share a mutual responsibility with all individuals to insure that our actions meet the needs of today without compromising the ability of future generations to satisfy their own needs.

What we believe:

We believe minerals are essential to meeting the needs of the present while contributing to a sustainable future.

The process of civilization is one of advancing intellectual, social and cultural development for all of humankind. Ah important part of the history of civilization is the history of scientific discoveries and technological advancements that have turned raw materials into resources, and in so doing provided the means for increased human well-being. The benefits and services derived from minerals, metals and fuels can contribute to the achievement of a sustainable future because it is the inherent characteristics of these resources that make productivity and consumption gains possible.

Achievement of a balance among economic prosperity, environmental health, and social equity will require significant changes in business strategies and operations, personal behaviors and public policies. Our minerals professional community can assist societies in balancing the need for mineral raw materials against the need to protect the environment and societies from unnecessary adverse impacts as they strive to improve quality of life.

Our vision for the future:

Our minerals community will support the transition to a sustainable future through integrated and experimental use of science, engineering and technologies as resources to people, catalysts for learning, providers of increased quality of life, protectors of the environmental and human health and safety.

What needs to be done to achieve our vision:

Professional Responsibility:

- Work to ensure that minerals and the capability to produce minerals are available to meet the needs of current and future generations.
- Encourage the development, transfer, and application of technologies that support sustainable actions throughout the product and mine life cycles.
- Give high priority to identifying solutions for pressing environmental and developmental challenges as related to sustainable development.
- Address social equity, poverty reduction and other societal needs as integral to minerals and mining related endeavors.
- Participate in the global dialog on sustainable development.
- Engage in all stages of the decision-making process, not only the project execution phase.

Education, Training, and Development:

- Attract the best people to the fields of mining and minerals by encouraging, facilitating and rewarding excellence.
- Build-up and maintain a critical mass of engineering, technical, scientific and academic capacity through improved education and training.
- Promote the teaching of sustainability principles in all engineering programs at all academic levels.
- Support and commit funding to the infrastructure that enables nations to provide mineral education, professional training, information, and research.
- Prevent the loss of core competencies.
- Transfer existing knowledge to the next generation of our community prior to and after retirement.

- Create a global exchange in academic training, as well as apprenticeship and internships programs.
- Support professional growth and interaction through books, articles, symposia, short courses and conferences on minerals and mining in sustainable development.

Communication:

- Share and disseminate sound information, knowledge, and technology.
- Share information on every aspect of minerals and mining to all appropriate audiences through journals, conferences, magazines, the Internet and other appropriate media.
- Disseminate technical information on sustainability, and the role of the minerals, metals and fuels in sustainable development, including information on the role of minerals in maintaining a high quality of life.
- Promote the achievements and capabilities of mineral community professionals to managers and executives, policy makers and the general public.

8.9 CONCLUDING REMARKS

'Responsible Mining' must be the message for now and especially in the future for everyone working in the mining industry. It must be practiced daily. Exceptions must be quickly identified and corrected. If we are to continue to have the privilege of producing the minerals that the world's people require, we must demonstrate that we are up to the task of extracting them in a responsible way. The three-legged stool image of economics, environment, and society should be kept clearly in mind.

REFERENCES AND BIBLIOGRAPHY

Abdel Sabour, S.A., and G. Wood. 2009. Modelling financial risk in open pit mine projects: implications for strategic decision-making. JSAIMM. 109(3): 169–175.

Alcor. 2005. Notes on Our Common Future. thttp://alcor.concordia.ca/∼raojw/crd/reference/reference 001377.html.

Allen, R. 1980. How to Save the World: Strategy for World Conservation. Kogan Page Limited, London. 150 pp.

Alvarez, S.M., and B.A. Ridolfi. 1999. Moon Creek reclamation project: creative solutions for an historic mine site. Pre-print Paper 99-088. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Anderson, B., Berry, D., and D. Shields. 2002. Sustainable development and its influence on mining operations on federal lands – A conversation in plain language. April. BLM/WO/GI-02/008+3000.

Anderson, K. 2004. Influence, rules and post-closure sustainability: Who will govern after the gates are closed? Pre-print Paper 04-167. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Anonymous. 2002a. ICMM Toronto Declaration. May 15.

Anonymous. 2002b. 2002 Sustainable Development Report: Sustaining a healthy balance. Noranda Inc./ Falconbridge Limited.

Anonymous. 2003. Milos statement on the contribution of the minerals professional community to sustainable development: Draft for comment. Feb 23.

Anonymous. 2005a. World Commission on Environment and Development. http://www.wsu.edu:8080/∼ susdev/WCED87.html.

Anonymous. 2009. Sustainable development initiatives gain momentum in mining. E/MJ. 210(7): 28–34. September.

Arell, P., and D. Folkes. 2004. The superfund hazard ranking system and mining sites. Pre-print Paper 04-48. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Baird, B.K., and P.C. Satchwell. 1999. Application of economic parameters and cutoffs during and after pit optimization. Pre-print Paper 99-069. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Barker, J.M., and V.T. McLemore. 2004. Sustainable development and industrial minerals. Pre-print Paper 04-171. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Barnes, H., Hawthorne, T., and D. Williams. 2000. Estimating site closure costs: A critical tool for reclamation planning. Paper 00-44. SME Annual Meeting. Feb 28–Mar 1. SLC, UT.

Basu, A.J., and V. Carabias-Hütter. 2004. Indicators of sustainability and reporting. Pre-print Paper 04-159. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Bearman, R.A. 2007. Total System Design for Comminution, AUSIMM Conference Proceedings?, AUSIMM Conference Proceedings, Ninth Mill Operators' Conference Fremantle, WA, March 2007, pp. 3–7.

Berry, R.C. 1999. Mining and the environmental impact statement: Guidelines for baseline studies. Pre-print Paper 99-024. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Bingham, N. 1994. Mining's image – what does the public really think. Mining Engineering. 46(3): 200–203.

Botha, C., Fourie, J.D., Botha, D., and C. Bischoff. 2012. Progress in implementing the Mineral and Petroleum Resources Development Act 28 of 2002 (MPRDA) provisions for the employment of women in mining. JSAIMM. 112(5): 395–404.

Botts, S.D. 2003. The Antimina Project: The challenge of sustainable development in Peru. Pre-print Paper 03-33. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.

Botts, S.D., and F. Cantuarias. 2003. Creating sustainability at Antimina. CIM Bulletin 96(1074): 84–86.

Brice, W.C., and C.G. Buttleman. 2000. Reclaiming aggregate mining sites in Minnesota using private/public partnerships. Pre-print Paper 00-103. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Broadhurst, J.L., Petrie, J.G., and H. von Blottnitz. 2006. Understanding Element Distribution During Primary Metal Production, AUSIMM Conference Proceedings, Green Processing Conference Newcastle, NSW, June 2006, pp. 9–24.

Brundtland, G.H. 1987. Foreword to Our Common Future: The World Commission on Environment and Development. 1987. Oxford University Press. Oxford. p. ix.

Cavender, B.W. 1994. Strategic ramifications of corporate environmental policy. Mining Engineering. 46(3): 204–207.

Cawood, F.T. An investigation of the potential impact of the new South African Mineral and Petroleum Resources Royalty Act. JSAIMM. 111(7): 443–453.

Chouinard, R., and A.J. Gunson. 2009. Sustainability in mining engineering – Strategies Through Education and Research. SDIMI Conference. Gold Coast, QLD. July 6–8. pp. 145–153.

Cooney, J.P. 2000. People, participation, and partnership. CIM Bulletin 93(1037): 41–45.

Corn, R.M. 2000. Consequences of the Forest Service CERCLA action at Mansfield Canyon. Pre-print Paper 00-77. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Cragg, A.W. 1998. Sustainable development and mining: Opportunity or threat to the industry? CIM Bulletin 91(1023): 45–50.

Crequer, D. 2006. The Golden Cross closure experience – a success story or a work without end? AUSIMM (through OneMine search), publication date: 1/1/06.

Czarnowsky, A.W. 2001. Controversial mining projects – Can you ever be prepared? Pre-print Paper 01-112. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.

Daniel, S.E. 2003. Environmental due diligence for exploration managers. CIM Bulletin 96(1074): 87–90.

Daniels, W.L., Schroeder, P., Nagle, S., Zelazny, L., and M. Alley. 2002. Reclamation of prime farmland following mineral sands mining in Virginia. Pre-print Paper 02-132. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.

Davis, G.A. 2004. An empirical investigation of mining and sustainable development. Paper 04-136. SME Annual Meeting. Feb 23–25. Denver, CO.

Davis, G.A. 2004. An empirical investigation of mining and sustainable development. Pre-print Paper 04-136. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Dorey, R., Duckett, R., and D. Salisbury. 1999. Operational management for closure and reclamation – Kennecott Ridgeway mine. Pre-print Paper 99-134. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Dunn, W.J. 2000. Beyond 'Bead'n Trinkets': A systematic approach to community relations for the next millennium. CIM Bulletin 93(1037): 41–45.

Ednie, H. 1999. The challenge of changing public perceptions of the industry. CIM Bulletin 92(1031): 9–10.
Edraki, T., Baumgartl, D., Mulligan, and R. Haymont. 2006. Post Closure Management of the Mt Leyshon Gold Mine – Water the Integrator, AUSIMM Conference Proceedings, 6th International Mining Geology Conference Darwin, August 2006, pp. 233–242.
Eger, P., Melchert, G., and J. Wagner. 2000. Using passive treatment system for mine closure – a good approach or risky alternative. Mining Engineering. 52(9): 78–84.
Evans, R., and C. Unger. 2010. What does sustainable development mean in the context of the minerals industry of Australia. AusIMM Journal. Issue 3. June.
Fitzgerald, A.D. 2004. Environmental and social responsibility practices: Moving to a global level playing field. CIM Bulletin 97(1078): 87–90.
Fox, F.D. 2003. Flambeau and Ridgeway Mines – Lessons learned. Pre-print Paper 03-141. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.
Fraser, R. 2007. Argyle Diamonds – Closing a Large Open Pit During a Mining Boom, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 1–4.
Gardner, J.S., and P. Sainato. 2005. Mountaintop mining and sustainable development in Appalachia. Pre-print Paper 05-067. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.
Garisto, N.C., et al. 2000. Environmental monitoring at mine sites – Future directions. CIM Bulletin 93(1038): 53–61.
Gascon, J-F., and S. Novak. 2009. Local resource development initiative – Developing a new project sustainability model (Ambatovy Project case study). Sustainability in mining engineering – Strategies Through Education and Research. SDIMI Conference. Gold Coast, QLD. July 6–8. pp. 171–179.
Gediga, J., Harsch, M., Saur, K., Schuckert, M., and P. Eyerer. 1996. Lifecycle assessment – An effective tool for environmental management. In Surface Mining 1996 (H.W. Glen, editor): 307–312. SAIMM.
GMI. 1998a. About ICMM. www.globalmining.com/about.php.
GMI. 1998b. Global Mining Initiative. www.angloamerican.co.uk/corporateresponsibility/sustainable development/gmi.
GMI. 1998c. GMI Issues. www.globalmining.com/gmi_issues_covered.php.
Gorton, W.T. 2004. Primer for engineering and technical professionals regarding reclamation and environmental surety bonds. Mining Engineering. 56(11): 36–38.
Gowens, J.K. 1998. A different role in designing new mining projects. CIM Bulletin 91(1026): 86–87.
Green, R. 2009. Holistic Management of Sulfides at Rio Tinto Iron Ore's Pilbara Mine Sites. Water in Mining Conference. Perth, WA. Sept 15–17. pp. 241–246.
Gu, X., and Q. Wang. 2011. Internalising ecological costs in evaluating and planning a mining project. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 547–555.
Hamaguchi, R.A. 1999. Innovative reclamation research at Highland Valley Copper. CIM Bulletin 92(1033): 78–84.
Hammer, S. 2008. Analyzing and allocating environmental risks in acquisitions. Paper 08-057. SME Annual Meeting. Feb 24–27. SLC, UT.
Harper, J. 2009. Dust management – thinking outside the square. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 207–210.
Harris, L. 1995. Yanacocha – Incas + 450 years: A model for investment in Peruvian mining. E/MJ. 196(8): 32–36.
Harris, L., and D. Malhotra. 2008. Cultural issues are important in the implementation of mining ventures. Mining Engineering. 60(9): 65–67. September.
Heath, J.S. 2000. Environmental alternative dispute resolution for mining – An introduction. Pre-print Paper 00-86. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.
Henry, D. 2007. New Technologies for Measurement of Operator Fatigue Helping to Deliver Improved Operator Health, Safety and Performance at the Mt Keith Operations, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 61–64.
Hodge, R.A. 2004. Tracking progress toward sustainability: Linking the power of measurement and story. Pre-print Paper 04-158. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.
Hodge, R.A. 2006. Tracking progress toward sustainability: linking the power of measurement and story. Mining Engineering. 58(9): 63–68. September.

Hogeman, D.C. 2002. Growing greener: A watershed approach. Pre-print Paper 02-121. 2002 SME Annual Meeting, Feb 26–28. Phoenix, AZ.

Holtom, G. 2010. The need for an environmental management system – and what this means for mines. E/MJ. 211(3): 46–49. April.

Horswill, D.H. 2000. Mining and sustainable development. Pre-print Paper 00-81. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Horswill, D.H. 2000. Mining and sustainable development: A Red Dog case study. Paper 00-81. SME Annual Meeting. Feb 28–Mar 1. SLC, UT.

Horswill, D.H., and H. Sandvik. 2000. Mining and sustainable development at Red Dog. Mining Engineering. 52(11): 25–31.

ICMM. 2002a. Mission. www.globalmining.com/about.php.

ICMM. 2002b. Organization. www.globalmining.com/about_organisat.php.

ICMM. 2002c. Principles. www.globalmining.com/icmm_principles.php.

ICMM. 2002d. ICMM Toronto Declaration. www.icmm.com/library_pub_detail.php?rcd=62.

Irawan, I., Mumbunan, C.E.F., and A. Ardianto. 2005. Community development in the urban area of a developing country – a case study of the Antam-Pongkor gold mine, Java Island, Indonesia. Mining Engineering. 57(2): 37–41. February.

James, P.M. 1999. 1999 Jackling Lecture: The miner and sustainable development. Mining Engineering. 51(6): 89–92.

Javier, M. 2012. Sustainability for nature and mankind. Paper 12-081. SME Annual Meeting. Feb 19–22. Seattle, WA.

Johnson, D., and K. Ward. 2001. EPA's EPCRA assistance visits: Getting prepared (EPCRA actually consists of sections other than 313?). Pre-print Paper 01-6. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.

Johnson, D.C., and H.F. Letient. 2004. Environmental planning in mine waste management – The Huckleberry Mines experience. CIM Bulletin 97(1077): 62–65.

Jones, H. 2011. Mine closure: plan for the end from the beginning. Mining Engineering. 63(5): 107–109. May.

Journeaux, D. 2005. Rock remediation techniques for mining and quarry operations. Pre-print Paper 05-112. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.

Joyce, S., and I. Thomsom. 2000. Earning a social license to operate: Social acceptability and resource development in Latin America. CIM Bulletin 93(1037): 49–53.

Kaldany, R. 2004. Sustainable mining development: listening to stakeholders. Mining Journal Supplement. London. January. pp. 8–11.

Kaliampakos, D.C., and A.G. Benardos. 2000. Quarrying and sustainable development in large urban centres: A contradiction in terms. CIM Bulletin 93(1040): 86–89.

Karmis, M. 2003. The Milos Statement: Contribution of the Minerals Professional Community to Sustainable Development. May. SME, Inc.

Kemp, D.J. 2005. Sustainable standards for the mining and metallurgical industry of the 21st century. CIM Magazine. May. pp. 253–255.

Kempton, J.H. 2000. Cost-saving strategies for environmental compliance. Pre-print Paper 00-125. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Kempton, J.H., and D. Adkins. 2000. Strategies for permitting mines with delayed environmental impacts. Pre-print Paper 00-126. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.

Kent, A., Ringwald, J., and M. Scoble. 2004. The evolution of sustainable mining practice: CIM's potential role. CIM Bulletin. 97(1083): 38–45.

Ketellapper, V.L. 2004. Conducting sustainable development concurrently with the reclamation of abandoned mines at the French Gulch site near Breckenridge, Colorado. Pre-print Paper 04-160. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Kral, S. 2004. Sustainable development keys SME annual meeting. Mining Engineering. 56(6): 58:67.

Kumral, M., and R. Dimitrakopoulos. 2008. Selection of waste dump sites using a tabu search algorithm. 108(1): 9–13.

Langer, W.H., Giusti, C., and G. Barelli. 2003. Sustainable development of natural aggregate with examples from Modena Province, Italy. Pre-print Paper 03-45. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.

Langer, W.H., and M.L. Tucker. 2004. Quarry expansion – Sustainable management of natural aggregate. Mining Engineering. 56(1): 23–28.

LaRosa, C.M., Ariail III, J.L., and K.A. Larsen. 2006. Integrating community capacity building and reuse planning into the reclamation of American mine lands: how U.S. regulators are cleaning up with communities. Mining Engineering. 58(3): 59–64. March.

Lawrence, D., and B. Hebblewhite. 2011. Sustainable minerals education – We care, but do you? AusIMM Journal. Issue 5. Oct.

Le Roux, W. 2011. When is a workplace safe or unsafe?: The safety criterion in terms of the Occupational Health and Safety Act and the Mine Health and Safety Act. JSAIMM. 111(8): 529–533.

Lima, H.M., and P. Wathern. 1999. Mine closure: a conceptual review. Mining Engineering. 51(11): 41–45.

Little, M.J., Bye, A.R., and T.R. Stacey. 2007. Safety and Financial Value Created by Good Slope Management Strategies and Tactics, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, 10–11 September 2007, pp. 77–85.

Livo, K.E., and D.H. Knepper, Jr. 2004. Non-evasive exploration in an environmentally sensitive world. Pre-print Paper 04-172. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Luba, L., Jakeman, M., Lefebvre, N., and R. Aseervatham. 2006. Mining for Water – Partnering for Sustainable Water Use in Semi-Arid Regions, AUSIMM Conference Proceedings, Water in Mining Conference Brisbane, QLD, November 2006, pp. 199–203.

Macedo, J.M., Campos, D., and S. Chadwick. 2004. Alumar: Improving local communities capacities for implementation: A sustainable development model, São Luis-Maranhão, Brazil. Pre-print Paper 04-173. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

Mackay, T., and P. Bangerter. 2006. Innovating your way to sustainability in the mining and mineral processing industries. JSAIMM. 106(10): 695–696.

McAllister, M.L., Scoble, M., and M. Veiga. 1999. Sustainability and the Canadian mining industry at home and abroad. CIM Bulletin 92(1033): 85–92.

McCullough, C., and M. Lund. 2006. Pit Lake Sustainability – What Is It, and How Do I Get It?, AUSIMM Conference Proceedings, Water in Mining Conference Brisbane, QLD, November 2006, pp. 323–329.

McLemore, V.T., and D. Turner. 2004. Sustainable development and exploration. Pre-print Paper 04-170. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.

McLemore, V.T., and D. Turner. 2006. Sustainable development and exploration. Mining Engineering. 58(2): 56–61. February.

Meyers, K.L., Espell, R., and C. Deringer. 2003. AA Pad reclamation performance review at Barrick's Goldstrike mine. Pre-print Paper 03-97. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.

Mitchell, R. 2012. Comparing EIA and ESHIA for evaluating mining projects. Paper 12-157. SME Annual Meeting. Feb 19–22. Seattle, WA.

Mitchell, R.E. 2012. Comparing EIA and ESHIA for evaluating mining projects Mining Engineering. 64(8): 87–92. August.

MMSD. 2000. MMSD project. www.globalmining.com/gmi_mmsd_project.php.

MMSD. 2002a. Breaking New Ground. Earthspan Publications.

MMSD. 2002b. Executive Summary for Breaking New Ground. www.iied.org/mmsd.

Moellenberg, D.L. 2005. Trends in hard rock mined land reclamation and financial assurance requirements. Paper 05-028. SME Annual Meeting. Feb 28–Mar 2. SLC, UT.

Moellenberg, D.L. 2005. Trends in hard rock mined land reclamation and financial assurance requirements. Pre-print Paper 05-027. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.

Moore, P. 2010. Mine closure – The end is nigh. Mining Magazine. October. pp. 32–34.

Mtegha, H.D., and O. Oshokoya. 2011. Mining fiscal environment in the SADC: status after harmonization attempts. JSAIMM. 111(7): 455–458.

Mudd, G.M. 2009. Sustainability reporting and mining – An assessment of the state of play for environmental indicators. SDIMI Conference Gold Coast, QLD. July 6–8. pp. 377-391–179.

Mudder, T., and K. Harvey. 1999. The state of mine closure: Concepts, commitments, and cooperation. Pre-print Paper 99-047. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.

Munro, D.A. 1980. Preface to 'How to Save theWorld: Strategy forWorld Conservation' by Robert Allen. Kogan Page Limited, London. 150 pp.

Nalven, J.G., and J.A. Sturgess. 1999. Toxic release inventory reporting and consequences for a typical gold mining facility. Pre-print Paper 99-048. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.
Nash, D. 2001. The importance of environmental compliance programs. Pre-print Paper 01-10. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.
Neuman, L. 2004. The ten most common mistakes that delay the review process. Pre-print Paper 04-20. 2004 SME Annual Meeting. Feb 23–25. Denver, CO.
NMA. 2002. The National Mining Association Sustainable Development Principles. http://www.nma.org/policy/sustainable_dev.asp.
Nzimande, Z., and H. Chauke. 2012. Sustainability through responsible environmental mining. JSAIMM. 112(2): 135–139.
Oraee, K., Siasirad, S., and M. Tavassoli. 2012. Mining development in the context of regional development. Paper 12-039. SME Annual Meeting. Feb 19–22. Seattle, WA.
Our Common Future: The World Commission on Environment and Development. 1987. Oxford University Press. Oxford. 383 pp.
Pinter, A. Burger, and A. Spitz. 2000. NEMA, mining and metallurgy and the social environment: Implications from the melting pot. JSAIMM. 100(3): 153–154.
Ramage, A., and D. Anderson. 2012. Decision tools for integrating sustainability into planning and feasibility phases of projects for extractive enterprises. Paper 12-012. SME Annual Meeting. Feb 19–22. Seattle, WA.
Rathbone, S. 2006. Open Cut Mining Adjacent to Ephemeral Creek and Spring Systems – Pilbara, Western Australia, AUSIMM Conference Proceedings, Water in Mining Conference Brisbane, QLD, November 2006, pp. 351–360.
Reichardt, C.L. 2009. The challenge of building local capacity to support the development of a sustainable mining industry in emerging mining nations. JSAIMM. 109(3): 163–168.
Rich, D. 2009. Gathering and managing environmental quality data for mining projects. Paper 09-005. SME Annual Meeting. Feb 22–25. Denver, CO.
Robbins, D.A. 1999. Mining industry benefits and concerns on environmental remote sensing. Pre-print Paper 99-050. 1999 SME Annual Meeting, Mar 1–3. Denver, CO.
Robertson, A., and S. Shaw. 2004. Use of the multiple accounts analysis process for sustainability optimization. Pre-print Paper 04-163. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.
Robertson, A., and S.S. Shaw. 2006. Use of the multiple-accounts-analysis process for sustainability optimization. Mining Engineering. 58(8): 33–38. August.
Roodt, N.J., Van Der Want, M., and C. Picas. 2010 Mine dumps under the amended Mineral and Petroleum Resources Development Act 28 of 2002. JSAIMM. 110(8): 469–472.
Ross, J. 2011. Risk management – Are we on the wrong track? AusIMM Journal. Issue 6. Dec.
Ross, C., and D. Crequer. 2006. Land rehabilitation at Golden Cross, 8 years after mine closure, AUSIMM (through OneMine search), publication date: 1/1/06.
Ryan, D. 2011. Keeping up with rising stakes of sustainability. AusIMM Journal. Issue 6. Dec.
Salazar, E., Cawrse, I., Bateman, H., Carnes, G., and S. Cawrse. 2009. Defining sustainability criteria in project design – A case study of the Olympic Dam expansion. SDIMI 2009 Sustainable Development Indicators in the Minerals Industry. Gold Coast, QLD. July 6–8. pp. 135–142.
Samson, M. 2006. Waihi's 2020 Vision: Part A: The Community's Blue-print, AUSIMM (through OneMine search), publication date: 1/1/06.
Scott, P. 1980. Foreword to 'How to Save the World: Strategy for World Conservation' by Robert Allen. Kogan Page Limited, London. 150 pp.
Semenoff, M.N. 2001. Talking trash? – The RCRA cleanup reforms. Pre-print Paper 01-50. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.
Sirolli, E., and F. Sirolli. 2012. Mining and community development: From rhetoric to practice. Paper 12-037. SME Annual Meeting. Feb 19–22. Seattle, WA.
Smith, G.L., Surujhlal, S.N., and K.T. Manyuchi. 2008 Strategic mine planning communicating uncertainty with scenarios. JSAIMM. 108(12): 163–168.
Smithen, A.A. 1999. Environmental considerations in the preparation of bankable feasibility documents. JSAIMM. 99(6): 317–320.
Sommerville, K., and J. Heyes. 2009. Be open to closure – It can save you money. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 317–322.

Stacey, J., Naude, A., Hermanus, M. and P. Frankel. 2010. The socio-economic aspects of mine closure and sustainable development. JSAIMM. 110(7): 395–413.
Stacey, J., Naude, A., Hermanus, M. and P. Frankel. 2010. The socio-economic aspects of mine closure and sustainable development: literature overview and lessons for the socio-economic aspects of closure. JSAIMM. 110(7): 374–394.
Stephenson, H.G., Van Den Bussche, B., and P. Curry. 1996. Reclamation, rehabilitation and development of abandoned mine land at Canmore, Alberta. CIM Bulletin 89(??):???.
Swarbrick, B. 2000. Mining in a multi-stakeholder environment: Falconbridge's Raglan project. Pre-print Paper 00-23. 2000 SME Annual Meeting, Feb 28–Mar 1. SLC, UT.
Terbrugge, P.J., Wesseloo, J., Venter, J., and O.K.H. Steffen. 2006. A risk consequence approach to open pit slope design. JSAIMM. 106(7): 503–511.
Thorley, U. 2012. Mine reclamation costs in the Alberta oil sands. Mining Engineering. 64(8): 93–99. August.
Tilton, J.E. 2003. Creating wealth and competitiveness in mining. Mining Engineering. 55(9): 15–21.
Tuttle, S., and R. Sisson. 1999. Closure plan for the proposed millennium project. CIM Bulletin 92(1026): 95–100.
Ubillus, M.J., and M. Wong. 2008. Building partnerships with indigenous people. E/MJ. 209(4): 52–55. May.
UNCED. 1992a. UN Conference on Environment and Development (1992). http://www.un.org/geninfo/bp/enviro.html.
UNCED. 1992b. Report of the United Nations Conference on Environment and Development. Annex I – Rio Declaration on Environment and Development. http://www.un.org/documents/ga/conf151/aconf15126-lannex1.htm.
UNCED. 1992c. Agenda 21 – Table of Contents. http://www.un.org/essa/sustdev/documents/agenda21/english/agenda21toc.htm.
UNCED. 1992d. Agenda 21 – Full text. http://www.un.org/esa/sustdev/agenda21.htm.
UN Doc.: A/CONF.48/14/Rev1 at 3(1973), 11 ILM 1416 (1972).
UNEP. 1972a. Report of the UN Conference on the Human Environment, Stockholm.
UNEP. 1972b. Report of the UN Conference on the Human Environment. http://www.unep.org/Documents/Default.asp?DocumentID=97.
Upton, B., Harrington, T., and S. Mendenhall. 2004. Sustainable development and mine closure planning. A case study: Golden Sunlight Mine, Jefferson County, Montana. Pre-print Paper 04-162. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.
Vezina, S. 2005. Environmental management in a global economy. CIM Magazine. December 2005/January 2006. pp. 438–440.
Wagner, D.M., Osmundson, M., Biggs, G., and C.J. Twaroski. 2001. Class 1 Areas Impact Analyses for a proposed expansion of a taconite processing facility in northeast Minnesota. Pre-print Paper 01-24. 2001 SME Annual Meeting, Feb 26–28. Denver, CO.
Wahrer, R.J. 2003. So you have a T/E species on your permit: How to survive. Pre-print Paper 03-102. 2003 SME Annual Meeting, Feb 24–26. Cincinnati, OH.
Wallace, J., Morris, B., and P. Howarth. 2008. Environmental management of mine sites with LiDAR altimetry and hyperspectral imaging. CIM Magazine. February. pp. 23–32.
Weber, I. 2005. Actualizing sustainable mining: 'Whole mine, whole community, whole planet' through 'industrial ecology'and community-based strategies. Pre-print Paper 05-102. 2005 SME Annual Meeting, Feb 28–Mar 2. SLC, UT.
Werniak, J. 2002. Working with nature and humanity. Canadian Mining Journal. August. pp. 16–18.
Williams, D.A., and D.R. Anstey. 2009. Integrated landforms for the storage of tailings and waste rock at magnetite mines in Western Australia. Iron Ore 2009 Conference, Proceedings, AusIMM Publication Series 7/2009. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 333–340.
Wilson, T.E., and T.M. Dyhr. 2004. Cost trends – Environmental management of mine operations. Pre-print Paper 04-125. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.
Wright, D., and J. Samborski. 2004. Sustainability: More than just reclamation. Pre-print Paper 04-174. 2004 SME Annual Meeting, Feb 23–25. Denver, CO.
WSSD. 2002a. Johannesburg Summit 2002 feature story – The Johannesburg Summit Test: What will change? http://www.johannesburgsummit.org/html/whats_new/feature_story41.html.

WSSD. 2002b. The World Summit on Sustainable Development. www.natural-resources.org/minerals/CD/sustdev.htm.
WSSD. 2002c. Plan of Implementation of the World Summit on Sustainable Development. www.un.org/esa/sustdev/documents/WSSD_POI_PD/English/WSSD_PlanImpl.pdf.
WSSD. 2002d. Johannesburg Declaration on SustainableDevelopment.www.un.org/esa/sustdev/documents/WSSD_POI_PD/English/POI_PD.htm.
Yearley, D.C. 2003. 2003 Jackling Lecture: Sustainable development for the global mining and metals industry. Mining Engineering. 55(8): 45–48.
Yernberg, W.R. 2001. Antamina – Peruvian mine of the future. Mining Engineering. 53(12): 19–21.
Younger, P.L. 2006. The Water Footprint of Mining Operations in Space and Time – A New Paradigm for Sustainability Assessments?, AUSIMM Conference Proceedings, Water in Mining Conference Brisbane, QLD, November 2006, pp. 13–21.

REVIEW QUESTIONS AND EXERCISES

1. What is meant by 'the bottom line'?
2. What is meant by 'the hurdle rate'?
3. Discuss the significance of equation (8.2).
4. In practice, how would you produce the 'Environmental cost/unit' and 'Social cost/unit' cost components?
5. Who are the 'stakeholders' in a mining project?
6. Discuss the meaning of 'Economic responsibility,' 'Environmental responsibility,' and 'Social responsibility' with regard to a mining project.
7. What is meant by the phrase 'Responsible mining'?
8. What is meant by the term 'Sustainable development'?
9. What is meant by 'Responsible mining practices'?
10. Summarize the five most important statements, in your opinion, contained in the proclamation made at the conclusion of the UNCHE.
11. From a mining viewpoint, list the ten most important principles included in the Stockholm Declaration. Be prepared to discuss the reasons for your selection.
12. What was the intention of the World Conservation Strategy?
13. How does the 'Strategy' relate to mining?
14. How does the 'Strategy' define sustainable development?
15. What are the obstacles cited? How should mining respond?
16. What were the goals of the 'World Commission on Environment and Development'? How do they affect the mining business?
17. How did the World Commission define 'sustainable development'?
18. Discuss the impact of the following statement on mining:
    'Sustainable development is considered to be a process of change in which the exploitation of resources, the direction of investments, the orientation of technological development, and institutional change are made consistent with future as well as present needs.'
19. What were identified by theWorld Commission as the two key concepts in sustainable development?
20. What is a non-renewable resource?
21. In the report 'Our Common Future', what were some of the important concepts regarding non-renewable resources?

22. What are some of the requirements to achieve sustainable development? Are they reasonable? Are they achievable?
23. What were the goals of the 'Earth Summit'?
24. From a mining viewpoint, summarize the ten most important principles, in your opinion, proclaimed? Be prepared to discuss your reasoning.
25. If you were in the mining business, which of the chapters in Agenda 21 would you be most interested in reading? Be prepared to discuss your reasoning.
26. Summarize the main things related to mining that came out of the World Summit on Sustainable Development. If you were a mining executive, how would you respond?
27. What was the Global Mining Initiative (GMI)? Who was the driving force behind it?
28. What were the main mining issues in question?
29. What is meant by the statement 'The mining industry has accepted that, alone, it cannot re-write its reputation'? What is the suggested way forward?
30. What were the three main activities identified by the GMI? What was the result?
31. What is the International Council on Mining and Metals (ICMM)?
32. What are the goals of the ICMM?
33. Who are the current members of the ICMM?
34. What are the principles of the ICMM? How do they relate to the principles put forth in the UN – sponsored conferences?
35. How is it expected that the ICMM-stated principles will get put into practice?
36. What was the Mining, Minerals and Sustainable Development (MMSD) project?
37. What were the objectives of the MMSD?
38. In what order would you read the MMSD report? Why?
39. The U.S. Bureau of Land Management (BLM) and the U.S. Forest Service are two major land administrators in the U.S. Why is it important to understand their view on sustainable development?
40. Summarize the view of these two government agencies with respect to sustainable development as expressed in section 8.7.5. As a citizen, are you comfortable with this view? As a miner, are you comfortable with this view? Be prepared to discuss your reasoning.
41. Summarize the point of view of the National Mining Association (NMA) with respect to environmental aspects as contained in their Sustainable Development Principles statement.
42. Summarize the point of view of the National Mining Association (NMA) with respect to social aspects as contained in their Sustainable Development Principles statement.
43. Summarize the point of view of the National Mining Association (NMA) with respect to economic aspects as contained in their Sustainable Development Principles statement.
44. In the opinion of Yearly, sustainable development, from the mining viewpoint, requires three things. What are they?
45. Discuss the concept that 'sustainable development is a journey rather than a destination.'
46. Discuss the need to make a 'business case' for sustainable development. Why is this important?
47. In what ways can sustainable development strategies help to minimize mining costs?
48. Comment on the statement 'Managing environmental and social impacts is not just a cost to be minimized.'

49. What are the key priorities for enhancing the contribution of mining, minerals and metals to sustainable development?
50. What role must mine engineering play in 'Responsible Mining'?
51. Is mine engineering a 'cost' center or a 'profit' center? Be prepared to discuss this.
52. In your own words, what is the practical meaning of the phrase 'Plan the mine, mine the plan'?
53. What is the way forward for mining in light of the substantial emphasis on sustainable development worldwide?
54. What is the significance of the Milos statement?
55. What is the Milos vision?
56. To achieve the Milos vision, what needs to be done? Summarize this in your own words.
57. How do you fit into the Milos vision and into its achievement?
58. Clearly state your view on the practical meaning of 'Responsible Mining.'

CHAPTER 9

Rock blasting

9.1 GENERAL INTRODUCTION TO MINING UNIT OPERATIONS

The term "unit operations" refers to the drilling, blasting, loading and hauling processes utilized in the extraction of an ore body. In their presentation, the authors have focused on the application of ANFO, rotary drilling, rope shovel loading and truck haulage. The reader is encouraged to further explore the various topics by referring to books such as "SME Mining Engineering Handbook, 3rd Edition" (SME, 2011), "The Blasters' Handbook" (ISEE, 2011) and "Blasting Principles for Open Pit Mining" (Hustrulid, 1999) as well as in the materials developed by equipment suppliers. The "Caterpillar Performance Handbook (Caterpillar, 2012)", the "Komatsu Specifications and Application Handbook (Komatsu, 2009)" and the "Blasthole Drilling in Open Pit Mining (Atlas Copco, 2011)" are excellent examples of the latter. The "Handbooks" may be located in pdf versions on their respective websites.

Proper rock fragmentation is the key first element of the ore winning process. It is a two-step activity in the sense that the holes for distributing the explosives within the rock mass must first be drilled (Step 1) as specified by the fragmentation plan. This is then followed by the controlled rubblization (Step 2) of the rock mass. The resulting product is then picked up (Step 3) and hauled away (Step 4). Hence, when considering the structure of a series of chapters on unit operations, it might be considered logical to begin with an integrated treatment of drilling and blasting under the heading of fragmentation. On the other hand, most mining books are organized by individual unit operation and the presentation is in the order in which they occur in the mine, i.e. drilling, blasting, loading, and hauling. This is also a very logical sequence from the miner's point of view. Today, one normally begins by designing the blasting pattern to meet the various imposed requirements and desired outputs. The requisite drilling system needed to drill the pattern is then selected. Following this logic, the authors have chosen to begin with rock blasting (Chapter 9) and then move to rock drilling (Chapter 10). The loading and hauling systems (Chapters 11 and 12, respectively) are largely sized based on the production requirements of the operation which controls the mine geometry. In following this logic, this series of chapters begins with blast design and continues with chapters on drilling, loading and hauling. Chapter 13 deals with equipment availability and utilization.

The mining world is, for better or worse, still bilingual with both English (Imperial) and SI units being used. In this series of chapters on unit operations no attempt has been made to follow one or the other. It is assumed that the reader can make the translation as necessary. The symbols used are consistent within any particular chapter but not necessarily between chapters. The authors apologize for any confusion that this may cause but it greatly simplifies the presentation for both the reader and the authors.

The original drafts for Chapters 10 through 13 were written about 20 years ago and although on first glance this might seem strange, the basic principles have not changed. The materials have been updated as required. Major assistance in this regard has been provided by equipment suppliers and mining companies – it has been invaluable. In the end, the content is the responsibility of the authors. Although care has been taken to avoid errors both in understanding and presentation, they unfortunately will be present in a work of this magnitude. The authors will be pleased if the reader will bring them to their attention so that corrections to future editions may be made.

Chapters 9 through 13 are primarily intended as an introduction to the unit operations for mining engineering students. However the material contained and the presentation form should make it of value to practicing engineers as well. The material presented in section 9.2 has been largely extracted from the book "Blasting Principles for Open Pit Mining (Hustrulid, 1999)."

9.2 ROCK BLASTING

9.2.1 *Rock fragmentation*

Important in the past and important today, carefully engineered blasting will be an even more important aspect of successful open pit mining in the future as (1) pits become deeper and steeper, (2) quality separation to avoid dilution and ore losses during blasting becomes paramount, and (3) due to energy cost concerns, greater attention is paid to optimizing the entire mine-mill fragmentation system. In the past and even today the mine and mill fragmentation sub-systems have often been treated separately with the only commonality being the primary crusher. As can be seen in Figure 9.1 the entire mine-mill flowsheet from in situ (with a "particle size" considered to be very large) to the point where it leaves the mill for further treatment (with a "particle size" often in the micron range) consists of two basic elements; fragmentation and transport.

Energy is an important ingredient in all aspects of mining today but it is especially important with regard to fragmentation. The energy required for size reduction is very dependent on particle size as well as on the means used to deliver the energy to the particles. In looking at Figure 9.1, it is natural to ask whether the application of additional explosive (chemical) energy during blasting might result in a reduction of more expensive energy input in later fragmentation stages. As early as the 1960's, there were serious discussions (and even tests) regarding this. Now, about 50 years later, mining companies are seriously looking at the possible tradeoffs and mine-mill fragmentation considerations are being included in the basic blast design evaluations. Therefore, both mining engineers and mining companies must have a firm grasp of blasting fundamentals and practice. This is irrespective of whether the actual design(s) and their implementation is done in-house or by contractors. In this section the authors have tried to bridge the gap between theory and practice and to present the underlying concepts in an understandable way. It is hoped that the contained material will provide a basis for engineers to improve (a) their blasting operations as well as (b) their ability to understand the content and potential application of papers appearing in the technical literature.

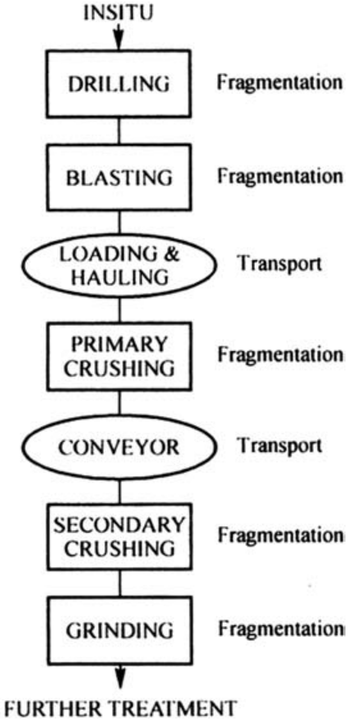

Figure 9.1. The mine-mill system represented as fragmentation and transport unit operations.

9.2.2 *Blast design flowsheet*

In the past only a relatively few explosive products with properties lying within a relatively narrow range were available for the fragmentation engineer to choose between. Today the products placed in the holes are many and their properties can be easily varied over the hole length. In addition initiation timing has been markedly improved. Thus the possibilities available to the blast designer are far more than those of a few short years ago. The challenge facing the mining engineer is how to most effectively use these possibilities. "Engineered fragmentation" as opposed to "blasting" or the epitome "military" blasting will be an even more important aspect of future mining. Figure 9.2 illustrates quite well the many controllable and uncontrollable variables involved in any given blast and the resulting outputs.

As can be seen the process of engineering a blast is a challenging process involving the marriage of

- explosive characteristics
- explosive fracturing phenomena
- layout geometry
- rock and rock mass properties
- timing
- sequencing

so that the desired degree of fragmentation as required by the down-stream operations as well as the other demands are satisfied.

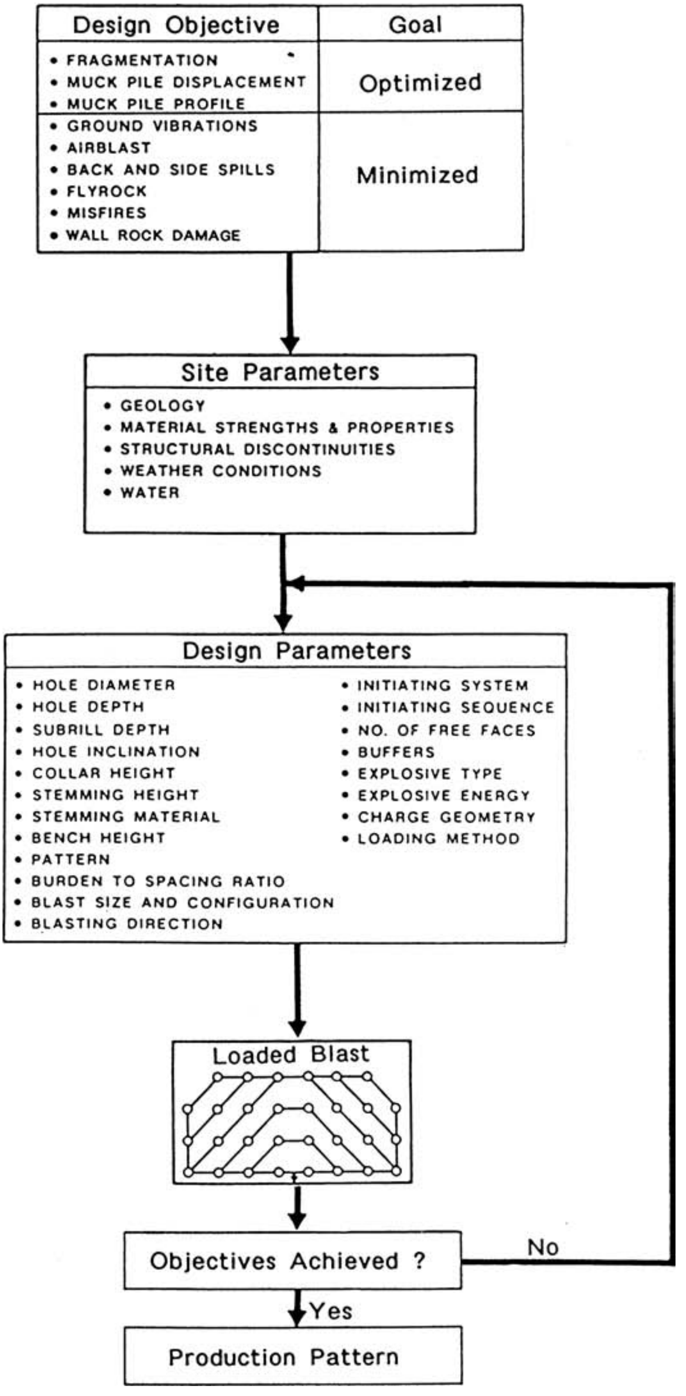

Figure 9.2. Blast design flowsheet logic (modified after Atlas Powder Company, 1987).

9.2.3 *Explosives as a source of fragmentation energy*

A common blasting agent used in open pit mines today is a combination of ammonium nitrate (NH_4NO_3) "AN" and No. 2 diesel oil/fuel oil (CH_2) "FO". The combination is called ANFO. Although neither of these components is an explosive in itself, under the proper conditions the mixture can be made to *detonate* (the explosion front will propagate along a column of explosive). Under other conditions, the mixture will simply *deflagrate* (burn) at a very rapid rate.

The chemical reaction for the process is given below

$$3NH_4NO_3 + CH_2 => 7H_2O \pm CO_2 + 3N_2 + \text{heat} \tag{9.1}$$

In this case, the AN is the oxidizer (it contains the oxygen) and the fuel oil is the fuel. The fuel oil is oxidized and the AN is reduced in a very, very short time. As can be seen, the products are gases at high temperature. The amount of energy liberated in the form of heat is 912 calories per gram of ANFO.

The reaction is carried along the column of explosive at the velocity of detonation (VOD) which for ANFO is of the order of 4529 m/second.

Although one knows that the power involved in an explosion is large, it is difficult to visualize just how large simply through the energy release value of 912 calories/gm. To help in this regard, consider a borehole 300 mm in diameter (D) and 8 m in length (L) filled with ANFO having a density (ρ_e) of 0.8 g/cm^3. The explosive column would have a volume (V_e) of

$$V_e = \frac{\pi D^2 L}{4} = \frac{\pi(0.30)^2(8)}{4} = 0.566\,\text{m}^3 \tag{9.2}$$

containing explosive with a mass (M_e) of

$$M_e = \rho_e V_e = 0.566(800) = 452\,\text{kg} \tag{9.3}$$

The total energy (E) unleashed would then be

$$E = 912\,\text{kcal/kg} \times 452\,\text{kg} = 412{,}000\,\text{kcal} \tag{9.4}$$

To obtain the energy in kilojoules one multiplies kilocalories by a factor of 4.184.

$$E = 4.184 \times 0.412 \times 10^6 = 1.72 \times 10^6\,\text{kJ} \tag{9.5}$$

Using a detonation velocity for ANFO of 4529 m/second, the time (t_e) required for the entire column to detonate is

$$t_e = \frac{L_e}{VOD} = \frac{8\,\text{m}}{4529\,\text{m/s}} = 1.77 \times 10^{-3}\,\text{seconds} \tag{9.6}$$

Thus the power (P_{ow}) generated is

$$P_{ow} = \frac{E}{t_e} = \frac{1.72 \times 10^6}{1.77 \times 10^{-3}} = 0.97 \times 10^6\,\text{MJ/second} \tag{9.7a}$$

or

$$P_{ow} = 3498 \times 10^6\,\text{MJ/hour} \tag{9.7b}$$

Note: *The prefix kilo (k) means 10^3, mega (M) means 10^6 and giga (G) means 10^9 of the unit in question. Thus 1 kcal means 10^3 calories.

By dividing the power expressed in MJ/hour by the factor 3.6 one obtains the energy in kW.

$$P_{ow} = 972 \times 10^6\,\text{kW} \tag{9.7c}$$

Since 1 horsepower (hp) is equal to 0.746 kW, the power output expressed in horse power is

$$P_{ow} = \frac{972 \times 10^6}{0.746} = 1.30 \times 10^9\,\text{hp} \tag{9.7d}$$

The challenge in blast design is to harness this power so that it performs the desired useful work.

9.2.4 *Pressure-volume curves*

An ANFO mixture will detonate when suitably confined (such as in a borehole) and initiated by a high explosive (called a primer) of sufficient intensity. The reaction progresses along the explosive column with a speed equal to the velocity of detonation (VOD). The pressure of the gas directly at the detonation front is called the *detonation* pressure (P_{DET}). For many explosives it may be approximated by

$$P_{DET}\,(\text{atm}) = 2.5\rho_e(\text{VOD})^2 \tag{9.8a}$$

$$P_{DET}\,(\text{MPa}) = 0.25\rho_e(\text{VOD})^2 \tag{9.8b}$$

where
ρ_e = density (kg/m^3)
VOD = detonation velocity (km/sec)
P_{DET} = detonation pressure

The *explosion pressure* (P_e) which denotes the gas pressure applied to the borehole walls just after detonation is approximately one-half of this value.

$$P_e = \frac{1}{2}P_{DET} \tag{9.9}$$

To demonstrate how this works, consider the simplified example of a 10 cm diameter borehole 200 cm in length filled with ANFO ($\rho_e = 0.8$ g/cm^3 and VOD = 4529 m/s).

The total volume (V_e) and mass (M_e) of explosive involved are respectively

$$V_e = \frac{\pi D_e^2 L_e}{4} = 0.0157\,\text{m}^3$$

$$M_e = \frac{\pi D_e^2 L_e \rho_e}{4} = 12.57\,\text{kg}$$

Using equations (9.8a) and (9.9), the estimated detonation (P_{DET}) and explosion (P_e) pressures are respectively

$$P_{DET} = (2.5)(800)(4.529)^2 = 41024\,\text{atm}$$

$$P_e = 20512\,\text{atm}$$

The given values are

$$P_{DET} = 43943\,\text{atm}$$

$$P_e = 19970\,\text{atm}$$

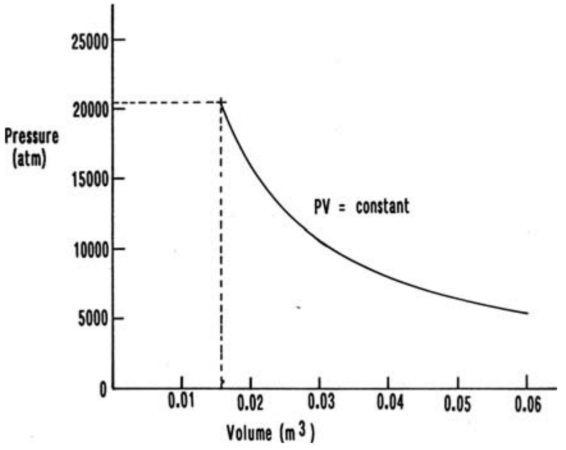

Figure 9.3. The predicted P-V curve assuming isothermal expansion.

Although not strictly applicable due to the very high temperatures and pressures involved, relationships based upon ideal gas behavior are very useful in demonstrating basic concepts. If the temperature is maintained constant (isothermal conditions) during the subsequent expansion of the explosive gases with an accompanying decrease in borehole pressure, then the right hand side (nRT) of the Ideal Gas Law

$$PV = nRT \tag{9.10}$$

where
P = Pressure (atm)
V = volume (liter/kg)
n = no. of moles of gas (moles/kg)
R = universal gas constant = 0.08207 liter-atm/(mole-°K)
T = temperature (°K)
is a constant. Equation (9.10) can be written as

$$PV = \text{constant} \tag{9.11}$$

An alternative form of this pressure-volume relationship familiar to all physics students is

$$P_1V_1 = P_2V_2 = P_eV_e \tag{9.12}$$

Knowing that
$P_e = 19970$ atm
$V_e = 0.01571\,m^3$
the pressure-volume curve shown in Figure 9.3 may be constructed.

However the temperature of the explosive gases *does not* remain constant. Initially it is very high (of the order of 2810°K) and then decreases with expansion to near ambient (298°K). A much better approximation to the true P-V curve describing this situation (a hole with a fully charged cross-section) is achieved by assuming that the expansion takes place adiabatically (there is no heat loss).

The appropriate Ideal Gas Law equation is

$$PV^{\gamma} = \text{constant} \tag{9.13}$$

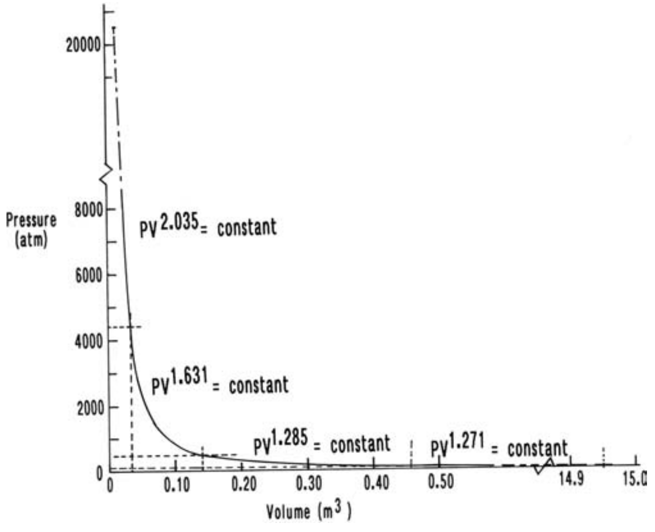

Figure 9.4. The idealized adiabatic P-V curve for ANFO over the entire expansion range.

where

γ = ratio of the specific heats.

For ANFO, the appropriate values of γ depend upon the pressure range. For this example the following values will be used:

Region 1:	$\gamma = 2.035$	$4500\,\text{atm} \le P \le 19971\,\text{atm}$
Region 2:	$\gamma = 1.631$	$500\,\text{atm} \le P \le 4500\,\text{atm}$
Region 3:	$\gamma = 1.285$	$100\,\text{atm} \le P \le 500\,\text{atm}$
Region 4:	$\gamma = 1.271$	$1\,\text{atm} \le P \le 100\,\text{atm}$

The idealized adiabatic P-V curve is shown in Figure 9.4.

The expansion energy can be obtained by finding the area (A) under the P-V curve. In this case it becomes

$$A = 161 + 133 + 66 + 103 = 463\,\text{atm-m}^3$$

This can now be converted into kilocalories by

$$A = 0.0242 \times 10^3 \times 463 = 11205\,\text{kcal}.$$

Since there are 12.57 kg of ANFO involved in the explosion the amount of energy released per kilogram calculated using the idealized P-V curve is

$$A = \frac{11205}{12.57} = 891\,\text{kcal/kg}$$

It should be recalled that the theoretical energy release is

$$Q = 912\,\text{kcal/kg}$$

and the difference between the two is heat energy (~21 kcal/kg) which remains trapped in the explosive products. The ratio between the amount of useful energy (A) available to the theoretical energy (Q) is called the mechanical efficiency (e).

$$e = \frac{A}{Q} \tag{9.14}$$

In this case it is

$$e = \frac{891}{912} = 0.977$$

which means that 97.7% of the theoretical explosive energy could do useful work if released in a controlled way down to a pressure of 1 atm.

9.2.5 *Explosive strength*

Although the discussion to this point has focused on ANFO, there are many other explosive types and variations. When selecting an explosive for a certain application, one of the more important characteristics to be considered is "strength". Over the years, various ways have been used by manufacturers to measure and describe the strength of their explosives. Today, unfortunately, there is no standard approach of producing and providing these data. It has become quite common, however, for manufacturers to include weight strengths and bulk strengths (both absolute and relative) on their product specification sheets.

The weight strength (S_{WT}) is defined as the explosive energy per unit weight (mass). For its calculation, the problem becomes that of defining which "energy" to use. The simplest is to use the theoretical heat of explosion (Q) calculated based upon the constituents. For ANFO (94.5%/5.5%) the value of Q is

$$Q = 912 \text{ kcal/kg}$$

Hence the weight strength is

$$S_{WT} = 912 \text{ kcal/kg}$$

The bulk strength (S_{BULK}) is defined as the explosive energy per unit volume and has units of kcal/m^3, cal/cm^3, etc. Since the cost per unit volume of hole created in the rock mass by drilling is substantial it is generally desired to pack as much explosive power into this volume as possible. Thus for most applications, the bulk strength is more important than the weight strength. The two are related through the density.

$$S_{BULK} = \rho_e S_{WT} \tag{9.15}$$

For ANFO with a density $\rho = 0.8$ g/cm^3, the bulk strength is therefore

$$\begin{aligned} S_{BULK} &= 0.8 \text{ g/cm}^3 \times 912 \text{ cal/gm} \\ &= 730 \text{ cal/cm}^3 = 730 \text{ kcal/m}^3 \end{aligned}$$

The "energy" used in the calculation could also be defined in some other way, i.e. that defined by the P-V curve, the gas bubble energy, etc. Manufacturers often publish relative weight strength and bulk strength values for their explosives. Most of the time, the explosive strengths are expressed relative to ANFO (94.5/5.5) of a given density, diameter and degree of confinement.

Assume for example that a certain explosive has a heat of explosion equal to 890 cal/g and a density of 1.3 g/cm^3. The weight strength of this explosive relative to ANFO is denoted by S_{ANFO}. Since for ANFO the heat of explosion is equal to 912 cal/gm and the density is 0.8 g/cm^3 then for the new explosive the relative weight strength is given by

$$S_{ANFO} = \frac{890}{912} = 0.976$$

On the other hand, the bulk strength relative to ANFO denoted by B_{ANFO} is

$$B_{ANFO} = \frac{890 \times 1.3}{912 \times 0.8} = 1.592$$

One might conclude that for the same hole diameter this explosive would be far superior for fragmenting the rock than ANFO. Unfortunately, there is not necessarily a 1 to 1 correlation between total energy applied and the fragmentation produced.

9.2.6 *Energy use*

In rock blasting the energy goes into

- creating new fractures
- extending old fractures
- displacing parts of the rock mass relative to others (loosening)
- moving the center of gravity forward (heave)
- undesirable effects: fly rock, ground vibrations, air blast, noise, heat.

Exactly how the energy is partitioned into these different categories depends upon

- the explosive
- the rock/rock mass
- the blast geometry

Some (hard, massive) rock types require the creation of new fractures for adequate fragmentation. The shock energies needed for new fracture generation are associated with high explosion pressures (high detonation velocity and high density).

Other rock types which are already cracked depend more upon the heaving/displacing action provided by gas pressures for breaking. This may be best accomplished by an explosive with a lower detonation velocity and density.

Two explosives could have exactly the same total energy (same areas under the curves) but as shown in Figure 9.5 quite different P-V curves.

Note that

- explosive A has a much higher peak pressure than explosive B
- the total energies of the two explosives are the same (Area 1 = Area 2).
- explosive B maintains a higher gas pressure with expansion than does explosive A.

Explosive A, termed a high brisance/low gas pressure explosive, would be recommended for use in hard, brittle rocks. Explosive B, on the other hand, a low brisance/high gas explosive for use in softer/more jointed rocks. To properly match explosive/rock/geometry and achieve optimum blasting results, it is therefore important to understand

- the rock mass failure process
- the partitioning of the explosive energy (shock energy/heave energy)
- explosive-rock interaction.

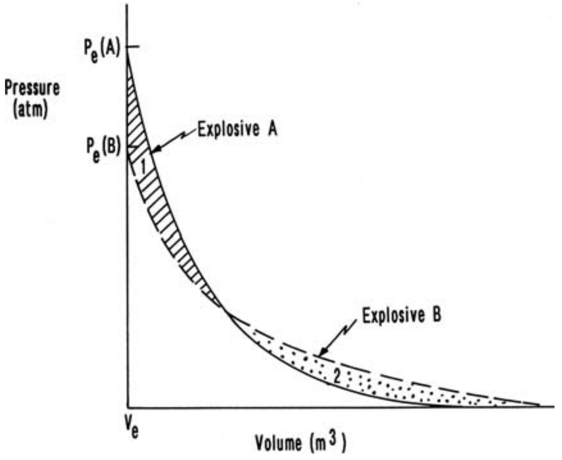

Figure 9.5. Diagrammatic representation of the P-V curves for two different explosives with the same energy.

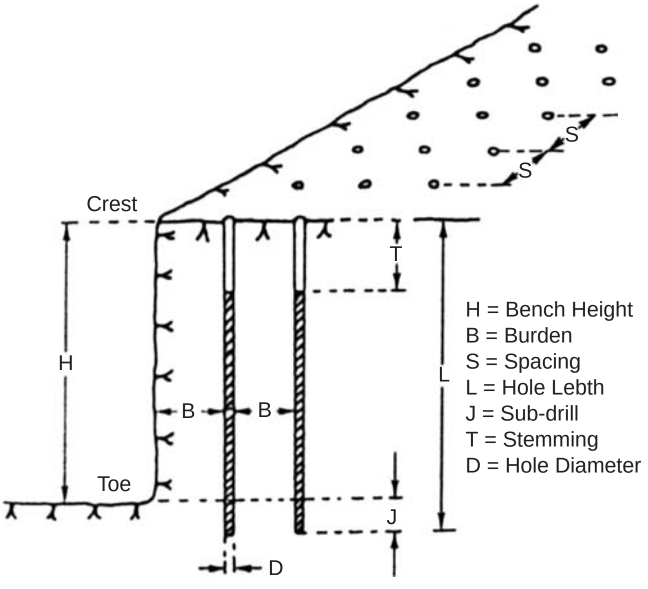

Figure 9.6. An isometric view of a bench showing blast geometry.

A discussion of these is beyond the scope of the current book. The interested reader is referred to Hustrulid (1999).

9.2.7 *Preliminary blast layout guidelines*

In this section guidelines which can be used for preliminary blast design will be provided. As a result of feedback from the field, the patterns can be then adjusted/optimized for the actual characteristics of the rock mass – explosive – geometry combination. The blasthole terminology which will be used is shown in Figure 9.6.

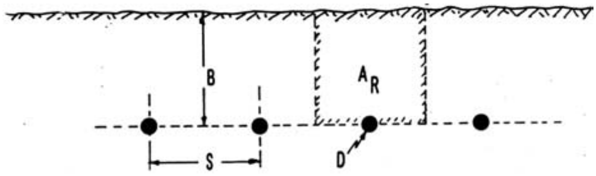

Figure 9.7. Bench plan view of the one row blast layout.

In bench blasting there is normally a long dimension of the bench and a short dimension. It will be assumed that the rows of blast holes are aligned parallel to the long dimension.

The drilled burden (B) is then defined as the distance between the individual rows of holes. It is also used to describe the distance from the front row of holes to the free face. When the bench face is not vertical the burden on this front row of holes varies from crest to toe. The spacing (S) is the distance between holes in any given row. Generally the holes are drilled below the desired final grade. This distance is referred to as the sub-grade drilling or simply the sub-drill (J). A certain length of hole near the collar is left uncharged. This will be referred to as the stemming length (T) whether it is left unfilled or filled with drill cuttings/crushed rock. The drilled length (L) is equal to the bench height (H) plus the sub-drill (J). The overall length of the explosive column (C) is equal to the hole length (L) minus the stemming (T). This column may be divided into sections (decks) containing explosives of various strengths separated by lengths of stemming materials. Sometimes the explosive strength is varied along the hole, i.e. a higher strength bottom charge with a lower strength column charge. As will be seen in the next section, the different dimensions involved in a blast design are not arbitrary but closely related to one another. The selection of one, for example the hole diameter, fixes within rather strict limits, many of the others.

9.2.8 *Blast design rationale*

This section presents a rationale for the type of geometrical design used in most open pit mines today. Five different design relationships will be introduced. Consider first the plan view (Fig. 9.7) of a bench in which the hole spacing (S) and burden (B) are as shown.

In viewing the figure it can be seen that the hole spacing can be expressed as a constant (K_s) times the burden

$$S = K_s B \tag{9.16}$$

where

K_s = constant relating spacing to the burden

This is the first of the fundamental design relationships.

Each hole of diameter D can be thought of as having to break its own individual area (A_R) as outlined by the dashed lines, in the figure.

$$A_R = B \times S \tag{9.17}$$

The volume required to be broken by a hole of unit length is

$$V_R = B \times S \times 1 \tag{9.18}$$

A certain amount of explosive energy per unit volume (E_v) must be applied to satisfactorily fragment the rock. The total energy (E_R) required is therefore

$$E_R = V_R \times E_v = B \times S \times E_v \tag{9.19}$$

Combining equations (9.16) and (9.19) the required energy becomes

$$E_R = K_s B^2 E_v \tag{9.20}$$

Hence the required fragmentation energy is proportional to the square of the burden.

$$E_R \propto B^2 \tag{9.21}$$

The amount of available explosive energy (E_A) is determined by the explosive volume (V_e) present in that unit length of borehole

$$V_e = \frac{\pi}{4} D_e^2 \tag{9.22}$$

where

D_e = explosive diameter

times the explosive bulk strength expressed as energy per unit volume (E_e)

$$E_A = \frac{\pi}{4} D_e^2 E_e \tag{9.23}$$

When using packaged explosives and at pit perimeters where perimeter blasting techniques are employed, the charge diameter (D_e) may be less than the (D) of the hole. However, in production blasting using bulk blasting agents, the entire cross-sectional area of the hole is filled with explosive. Thus the hole diameter (D) and the explosive diameter (D_e) are the same. This assumption will be used here.

Thus the available energy is proportional to the square of the hole diameter

$$E_A \propto D^2 \tag{9.24}$$

Setting the available and the required explosive energies equal to one another one finds that the burden is proportional to the hole diameter.

$$B \propto D \tag{9.25}$$

Introducing the proportionality constant K_B, the relationship can be written as

$$B = K_B D \tag{9.26}$$

where
K_B = constant relating burden to the hole diameter.

This is the second of the fundamental design relationships. The constant K_B, as will be discussed in Section 9.2.6, incorporates both explosive energy factors and the rock density. The design relationship (9.26) suggests a linear increase in the burden with hole diameter assuming the same explosive is used (Fig. 9.8).

The toe region (Fig. 9.9) is highly confined and extra explosive energy must be applied to assure adequate fragmentation.

This extra explosive energy is generally provided by extending the drill hole below the toe elevation, the so-called sub-drill length (J), with explosive. There are several different rationales used for selecting the appropriate length. One will be included under the discussion

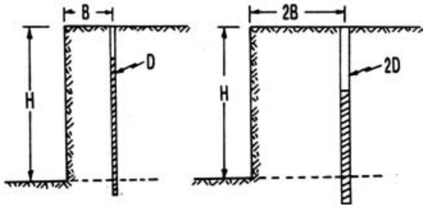

Figure 9.8. Diagrammatic representation showing the effect of hole diameter on burden.

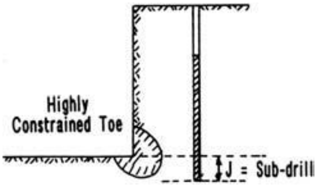

Figure 9.9. The geometrical relationship between the subdrill and the highly constrained toe.

of the stemming length. Here an explanation based upon explosive run-up distance will be presented. There is a certain distance (called the run-up distance) characteristic of the initiating system/explosive which the shock wave must travel away from the point of initiation before steady state conditions are reached in the explosive column. To break the confined toe, the borehole pressure should be as high as possible. Since the explosion (borehole wall) pressure (P_e) is proportional to the square of the detonation velocity

$$P_e \propto (VOD)^2 \tag{9.27}$$

the elevation in the hole at which steady state velocity is reached should not be higher than the bench toe elevation. To be conservative the minimum run-up distance will be assumed to be 6D.

In addition, the primer is seldom placed directly at the bottom of the blasthole due to the presence of cuttings and water. A normal offset is of the order of 2D. Therefore, the distance from the drilled end of the hole to the toe elevation (the sub-drill distance J) should be

$$\mathrm{J} \approx 8\mathrm{D} \tag{9.28}$$

as has been shown

$$B \propto D$$

and therefore the sub-drill J may be expressed as

$$\mathrm{J} = \mathrm{K_J B} \tag{9.29}$$

where

$\mathrm{K_J}$ = constant relating the sub-drill distance to the burden

This is the third of the design relationships. As will be seen later, the burden dimension (B) for most bulk explosives and rock types is of the order of

$$\mathrm{B} = (25 - 35)\mathrm{D} \tag{9.30}$$

Using equations (9.28) and (9.30), one would therefore expect that

$$K_J = 0.23 \rightarrow 0.32 \tag{9.31}$$

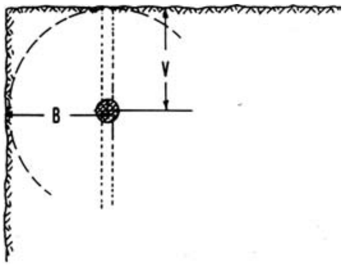

Figure 9.10. Section view showing a spherical charge located near the collar.

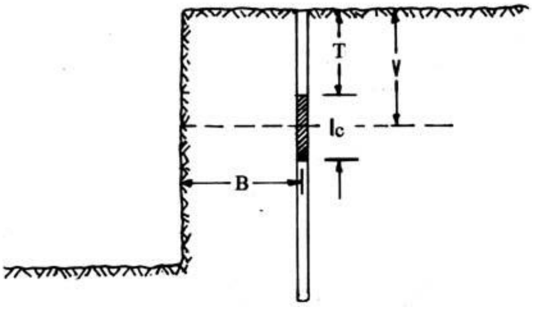

Figure 9.11. Section view showing a short cylindrical charge located at the hole collar.

A typical value used for design is $K_J = 0.3$.

Near the hole collar, the rise of the explosive should be controlled so that the possibility of breaking upward toward the horizontal free surface should be "as difficult" or "even more difficult" than breaking, as is desired, toward the vertical free face. This could be satisfied, for example, by the placement of a *spherical* charge capable of breaking burden 'B' at a distance of 'B' below the collar (Fig. 9.10).

The general constraint would be written as

$$V \geq B \tag{9.32}$$

The spherical charge geometry is not a practical one for most surface mining applications. However there is a practical equivalence in breaking effect between spherical and cylindrical charges. In Figure 9.11 the spherical charge has been replaced by a cylindrical charge of length T_C having the same total weight and effect.

Obviously the degree of 'equivalence' of the charges will depend upon the proximity to the charge. As a first approximation it will be assumed that T_C is linearly related to the distance of interest which in this case is B.

$$T_C = K_{TC}B \tag{9.33}$$

For B large, then T_C is large and vice versa. The general expression for the uncharged hole length (T) may be written as

$$T = V - \frac{1}{2}T_C \tag{9.34}$$

If one used the "as difficult" breaking constraint in equation (9.32), then

$$V = B$$

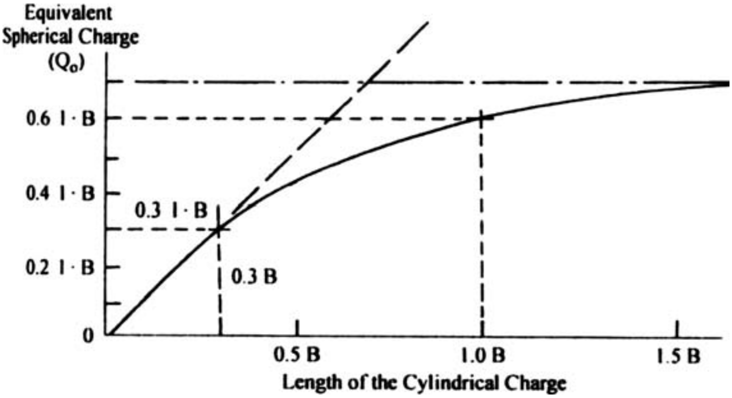

Figure 9.12. Toe breaking equivalence of spherical and cylindrical charges (Langefors and Kihlstrom, 1963).

Combining equation (9.33) and equation (9.34) subject to this condition yields

$$T = B - \frac{1}{2}K_{TC}B$$

$$T = \left(1 - \frac{1}{2}K_{TC}\right)B \tag{9.35}$$

Equation (9.35) can be simplified to

$$T = K_T B \tag{9.36}$$

where

$$K_T = 1 - \frac{K_{TC}}{2} \tag{9.37}$$

This becomes the fourth of the fundamental relationships. The problem is then the determination of K_{TC}.

For bench blasting Langefors and Kihlström (1963) have suggested that the spherical/cylindrical charge equivalence is as shown in Figure 9.12.

To explain the significance of the curve, consider a bench containing two side-by-side vertical blastholes. The burden is the same for both. Rather than discussing the collar region which is the subject of this portion, this example will involve the toe region. The reason for this is that the explanation is easier and the principle is the same. Consider a spherical charge of quantity Q_o placed at the toe elevation in one of the holes. In the second blasthole a column charge with a linear charge concentration of 1 kg/m of hole is emplaced. The bottom of the charge is at toe elevation and then the column extends upward towards the collar. The length of the elongated charge is expressed in multiples of the burden B. For a column charge of length B, the total charge would be B × 1. From Figure 9.12 one can see that at the toe this elongated charge has only the equivalent breaking power of a spherical charge of weight 0.6 × 1 × B. This is understandable since the energy contained in that part of the elongated charge near the collar must travel a much longer distance to reach the toe and in the process the energy is spread over a much larger volume of rock. The energy density by the time it reaches the toe is much less than that produced by energy which has

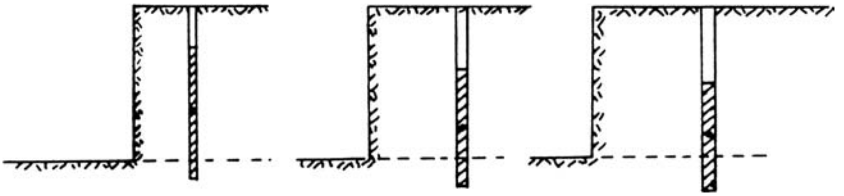

Figure 9.13. The effect of charge diameter on the charge center of gravity location.

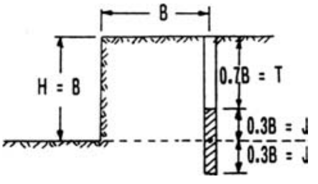

Figure 9.14. The limiting geometry for bench blasting.

traveled a shorter distance. For a linear charge of length 0.3B the total charge has a mass of $0.3 \times 1 \times$ B. From the curve it is seen that this has the same effect at the toe as a spherical charge placed directly at the toe elevation with a mass $0.3 \times 1 \times$ B. For charges shorter than 0.3B this relationship holds as well, i.e. the elongated charge of a given weight has the same effect at the toe as a spherical charge of the same weight. For elongated charges with lengths greater than 0.3B, the effect at the toe diminishes rapidly with increasing length. The same effect could be achieved by considering the elongated charge extending from the toe elevation downward. Thus an elongated charge extending from 0.3B below the toe to 0.3B above the toe elevation(for a total explosive weight of $0.6 \times 1 \times$ B) would, according to the curve have the same breaking capacity as a spherical charge with a weight of 0.6×1 B placed directly at the toe elevation.

In transferring this concept to the collar region one finds that

$$T_C \leq 0.6\text{B} \tag{9.38}$$

Comparing equations (9.33) and (9.38) one finds that

$$K_{TC} \leq 0.6 \tag{9.39}$$

Substituting equation (9.39) into equation (9.37) yields

$$K_T \geq 1 - \frac{0.6}{2} = 0.7$$

Thus

$$K_T \geq 0.7 \tag{9.40}$$

To this point in the discussion there has been no specific mention of the bench height. If one continues to increase the scale (hole diameter) as shown in Fig. 9.13, the center of charge progresses further and further down the hole.

The limiting condition is when the center of charge reaches the toe elevation (Fig. 9.14). This occurs for a hole diameter which yields a burden just equal to the bench height.

The fifth and last of the fundamental relationships is

$$H = K_H B \tag{9.41}$$

where

K_H = constant relating bench height to the burden

The value of K_H is

$$K_H \geq 1 \tag{9.42}$$

For most open pit operations today K_H is between 1.5 and 2. Combining equations (9.41), (9.42) and (9.26) one finds that

$$H \geq K_B D \tag{9.43}$$

Rearranging equation (9.43) yields

$$D \leq \frac{H}{K_B} \tag{9.44}$$

which provides a rule of thumb for limiting the choice of hole diameter.

9.2.9 *Ratios for initial design*

In the previous section the following five relationships were developed for preliminary blast design

Relationship 1: Spacing-Burden

$$S = K_S B$$

Relationship 2: Burden-Diameter

$$B = K_B D$$

Relationship 3: Subdrill-Burden

$$J = K_J B$$

Relationship 4: Stemming-Burden

$$T = K_T B$$

Relationship 5: Bench height-Burden

$$H = K_H B$$

In this section numerical values for the ratios K_S, K_B, K_J, K_T, and K_H will be presented for use during initial design.

Ratio K_S

As will be discussed more fully in the following section, the ratio of the as-drilled spacing and burden is based upon energy coverage of the bench. For a square pattern, the best energy coverage is achieved with $K_S = 1$ although there isn't too much difference when K_S is varied between $K_S = 1$ to $K_S = 1.5$. For a staggered drilling pattern, the best energy coverage is with $Ks = 1.15$. The efficiency of coverage is not substantially different for $K_S = 1.0$ to 1.5. A staggered pattern yields a much better uniformity of energy coverage than does a square one.

Ratio K_B
In section 9.2.11 a detailed examination of this factor is presented. In brief, it has been found that $K_B \approx 25$ when using ANFO ($\rho = 0.80\,g/cm^3$, $S_{ANFO} = 1$) in rock of medium density ($\rho_R = 2.65\,g/cm^3$). When using other explosives

ρ_e = explosive density

explosive weight strength = S_{ANFO}

in rock of medium density one can use, as a first approximation,

$$K_B = 25\sqrt{\frac{\rho_e \cdot S_{ANFO}}{0.8 \cdot 1}} \tag{9.45}$$

If for example the explosive has a density of $1.2\,g/cm^3$ and a weight strength relative to ANFO of 1.1, the appropriate K_B becomes

$$K_B = 25\sqrt{\frac{1.2}{0.8}\left(\frac{1.1}{1}\right)} = 32.3$$

Ratio K_J
The most common value of K_J is 0.3. In certain sedimentary deposits with a parting plane at toe elevation sub-drilling may not be required. In very hard toe situations, the sub-drilling may be increased over that indicated by using $K_J = 0.3$. However it is probably better to consider using a more energetic explosive. It must be remembered that the sub-drill region generally forms the future crest/bench top for the bench below. Unwanted damage done at this stage may have a long and costly life. In addition, excessive sub-drill results in

1. A waste of drilling and blasting expenditures
2. An increase in ground vibrations
3. Undesirable shattering of the bench floor. This in turn creates drilling problems, abandoned blastholes and deviations for the bench below.
4. It accentuates vertical movement in the blast. This increases the chances for cutoffs (misfires) and overbreak.

Ratio K_T
The minimum recommended value for K_T for large hole production blasting is $K_T = 0.7$. Some specialists suggest the use of $K_T = 1.0$. Placing the charge too close to the collar can result in backbreak, flyrock and early release of the explosive gases with resulting poor fragmentation. On the other hand, increasing the length of stemming may reduce the energy concentration in the collar region to the point where large boulders result.

Ratio K_H
Currently most open pit operations have K_H values which are of the order of 1.6 or more. In some operations the burden is of the same order as the bench height ($K_H = 1$) which means that the blasting is similar to cratering with two free surfaces.

9.2.10 *Ratio based blast design example*

To illustrate the use of the geometrical relationships developed in Sections 9.2.8 and 9.2.9 assume that the initial design parameters are

- rock = syenite porphyry (SG = 2.6)
- explosive = ANFO ($\rho = 0.8\,g/cm^3$, $S_{ANFO} = 1$)

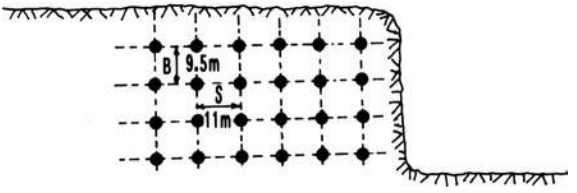

Figure 9.15. Layout for the corner blast.

- bench height (H) = 15 m
- hole diameter (D) = 381 mm (15 ins)
- square drilling pattern, vertical holes
- 4 rows of holes each containing 6 holes make up 1 blast

Using the design relationships, the following results are obtained

$$\begin{aligned} K_B &= 25 \\ B &= 25(0.381) = 9.5\,\text{m} \\ S &= 1.15B = 11\,\text{m} \\ T &= 0.7B = 6.5\,\text{m} \\ J &= 0.3B = 3\,\text{m} \\ L &= H + J = 15\,\text{m} + 3\,\text{m} = 1\,8\text{m} \end{aligned}$$

The value of K_H is calculated to be

$$K_H = \frac{15}{9.5} = 1.6 \text{ (acceptable)}$$

The layout for this corner blast would be as shown in Figure 9.15. The burden (B) and hole spacing (S) dimensions (the pattern to be drilled) have been laid out with respect to the long face.

The volume (V_e) and mass (W_e) of explosive loaded into each hole is given by, respectively

$$V_e = \frac{\pi}{4}D^2(L - T) = \frac{\pi}{4}(0.381)^2(18 - 6.5) = 1.31\,\text{m}^3$$

$$W_e = V_e\rho = 1.31\,\text{m}^3 \times 800\,\text{kg/m}^3 = 1049\,\text{kg}$$

Since there are 24 holes in the round the total amount of explosive required (T_{EXP}) is

$$T_{EXP} = W_e \times n = 24 \times 1049 = 25176\,\text{kg}$$

where

n = number of holes

The volume of rock which will be broken is

$$V_R = nB \times S \times H$$

Thus

$$V_R = 24(9.5)(11)(15) = 37620\,\text{m}^3$$

Using a rock density of 2.6 t/m^3, a total of

$$T_R = \rho_R V_R = 97812 \text{ tons}$$

will be broken.

The resulting powder factor (PF) defined as the amount of explosive required to break one ton of rock is

$$PF_{ANO} = \frac{T_{EXP}}{T_R} = \frac{25176}{97812} = 0.26 \text{ kg/ton}$$

The subscript ANFO has been added to the powder factor designation since it is explosive dependent.

To complete the design, decisions have to be made regarding hole sequencing. There are a large number of factors to be considered when deciding on the sequencing of holes:

- type of fragmentation desired
- surface or in-hole delays
- firing direction
- shape of muck pile/loading equipment
- number of delays available
- type of trunkline system
- environmental constraints (ground vibration/air blast, etc.)

In some cases a maximum delay time is specified to avoid cutoffs between holes. In other cases the minimum amount of time between holes or rows of holes to achieve the best fragmentation may control. Environmental constraints may determine the maximum number of holes which can be shot on the same delay.

It will be assumed that

- there are no environmental restrictions on the number of holes to be shot/delay
- detonating cord trunk lines with surface delays will be used
- the firing direction will be at 45° to the short face (V1 design)
- to avoid cutoffs, a maximum surface delay of 3.3 ms/m (1 ms/ft) can be used
- to minimize the possibility of a misfired round, there should be 2 detonating cord routes to each hole
- a minimum number of delays are to be used.

The detonating cord hookup is shown in Figure 9.16.

Detonating cord of strength 5 g/m is selected. As can be seen, because of the initiation pattern, the effective burden (B_e) is less than the drilled burden (B)

$$B_e = \frac{B}{\sqrt{2}} = \frac{9.5}{\sqrt{2}} = 6.7 \text{ m}$$

The effective spacing on the other hand increases to

$$S_e = S\sqrt{2} = 9.5\sqrt{2} = 13.4 \text{ m}$$

Using the effective burden dimension of $B_e = 6.7$ m, one finds that the maximum recommended surface delay between rows is

$$\text{Delay} = 6.7 \text{ m} \times 3.3 \text{ ms/m} = 22 \text{ ms}$$

A standard period #1 millisecond delay from some suppliers is

$$\text{D}_1 = 30 \text{ ms}$$

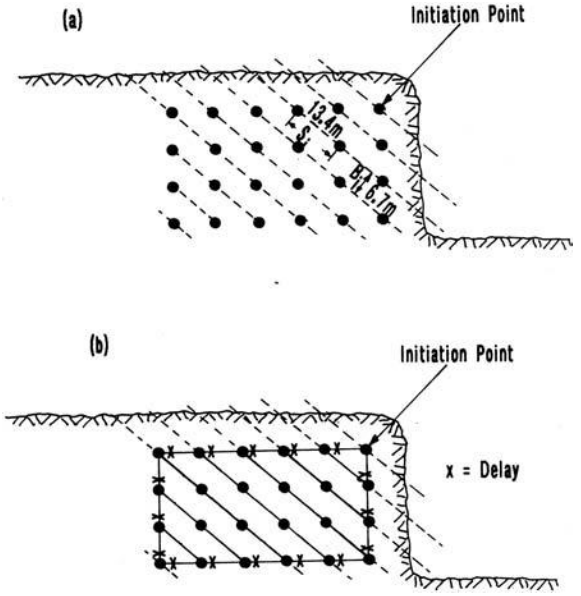

Figure 9.16. Bench round shot in a V1 pattern.

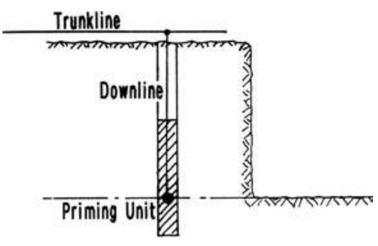

Figure 9.17. Cross-section showing the charging and initiating design for a single hole.

For others it would be

$$D_1 = 25\,\text{ms}$$

Both are somewhat greater than desired although the 25 ms delay could probably be successfully used.

Each hole has a detonating cord downline which will be tied into the trunkline. Tied to the bottom of the downline is a cast 1 lb (454 g) booster. The downline has a strength of 10 g/m. This is strong enough to initiate the booster but weak enough so that the explosive column is not initiated by the cord itself. The cast booster is located at the toe elevation (See Fig. 9.17).

According to Langefors and Kihlström (1978), for burdens between 0.5 and 8 m there is a linear relationship between the delay time (τ) which yields the best fragmentation and the burden. This is expressed as

$$\tau = K_{DT}B \tag{9.46}$$

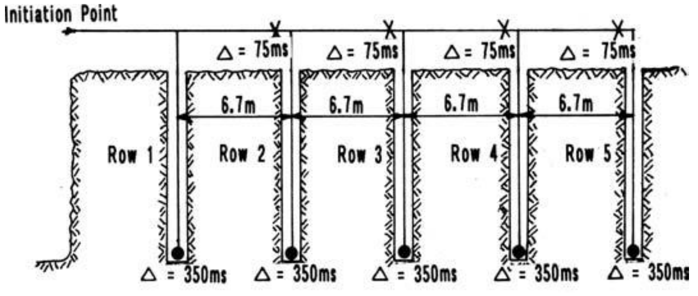

Figure 9.18. Combination of in-hole and surface delays.

where
τ = delay time (ms)
K_{DT} = constant = 3 to 5 ms/m
B = burden (m)

Using this rule, the time delay between rows from a fragmentation viewpoint is of the order of magnitude of

$$\tau = 20 \text{ to } 33\,\text{ms}$$

In theory, only one delay number (25 ms) would be required for the round. However with so many holes involved, the last holes to fire could be quite heavily choked (due to the decreasing free forward movement with row number) when using such a delay. If sufficient time (delay) is not provided between holes there will be no place for the muck to move. This will result in

- hard, high bottom
- excessive toe on the next shot
- a great deal of ragged cratering and attendant flyrock in the upper region of the borehole

Today, the tendency is to use longer delays (3 to 5 ms/ft of burden) to permit sufficient movement of the rock in earlier rows so that a free surface is provided for the hole(s) in question. If this rule is used, then one obtains

$$T_{\min} = 66 \text{ to } 110\,\text{ms}$$

This cannot be accomplished with surface delays alone and a combination of surface and in-hole delays is required. The surface delays are selected to provide the desired rock breakage and movement. The bottom-hole delay placed in each hole is selected so that the desired number of hole rows is energized before the first hole detonates. Assume in this case that the surface delay between rows is selected to be 75 ms and that 5 rows of holes are to be energized prior to the detonation of the first hole. To account for cap scatter and delays introduced by the surface lines, a 350 ms bottom hole delay has been selected. This delay is placed in the bottom of each hole as the holes are loaded with explosive. This is convenient for the chargers since it is the same delay number in each hole. The surface delays are then added during the final tie-in. To allow as much flexibility in the surface tie-in as possible, the initial bottom hole delay should be large. The delay pattern is shown in Figure 9.18.

Although detonating cord has been used in this example, it is more common today to use non-electric trunk lines and down lines. Because the detonation velocity of the non-electric

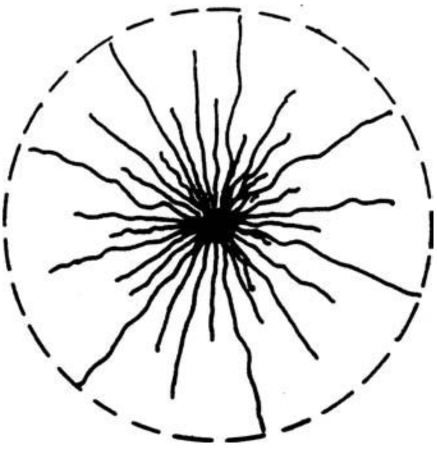

Figure 9.19. An idealized representation of the radial cracking surrounding a single hole.

lines is approximately half that of detonating cord, an additional time delay is introduced and should be included in the calculations. The use of electronic delays eliminates the need to include detonating cap scatter in the calculations and offers considerable timing flexibility.

9.2.11 *Determination of K_B*

Figure 9.19 shows the type of radial cracking which one might expect when blasting a single hole in a brittle, massive rock formation. There will be relatively few long cracks (6–8) spaced uniformly around the hole. As one approaches the hole the cracks will be shorter and more numerous.

The maximum length (R_e) of the radial cracks for a given explosive and rock type can be shown to be directly dependent on the hole radius. Thus as the hole diameter is increased from 150 mm to 310 mm the length of the longest cracks would be expected to about double. This is consistent with the design relationship

$$B = K_B D$$

presented earlier since the burden should be related to the lengths of the cracks generated

$$B \propto R_e \tag{9.47}$$

If the strength of the explosive used in the hole of a given diameter is increased or decreased, the outer crack radius should change accordingly. This is reflected in the value of K_B chosen.

As indicated, the key dimensions required in the development of a blast design are based upon the burden which, in turn, is related to the borehole diameter through the burden factor K_B.

$$B = K_B D \tag{9.26}$$

The value for K_B

$$K_B = 25$$

has been found by the present author and others (Ash, 1963), for example) to work well for a wide range of hole diameters when using ANFO in rocks of medium density (SG = 2.65).

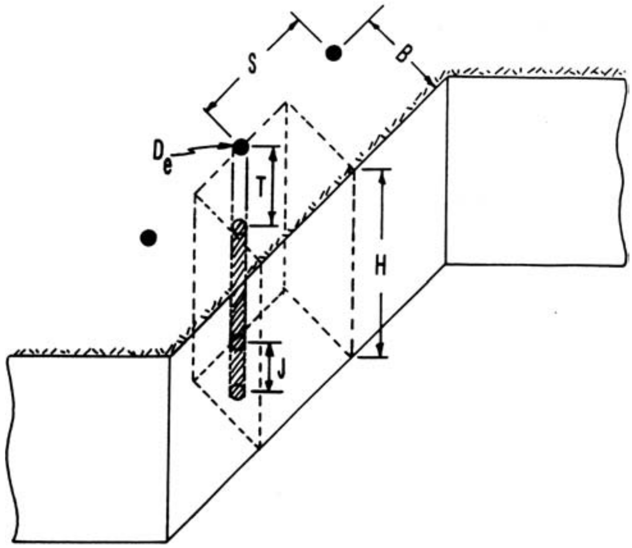

Figure 9.20. The rock volume associated with a blast hole.

Some guidance regarding the selection of K_B when using explosives in rocks of other densities is needed. The approach described in this section is proposed as a first approximation. The development of the basic equation for K_B will be first done using units of the metric system and then the equivalent formulae in the English system will simply be stated.

In addition to those parameters already introduced the following are needed.

SG_E = specific gravity of the explosive
SG_R = specific gravity of the rock
PF_{EXP} = powder factor (kg/ton)
TF = tonnage factor (m^3/ton)

The basic geometry is shown in Figure 9.20 where one blasthole from the round has been isolated.

The number of tons (T_R) broken is given by

$$T_R = K_S K_H B^2 \times SG_R \times \rho_{H_2O} \tag{9.48}$$

where

B = burden (m)
ρ_{H_2O} = density of water (mt/m^3)

Since in the metric system

$$\rho_{H_2O} = 1\,mt/m^3$$

this term will not be carried through the remaining equations. Knowing the powder factor required to provide the desired degree of fragmentation (PF_{EXP}), the amount of explosive required (E_{RQD}) is

$$E_{RQD} = T_R \times PF_{EXP} = K_S K_H B^2 \times SG_R \times PF_{EXP} \tag{9.49}$$

The total amount of explosive available (E_{AVL}) is

$$E_{AVL} = \frac{\pi}{4}(D_e)^2[K_H B + K_J B - K_T B]\, SG_{EXP} \tag{9.50}$$

where

D_e = explosive diameter (m)

Setting the amount of explosive required to that available yields

$$SG_R\, K_S\, K_H\, B^3\, PF_{EXP} = B\frac{\pi}{4}(D_e)^2(K_H + K_J - K_T)SG_{EXP} \tag{9.51}$$

Solving equation (9.51) for B one finds that

$$B = D_e\left[\left(\frac{\pi}{4}\right)\left(\frac{SG_E}{SG_R}\right)\left(\frac{1}{PF_{EXP}}\right)\left(\frac{K_H + K_J - K_T}{K_H K_S}\right)\right]^{1/2} \tag{9.52}$$

As can be seen by comparing equation (9.52) to equation (9.26), K_B is equal to

$$K_B = \left[\left(\frac{\pi}{4}\right)\left(\frac{SG_E}{SG_R}\right)\left(\frac{1}{PF_{ANFO}}\right)\left(\frac{K_H + K_J - -K_T}{K_H K_S}\right)\right]^{1/2} \tag{9.53}$$

The powder factor based on the actual explosive used (PF_{EXP}) will be replaced in equation (9.53) by the equivalent ANFO powder factor (PF_{ANFO})

$$PF_{EXP} = \frac{PF_{ANFO}}{S_{ANFO}} \tag{9.54}$$

where

S_{ANFO} = relative weight strength of the explosive EXP to ANFO

Equation (9.53) then becomes

$$K_B = \left[\left(\frac{\pi}{4}\right)\left(\frac{SG_E}{SG_R}\right)\left(\frac{S_{ANFO}}{PF_{ANFO}}\right)\left(\frac{K_H + K_J - K_T}{K_H K_S}\right)\right]^{1/2} \tag{9.55}$$

This is quite a powerful formula. One of the major ways that the equation can be used is to study the effect of changes in the explosive on the blasting pattern while keeping the other factors of the design constant. It can also be used to evaluate the effect of changing other variables. Rock density is one such parameter of interest.

9.2.12 *Energy coverage*

In examining the best type of "as-drilled" and "as-initiated" patterns to be used, consideration must be given to effective energy coverage of the volume to be fragmented and then selection of the initiation geometry to make best use of the energy. The concept of cylindrical fragmented plugs of rock around each charge (in the absence of a free surface) is a useful tool in this regard. One considers the rock influenced by each blast hole to be bounded by

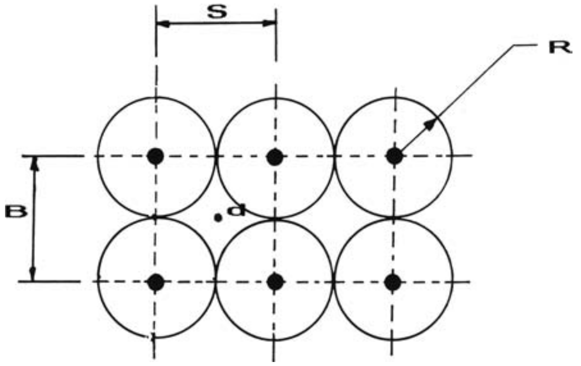

Figure 9.21. Square layout with the hole influence regions touching (S/B = 1).

a cylinder of influence radius R. Rock lying outside of this radius will also be affected but in a very minor way. Figure 9.21 is a view looking down on a bench for which the "plugs" have been drawn.

In this particular square design the influence radii just touch in both the burden and spacing directions. Thus

$$S = 2R \tag{9.56}$$

$$B = 2R \tag{9.57}$$

and hence

$$S = B$$

The plan area of bench assigned per hole by this layout is A_H

$$A_H = B \times S = B^2 \tag{9.58}$$

and the plan area (A_r) per hole which is influenced by the explosive is

$$A_r = \pi R^2 = \frac{\pi B^2}{4} \tag{9.59}$$

Thus the percent of the plan area assigned to the hole influenced by the charge (% I) is

$$\%I = \frac{100A_r}{A_H} = \frac{100\pi}{4} = 78.5\% \tag{9.60}$$

The staggered hole layout shown in Figure 9.22 is an alternative.

The hole spacing (S), the burden (B) and the influence radius (R) have all remained the same. The only change is that the rows have been translated by a distance R from their positions in Figure 9.21. The area assigned (A_H), the area of influence (A_I) and the percent area influenced (% I) are the same as for the square layout. However, the fragmentation results are often better with the staggered pattern. The reason for this is that even though the total "non-influenced" or "un-touched" area is the same in both cases, for the staggered pattern the "un-touched" area is broken down into two smaller areas rather than one larger area (compare Figs 9.21 and 9.22). For the square pattern, the distance (d_o) from the nearest hole to the center of the untouched region is

$$d_o = B\sqrt{2} = R\sqrt{2} = 1.414R \tag{9.61}$$

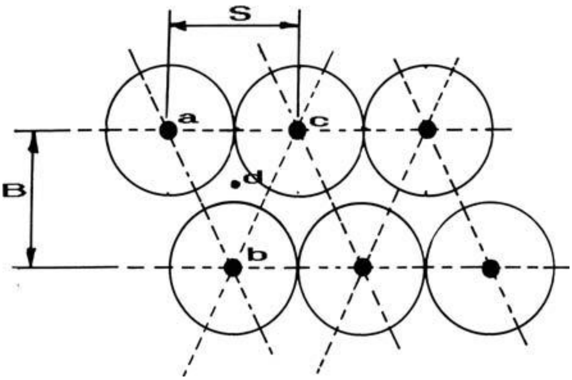

Figure 9.22. Staggered layout with the hole influence regions touching (S/B = 1).

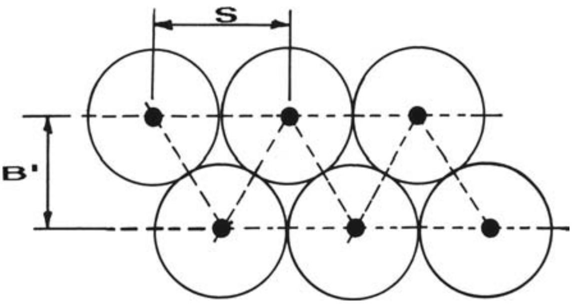

Figure 9.23. Staggered layout with the row influence regions touching (S/B′ = 1.15).

For the staggered pattern (Fig. 9.22), the figure abc constructed by connecting the nearest holes is an isoceles triangle. The sides ac and bc are of equal length. The point *d* which is equi-distant from each corner lies along the altitude line. The distance from the three corners is

$$d_n^* = 1.2R \tag{9.62}$$

In summary, even though the percent energy coverage has not changed with this staggered design, the un-touched area has been redistributed into two smaller pieces and the maximum distance from any charge has been reduced from 1.414R to 1.2R. Thus the fragmentation is expected to be better.

In reviewing the layout in Figure 9.22 it is obvious that the easiest way of reducing the untouched area is to reduce the burden dimension while keeping the spacing constant. In Figure 9.23 the burden has been reduced until the radii of influence are just touching.

The center-to-center spacing to all adjacent charges is now equal to 2R. The triangle abc formed is an equilateral triangle with included angles of 60°. The length (L_A) of the altitude of such a triangle is

$$L_A = \sqrt{3}R \tag{9.63}$$

which is just equal to the burden (B) dimension

$$B' = \sqrt{3}R \tag{9.64}$$

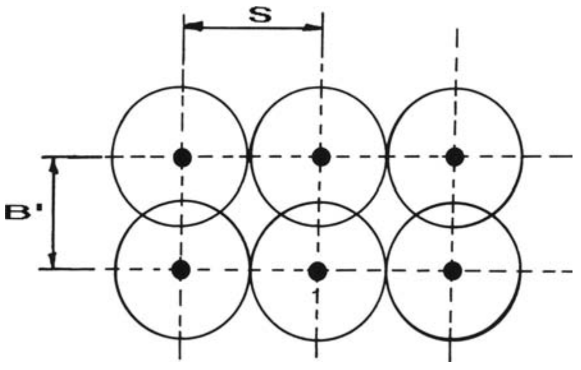

Figure 9.24. The square pattern ($S/B' = 1.15$) with contact in the row direction and overlap in the burden direction.

Since the spacing is

$$S = 2R$$

the burden to spacing ratio is

$$\frac{B'}{S} = \frac{\sqrt{3}R}{2R} = 0.866 \tag{9.65}$$

The inverse of equation (9.65), the spacing to burden ratio, which is normally presented is

$$\frac{S}{B'} = \frac{2}{\sqrt{3}} = 1.155 \tag{9.66}$$

The new bench area assigned to each hole with this layout is

$$A_{H'} = 2R(\sqrt{3}R) = 2\sqrt{3}R^2 \tag{9.67}$$

The area influenced by the explosive is

$$A_r = \pi R^2 \tag{9.68}$$

and hence the percent coverage is

$$\text{I\%} = \frac{A_r}{A_{H'}} = \frac{100\pi}{2\sqrt{3}} = 90.7\% \tag{9.69}$$

If the spacing

$$S = 2R$$

and burden

$$B' = \sqrt{3}R \tag{9.70}$$

are maintained but the rows translated with respect to one another to form a square layout, the result is as shown in Figure 9.24.

In the burden direction there is an overlap of the influence circles and in the spacing direction, they just touch. The specific drilling (the number of drill holes per plan area) is the same for both Figures 9.23 and 9.24. Now the area per hole affected by explosive energy can be shown to equal

$$A_R = \left(\frac{2\pi}{3} + \frac{\sqrt{3}}{2}\right)R^2 \tag{9.71}$$

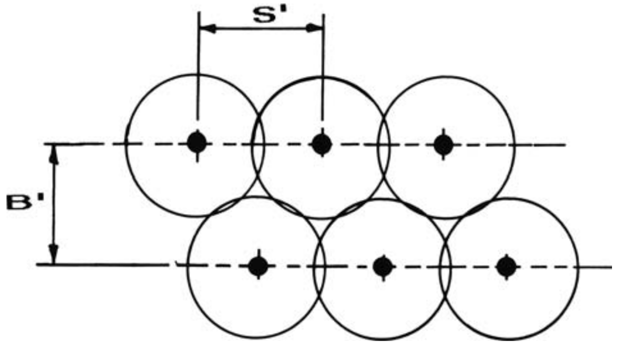

Figure 9.25. Staggered pattern ($S'/B' = 1.0$) with the spacing reduced to achieve total coverage.

while the assigned plan area to the hole is

$$A_H = 2\sqrt{3}R^2 \tag{9.72}$$

Thus the efficiency of the energy coverage is

$$I\% = \frac{A_r}{A_{H'}} = 100\frac{A_r}{A_{H'}} = 85.5\% \tag{9.73}$$

which is substantially less than for the staggered pattern. The distance from the nearest charge to the center of the untouched region is, as before, also greater than with the staggered pattern.

Beginning with the staggered pattern shown in Figure 9.23 one would now like to reduce the untouched area to zero. The condition that there should be no untouched region means that the distance d_n from each corner to the midpoint should be the radius of influence R

$$d_n = R \tag{9.74}$$

Working through the geometry it can be shown that the new hole spacing (S') should be

$$S' = \sqrt{3}R \tag{9.75}$$

This design is shown in Figure 9.25.

There remains an untouched area because the burden has not as yet been adjusted. Maintaining the ideal staggered geometry in which

$$B'' = \frac{\sqrt{3}}{2}S' \tag{9.76}$$

one finds that

$$B'' = \frac{3}{2}R \tag{9.77}$$

Figure 9.26 is the result of reducing the burden from B′ to B″ and the spacing from S to S'.

As can be seen, the region untouched by the explosive charge has now vanished.

$$I\% = 100\%$$

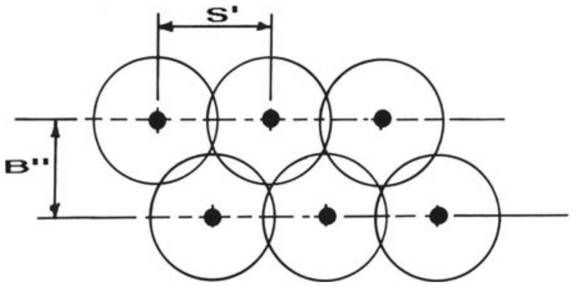

Figure 9.26. Staggered pattern ($S'/B'' = 1.15$) with the burden reduced to achieve total coverage.

Table 9.1. Effect of drilling patterns and S/B ratios on the area covered by the fracture circles. Equilateral triangular layout = 100% (AECI, 1978b).

S/B ratio	Square pattern %	Staggered pattern %
1	77	98.5
1.15Δ	76	100
1.25	75	99.5
1.5	71	94.6
2.0	62	77

The layout area for each hole is now

$$A_{H''} = B'' \times S' = \frac{3}{2}\sqrt{3}R^2 = 2.6R^2 \tag{9.78}$$

as opposed to

$$A_H = 4R^2 \tag{9.79}$$

in the original design shown in Figures 9.22 or 9.23. This means that the specific drilling has increased by a factor of 1.54 while increasing the energy coverage from 78.5% to 100%. An alternative to this is to try and increase the radius of influence R by changing to a more energetic explosive.

Table 9.1 presents the relative efficiencies for different burden to spacing patterns. The efficiencies given are relative to the pattern shown in Figure 9.27 which has an efficiency of 100%. When examining patterns and pattern modification one must keep firmly in mind (a) the changes which occur in the specific drilling and (b) the practicality of drilling the pattern as designed.

As can be seen from Table 9.1, the staggered pattern produces a more uniform distribution of the fracture circles and thus more even fragmentation in the rock pile for the same powder factor.

Optimum coverage is obtained with equilateral triangles, however it varies rather little over the range from $S/B = 1$ to $S/B = 1.5$.

9.2.13 *Concluding remarks*

In this chapter, the authors have largely focused on ANFO since it is the most commonly used blasting agent in mining operations today. There are a host of other explosives, particularly pumped emulsions, which are finding favor. In this regard, the interested reader is encouraged to consult "The Blasters' Handbook" (ISEE, 2011) and "Blasting Principles for Open Pit Mining" (Hustrulid, 1999).

REFERENCES AND BIBLIOGRAPHY

Adel, G., Smith, B., Kojovic, T., Thornton, D., & J.M. Richardson. 2005. Mine-to-mill optimization of aggregate production. Paper 05-009. SME Annual Meeting. Feb 28–Mar 2. SLC, UT.

Adel, G., Smith, B., Kojovic, T., Thornton, D., & J.M. Richardson. 2006. Application of mine-to-mill optimization to the aggregate industry. Paper 06-056. SME Annual Meeting. Mar 26–29. St. Louis, MO.

AECI, African Explosives and Chemicals Limited. 1978a. Parameters in surface blasting. Explosives Today. Series 2, No. 11, March.

AECI, African Explosives and Chemicals Limited. 1978b. Blasthole drilling and initiation patterns in surface blasting. Explosives Today. Series 2, No. 12, June.

AECI, African Explosives and Chemicals Limited. 1986. The design of surface blasts. Explosives Today. Series 2, No. 41, March.

Anonymous. 2011. Drill and blast – speaking the same language. Mining Magazine. April. pp. 34–36.

Ash, R.L. 1963a. The mechanics of rock breakage (Part 1). Pit and Quarry. 56(2): 98–100.

Ash, R.L. 1963b. The mechanics of rock breakage (Part 2) – Standards or blasting design. Pit and Quarry. 56(3): 119–122.

Ash, R.L. 1963c. The mechanics of rock breakage (Part 3) – Characteristics of explosives. Pit and Quarry. 56(4): 126–131.

Ash, R.L. 1963d. The mechanics of rock breakage (Part 4) – Material properties, powder factor, blasting cost. Pit and Quarry. 56(5): 109–118.

Ash, R.L. 1968. The design of blasting rounds. In Surface Mining (E.P. Pfleider, editor). AIME, New York. pp. 373–397.

Ash, R.L. 1992. Section 6.2.2 Design of Blast Rounds. In Surface Mining, 2nd Edition (B. Kennedy, editor). SME, Port City Press, Baltimore. pp. 565–583.

Atlas Copco. 2011. Blasthole Drilling for Open Pit Mining. 2nd Edition.

Atlas Powder Company. 1987. Explosives and Rock Blasting. Maple Press. 662 pp.

Badroddin, M., Khoshrou, H., and A. Siamaki. 2011. Prediction of burden at the Sungun Copper mine by artificial neural network. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 877–886.

Banda, R., & N. Rhodes. 2005. Electronic delay detonators-a unique solution to pertinent mining problems. JSAIMM. 105(9): 615–618.

Batten, C. 2009. Orica – Technology to Kick off a Career with a Bang. AusIMM New Leaders Conference. Brisbane, QLD. April 29–30. pp. 29–32.

Bedell, G. 2012. Personal communication. April.

Bennett, J.W., Askraba, S., and P. Mackenzie. 2005. A New Method to Characterise Black Shales in Pilbara Iron Mining Operations, Iron Ore 2005 Conference, Proceedings, AusIMM Publication Series 8/2005. Australasian Institute of Mining and Metallurgy, Carlton, Victoria, pp. 179–182.

Berry, L. 2011. Managing the drill and blast process. Paper 11-082. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Bhushan, V., and P. Grouhel. 2007. Emerging Blasting Technology and Blast-Based Services, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 19–24.

Bilodeau, M., Labrie, D., Bolsclair, M., Beaudoin, R., Roy, D., & G. Caron. 2007. Impact of electronic detonators on downstream operations of a quarry. Paper 07-041. SME Annual Meeting. Feb 25–28. Denver, CO.

Boshoff, D., & R.C.W. Webber-Youngman. 2011. Testing stemming performance, possible or not? SAIMM. 111(12): 871–874.

Cameron, A., Corkery, D., MacDonald, G., Forsyth, B., and T. Gong. 2007. An Investigation of Ammonium Nitrate Loss to Mine Discharge Water at Diavik Diamond Mines, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 21–33.

Carter, R.A. 2008. Explosive situations. E/MJ. 209(6): 44–48. July/August.

Caterpillar. 2012. Caterpillar Performance Handbook. Edition 42.

Chadwick, J. 2005. Surface drilling and blasting: Big bangs. Mining Magazine. March. pp. 18–24.

Clark, G.B. 1987. Principles of Rock Fragmentation. John Wiley and Sons, New York. 610 pp.

Cook, M.A. 1958. The Science of High Explosives. Reinhold Publishing Corporation, New York. 440 pp.

Cook, M.A. 1974. The Science of Industrial Explosives. Graphic Service and Supply, Inc. Salt Lake City, Utah. 449 pp.

E.I. DuPont de Nemours and Co. 1977. DuPont Blasters' Handbook. 175th Anniversary Edition. Wilmington, Delaware. 494 pp.

Ellenbecker, S. 2012. Smaller holes, patterns help control the blast. Mining Engineering. 64(5): 18–22. May.

Firth, I.R., & D.L. Taylor. 2001. Bench blast modeling using numerical simulation and mine planning software. Paper 01-40. SME Annual Meeting. Feb 26–28. Denver, CO.

Fjellsprengningsteknikk. 1974. Anders M. Heltzen, Chief Editor. Teknologisk Forlag, Oslo. 552 pp. In Norwegian.

Gustafsson, R. 1973. Swedish Blasting Technique. SPI Publishing. Gothenburg, Sweden. 327 pp.

Harcus, M. 2011. Drill and blast: On the advance. Mining Magazine. April. pp. 16–33.

Hudaverdi, T., Kulatilake, P., & C. Kuzu. 2011. A new model for blast fragmentation prediction based on multivariate analysis. Paper 11-003. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Hustrulid, W. 1999. Blasting Principles for Open Pit Mining – Volume 1 General Design Concepts. A.A. Balkema. Rotterdam. 382 pp.

ISEE, 2011. Blaster' Handbook 18th Edition. Society of Explosives Engineers. Cleveland, Ohio. 1030 pp.

Jimeno, C.L., Jimeno, E.L., and F.J.A. Carcedo. 1995. Drilling and Blasting of Rocks. A.A. Balkema, Rotterdam. 391 pp.

Jimeno, C.L., Jimeno, E.L., and P.G. Bermúdez. 2003. Manual de Perforacion y Voladura de Rocas. U.D. Proyectos. Madrid. 779 pp. In Spanish.

Katsabanis, P.D., Gregersen, S., Kunzel, G., Pollanen, M., Pelley, C., & S. Kelebek. 2005. Effects of blasting on damage and grindability of impacted rock. CIM Magazine. December 2005/January 2006. pp. 446–452.

Kim, K., & J. Kemeny. 2011. Site specific blasting model for mine-to-mill optimization. Paper 11-118. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Komatsu. 2009. Specifications and Application Handbook. Edition 30.

Langefors, U., and B. Kihlström. 1963. The Modern Technique of Rock Blasting. John Wiley and Sons. New York. 405 pp.

Little T.N. 2007. Flyrock Risk, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 35–43.

Lovejoy, C. 2009. Management in action – Blasting technology: The big bang. Mining Magazine. May. pp. 24–26.

Lownds, C.M., & U. Steiner. 2010. Safety of blasting with electronic detonators. Paper 10-015. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.

Mandal, S.K., Singh, M.M., & S. Dasgupta. 2007. Air pressure and noise vis-á-vis detonating fuse. CIM Magazine. May. pp. 95–100.

Moore, P. 2006. Management in action – Surface blasting: Having a blast. Mining Magazine. November. pp. 30–42.

Moore, P. 2007. Equipment – Explosives: Explosives delivery. Mining Magazine. November. pp. 42–51.

Moser, P., Lurf, R., and C. Weber. 2007. Operational Excellence in Rock Blasting – From a European Point of View, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 7–12.

Olofsson, S.O. 1997. Applied Explosives Technology for Construction and Mining. APPLEX, Ärla, Sweden. 304 pp.

Onederra, I., Ruest, M., and G. Chitombo. 2007. Burden Movement Experiments Using the Hybrid Stress Blasting Model (HSBM), AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 177–183.

Ozkahraman, H.T. 2010. Breakage mechanisms and an encouraging correlation between the Bond parameters and the friability value. JSAIMM. 110(3): 153–159.
Pereira, P.R. 2005. Better utilization of drilling and blasting resources to increase mining profitability. Paper 05-114. SME Annual Meeting. Feb 28–Mar 2. SLC, UT.
Persson, P-A., Holmberg, R., and J. Lee. 1994. Rock Blasting and Explosives Engineering. CRC Press, Boca Raton, Florida. 640 pp
Potts, A. 2005. Equipment focus – Drilling and blasting: Boosting the bottom line. Mining Magazine. August. pp. 22–27.
Rantapaa, M., Mckinstry, R., & T. Bolles. 2005. Drill-to-mill: Efficient drilling and blasting resulting in increased mill throughput at Barrick Goldstrike. CIM Magazine. January/February. pp. 160–162.
Refahi, A., Aghazadeh Mohandesi, J., & B. Rezai. 2009. Comparison between bond crushing energy and fracture energy of rocks in a jaw crusher using numerical simulation. JSAIMM. 109(12): 709–717.
Richards, A.B., and A.J. Moore. 2007. Effect of Blasting on Infrastructure, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 45–50.
Roy, D., Beaudoin, R., Bilodeau, M., Labrie, D., and G. Caron. 2007. Impact of Electronic Blasting Detonators on Rock Quarry Environment, Productivity and Energy Savings, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 135–146.
Schwengler, B., Moncrieff, J., and P. Bellairs. Reduction of the Blast Exclusion Zone at the Black Star Open Cut Mine, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 51–58.
Singh, R.K., Sawmiliana, C., & P. Pal Roy. 2007. Environmental impact of blast-emitted noise and air overpressure: prediction and control measures. CIM Magazine. May. pp. 90–94.
Tamrock. 1989. Surface Drilling and Blasting. 477 pp.
Thornton, D. 2009. The Implications of Blast-Induced Movement to Grade Control. Seventh Int, Mining Geology Conf. Perth, WA. August 17–19. pp. 147–154.
Thornton, D. 2011. Blast movement modelling and measurement. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 397–309.
Thornton, D. 2011. Blast movement monitoring – not just for grade control. AusIMM Journal. Issue 6. Dec.
Wright, D. 2008. The Use of Electronic Detonators as a Key to Reducing Overall Mining Costs, AUSIMM Conference Proceedings, The AusIMM New Leaders' Conference Wollongong, NSW, July 2008, pp. 67–71.
Yang, R., and D.S. Scovira. 2007. A Model for Near-Field Blast Vibration Based on Signal Broadening and Amplitude Attenuation, AUSIMM Conference Proceedings, EXPLO Conference, Wollongong, NSW, September 2007, pp. 59–66.

Blasting References Especially Selected from the Proceedings, Annual Conference on Explosives and Blasting Technique, ISEE, Cleveland, Ohio. www.isee.org.

Arellano, M. & J. Delgado. 2001. Bench increase angle. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.
Arellano, M., E. Poulakidas & J. Flores. 2008. Efficient use of energy to wall control. Proceedings, 34th Annual Conference on Explosives and Blasting Technique, Jan. 27–30. New Orleans, LS.
Christopherson, K. & V. de Viterbo. 2011. Trim blasting designs in the Morenci mine. Proceedings, 37th Annual Conference on Explosives and Blasting Technique, Feb. 6–9. San Diego, CA.
Cunningham, C. 2002. Defining detonation energy for practical blasting. Proceedings, 28th Annual Conference on Explosives and Blasting Technique, Feb. 10–13. Las Vegas, NV.
Eloranta, J. 2001. Improved milling through better powder distribution. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.
Eloranta, J. 2002. The role of blast operations in metal mining. Proceedings, 28th Annual Conference on Explosives and Blasting Technique, Feb. 10–13. Las Vegas, NV.
Everett, W. & J. Eloranta. 2006. Digital delay blasting at United Taconite LLC. Proceedings, 32nd Annual Conference on Explosives and Blasting Technique, Jan. 29–Feb. 1. Dallas, TX.
Favreau, R. & P. Favreau. 2002. How to design a blast with computer simulations. Proceedings, 28th Annual Conference on Explosives and Blasting Technique, Feb. 10–13. Las Vegas, NV.

Favreau, R. & P. Favreau. 2012. Evaluation of the fragmentation size distribution resulting from blasting as derived from basic principles. Proceedings, 38th Annual Conference on Explosives and Blasting Technique, Feb 12–15. Nashville, TN.

Hendricks, C. 2002. The application of GPS positioning and drill monitoring technology. Proceedings, 28th Annual Conference on Explosives and Blasting Technique, Feb. 10–13. Las Vegas, NV.

Hunter, M. 2003. Computerized drill for quarrying automates the drilling process. Proceedings, 29th Annual Conference on Explosives and Blasting Technique, Feb. 2–5. Nashville, TN.

Hustrulid, W. 2007. A practical, yet technically sound, design procedure for pre-split blasts. Proceedings, 33rd Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Nashville, TN.

Martin, P. 2004. Drill and blast optimization at the Sparkhule limestone quarry. Proceedings, 30th Annual Conference on Explosives and Blasting Technique, Feb. 1–4. New Orleans, LS.

Palangio, T. & J. Eloranta. 2007. Size matters in the Mesabi Range. Proceedings, 33rd Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Nashville, TN.

Paley, N. & T. Kojovic. 2001. Adjusting blasting to increase SAG mill throughput at the Red Dog mine. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.

Paley, N. 2010. Testing electronic detonators to increase SAG mill throughput at the Red Dog mine. Proceedings, 36th Annual Conference on Explosives and Blasting Technique, Feb. 7–10. Orlando, FL.

Paley, N. 2012. Fragmentation from pattern expansion with electronic detonators at the Red Dog mine. Proceedings, 38th Annual Conference on Explosives and Blasting Technique, Feb. 12–15. Nashville, TN.

Paterson, L. 2000. Drilling efficiency and the associated benefits to the mining process. Proceedings, 26th Annual Conference on Explosives and Blasting Technique, Feb. 13–16. Anaheim, CA.

Paventi, M., W. Lidken & L. Mirabelli. 2001. Mine automation program: Drill and blast analysis. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.

Preece, D. & S. Chung. 2002. Multi-blasthole, multi-row, detonation delay timing simulation of rock blasting using DMCBLAST 3D. Proceedings, 28th Annual Conference on Explosives and Blasting Technique, Feb. 10–13. Las Vegas, NV.

Preece, D. 2008. 3D computer simulation of bench blasting with precise delay timing. Proceedings, 34th Annual Conference on Explosives and Blasting Technique, Jan. 27–30. New Orleans, LS.

Proulx, R. & R. Pinksen. 2011. Oversize reduction project at the Iron Ore Company of Canada. Proceedings, 37th Annual Conference on Explosives and Blasting Technique, Feb. 6–9. San Diego, CA.

Rogers, W., S. Kanchibotla, A. Tordoir, S. Ako, E. Engmann & B. Bisiaux. 2012. Understanding blast movement and its impacts on grade control at Ahafo Gold Mine in Ghana. Proceedings, 38th Annual Conference on Explosives and Blasting Technique, Feb. 12–15. Nashville, TN.

Scott, D. 2012. An energy equality approach to blasting. Proceedings, 38th Annual Conference on Explosives and Blasting Technique, Feb. 12–15. Nashville, TN.

Spathis, A., G. Smith, I. Yacob & A. Labriola. 2001. Wall control at the Freeport Grasberg open cut mine: Vibration and gas penetration measurements as a precursor to improvements. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.

Vynne, J. 2001. Drill monitoring and GPS developments and their impacts on the drill to mill process. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.

Vynne, J, 2007. Drill monitoring & navigation systems at BHP's Navajo mine. Proceedings, 33rd Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Nashville, TN.

Vynne, J. 2011. Drill monitoring systems and the integration with drill & blast software. Proceedings, 37th Annual Conference on Explosives and Blasting Technique, Feb. 6–9. San Diego, CA.

Wall, J. 2012. Perimeter control utilizing electronic detonators. Proceedings, 38th Annual Conference on Explosives and Blasting Technique, Feb. 12–15. Nashville, TN.

Webber, R. & S. Scovira. 2008. Record blast at Iron Ore Company of Canada. Proceedings, 34th Annual Conference on Explosives and Blasting Technique, Jan. 27–30. New Orleans, LS.

Wiggin, M. & L. Logan. 2001. Unlocking the wealth from benefit sharing in drill and blast. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.

Williams, J. & K. Oakes. 2004. Blasting for wall control and stability using electronic detonators at Bald Mountain mine, Nevada. Proceedings, 30th Annual Conference on Explosives and Blasting Technique, Feb. 1–4. New Orleans, LS.

Workman, L. 2001. Potential for the unifying of drilling, blasting and downstream operations by the application of technology. Proceedings, 27th Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Orlando, FL.

Workman, L. 2003. The role of blasting in crushing and grinding efficiency and energy consumption. Proceedings, 29th Annual Conference on Explosives and Blasting Technique, Feb. 2–5. Nashville, TN.
Workman, L. 2007. Important considerations when selecting blasthole drills for high altitude use. Proceedings, 33rd Annual Conference on Explosives and Blasting Technique, Jan. 28–31. Nashville, TN.
Workman, L. & J. Eloranta. 2009. Considerations on the effect of blasting on downstream performance. Proceedings, 35th Annual Conference on Explosives and Blasting Technique, Feb. 8–11. Denver, CO.
Zink, C. 2006. Hole cleaning improvements through tricone bit lug design. Proceedings, 32nd Annual Conference on Explosives and Blasting Technique, Jan. 29–Feb. 1. Dallas, TX.

ACKNOWLEDGEMENTS

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REVIEW QUESTIONS AND EXERCISES

1. Explain why it make sense to begin a discussion of the “Unit Operations” with blasting?
2. What are the “design” components in blasting, drilling, loading and hauling?
3. Discuss the relevance of the “mine to mill” concept.
4. The blast design flowsheet logic has been presented. One begins with the “Design Objectives”. Discuss the significance of each of the items listed. Is the list as presented complete?
5. What is the difference between “deflagration” and “detonation”?
6. The power released by the detonation of an ANFO charge 300 mm in diameter and 8 m long has been determined. Repeat the calculation for a charge 381 mm in diameter and 10 m long. Try to find the power output of a power plant in your vicinity. How does its power output compare to this explosive charge? Conclusion?
7. You have been shown the derivation of a pressure-volume curve for ANFO. The energy released by the explosive is the area under the PV curve. Using the given PV curve, go through the steps to determine the energy. Does your result agree with that given?
8. If the confinement is low, much of the explosive energy is released to the atmosphere rather than creating fractures in the rock. If the release occurs at a pressure of 4500 atm, how much energy has gone into fragmentation? How much goes into heave, air blast, etc.? What is your conclusion?
9. Define the terms:
   - Weight strength
   - Bulk strength
   - Relative weight strength
   - Relative bulk strength
10. There is some variance amongst different groups concerning the weight strength assigned for ANFO. Check the literature from the major explosive suppliers and see what they give. What is the reason for the differences do you think?
11. In the literature there are differences in the values provided for ANFO density and also the velocity of detonation. Why is that? How does it affect you as a designer?

12. The weight strength of an emulsion explosive A is given as 1.12 when compared to ANFO which has a weight strength of 1.00. The specific gravity of explosive A is 1.17 whereas for ANFO the value is 0.80. What is the weight strength of explosive 'A' relative to ANFO?
13. Define the terms:
    - Burden
    - Spacing
    - Subdrill
    - Stemming
14. In Figure 9.6, the bench face is shown as vertical for simplicity. In a typical open pit mine, how might it really appear? How does this affect your design?
15. In some mines, the upper part of the hole, the "stemming" region, is left empty. In other mines it is filled with drill cuttings. In still others, special plugs are inserted. Discuss the differences one might expect in blast performance, if any.
16. Summarize the energy coverage approach to blast design.
17. The proper placement of the detonator/primer in the hole is important. One common rule is that it should be placed at the toe elevation. Why does this make sense?
18. Summarize the Ash design rules. Learn/memorize the standard values for K_B, K_S, K_J, K_T and K_H.
19. How do you expect the fragmentation to depend on the hole diameter? If you wanted to achieve a fine fragmentation in a massive rock, would large or small diameter holes be better? Explain your reasoning using drawings?
20. In problem 19, would your answer change if the rock was jointed? Highly jointed? Explain.
21. In the Ash-based design formulas, the constant K_B is used to relate the burden to the hole diameter. For ANFO in rock with density 2.65 g/cm^3, $K_B = 25$. What value would you use for K_B when explosive 'A' with a relative bulk strength of 1.15 is used? Hint: the suggested formula is

$$K_B\ (\text{explosive 2}) = K_B\ (\text{explosive 1})\sqrt{\frac{\rho \cdot s\ (\text{explosive 2})}{\rho \cdot s\ (\text{explosive 1})}}$$

22. At a particular open pit mine, the bench height is 35 ft, the hole diameter is 9″ and the rock has a specific gravity of 3.0. For ANFO and explosive 'A' determine:
    a. Burden
    b. Spacing
    c. Subdrill
    d. Stemming
    e. Hole length
    f. Charge length
    g. Rock broken/hole (tons)
    h. Explosive factor (lbs/ton)
23. Discuss the difference between "as-drilled" burden and spacing and "as-shot" burden and spacing. Is the explosive energy density affected? What differs?
24. If the following explosive prices apply, calculate the explosive cost/hole and per ton of rock from the results of problem 22.
    - 1 lb cast primer = $4.50 each
    - Reinforced detonation cord = $150/1000 ft

– ANFO (bulk) = \$50/100 lbs
– Explosive A (bulk) = \$70/100 lbs

25. At a surface mine in Canada, P&H rotary drills are used to drill 9″ diameter holes to a depth of 39 ft. The bench height is 33 ft. The burden is 20 ft and the spacing is 22 ft. ANFO (specific gravity of 0.85) is the primary explosive. The explosive factor is 0.50 lbs/ton or 1.08 lbs/bank cubic yard (bcy). Approximately 580 lbs of ANFO are used per hole.
    a. What is the specific gravity of the rock in place?
    b. Using the given data, what is the stemming length?
    c. How does this pattern compare with that suggested using the Ash-based design guidelines?
26. Blasting at a limestone mine is done using 6″ diameter holes. Although the thickness of the layer is 50 ft. thick, the benches are 45 ft high. There is a natural parting at the 45 ft. depth and no subdrilling is done to avoid dilution (the rock is used for cement production. The bench faces are vertical and back break accounts for 10% of the total tonnage. A single row of holes is used with S = 1.25 B. The explosive is ANFO ($K_B = 25$), density = 0.85 g/cm^3). The bank density of the limestone is 2.49 g/cm^3, percent swell = 66%, swell factor = 0.60.
    a. What is the tonnage factor for the rock?
    b. What is the powder factor for this case?
27. If the bench height at a particular mine is 25 ft, what is the maximum recommended hole diameter? The explosive is assumed to be ANFO (SG = 0.8). The rock has a density of 2.4 g/cm^3. Calculate the burden, spacing, subdrill and stemming. Draw a cross-section showing the different dimensions. Calculate the explosive factor.
28. Define the following blasting terms:
    a. Powder factor
    b. Explosive factor
29. The in-place density of the rock at the Milly Dilly copper mine in British Columbia is 2.64 g/cm^3. P&H rotary drills are used for drilling 10-5/8″ diameter blast holes. An emulsion explosive is used. It has a density of 1.2 g/cm^3 and a weight strength with respect to ANFO $\rho = 0.85$ g/cm^3) of 0.87. In the hardest rock, a pattern 24 ft. × 27.6 ft (B × S) is used. The bench height is 40 ft, the subdrill is 5 ft and the stemming is 21 ft. A staggered pattern with S = 1.15 B is used.
    a. Why is a "staggered" rather than a "square" pattern used?
    b. Make plan and section drawings showing the blast pattern.
    c. Calculate the amount of rock attributed to each hole.
    d. Calculate the amount of explosive in each hole.
    e. Calculate the explosive factor (lbs/ton).
    f. What is the value of K_B appropriate for this rock and explosive?
    g. How do the burden, spacing, stemming and subdrill values selected compare with the Ash-based rules?
    h. The cost of the emulsion explosive is about \$75/100 lbs. The cost of the recommended 12 oz primer is \$4.00 and the Nonel downline (50 ft length) with detonator is \$5.00. What would be the explosive cost per ton of ore?
30. A mine in the planning stages is trying to decide whether to use 12-1/4″ or 15″ diameter blastholes. The bench height has been fixed at 40′ and ANFO having of specific gravity of 0.85 will be used. Two rows of holes will be drilled. The rock has a specific gravity

of 2.65. Design the blast patterns for each hole diameter (round off the distance to the nearest foot).

a. Burden
b. Spacing
c. Subdrill
d. Stemming
e. Hole length
f. Loaded length
g. Explosive/hole (lbs)
h. Rock broken/hole (tons)
i. Explosive factor (lbs/ton)
j. Delay time (ms)

31. An explosive has a density of 1.1 g/cm^3 and a weight strength relative to ANFO of 1.08.
    a. What would the value of K_B be when used in rock of medium density?
    b. How should K_B change when used in rock having a density of 3.1 g/cm^3?
32. Discuss the effect of hole diameter and rock structure on fragmentation.
33. Draw figures demonstrating the energy coverage of:
    a. Square pattern, S/B = 1
    b. Square pattern, S/B = 1.15
    c. Square pattern, S/B = 1.25
    d. Staggered pattern, S/B = 1
    e. Staggered pattern, S/B = 1.15
    f. Staggered pattern, S/B = 1.25
34. The blasting agent ANFO (SG = 0.85) is being used as the primary explosive at a large mine. The bench height is 40 ft and the hole diameter is 9-7/8″. A square pattern with B = S is being used. The rock has a bank density of 5200 lbs/yd^3 and a loose density of 3600 lbs/yd^3. The powder factor is 0.68 lbs/ton. The subdrilling is 0.3 B and the stemming is 0.7 B.
    a. Draw a picture of the situation. Assume a vertical bench face. Label the drawing carefully.
    b. Based upon the given powder factor calculate the burden being used.
    c. How does the value in (b) compare with that determined using the Ash-based formula? What might be the reason for this?
    d. An emulsion explosive with a density of 69 lbs/ft^3 and a weight strength of $S_{ANFO} = 0.9$ has been suggested for testing. What would be the first pattern tested and why? What would be the expected result?
35. Much useful information concerning blasting and explosives is available on the Internet. In problem 33 an unidentified explosive with a density of 69 lbs/ft^3 and a relative weight strength of 0.9 has been suggested for testing. What commercial product might this be? Check the two websites: www.oricaminingservices.com and www.dynonobel.com.

CHAPTER 10

Rotary drilling

10.1 BRIEF HISTORY OF ROTARY DRILL BITS

Up until the beginning of the 20th century, the drilling of oil wells was done using bits of various fishtail designs. One such design is shown in Figure 10.1.

As force is applied during bit rotation, the rock is literally scrapped away from the bottom of the hole. As one can imagine, this practice could and did work well in soft formations but in the harder, more abrasive formations the wear was high and the penetration rate low. In 1907, Walter Benona Sharp and Howard R. Hughes, Sr. were involved with drilling two test oil wells at Goose Creek and Pierce Junction, Texas. According to the Handbook of Texas (2012), both wells had to be abandoned because of the hard rock encountered. As a result, the two men began to consider the possibility of developing a roller rock bit. It was eventually arranged for Hughes to proceed with the design and construction of a bit with capital provided equally by Sharp and J.S. Cullinan. The result was the Sharp-Hughes rock bit (see Figs 10.2 through 10.4). In 1908, the Sharp-Hughes Tool Company was formed to manufacture the bit and a factory was built at Houston, Texas. Hughes received 50 percent of the stock for his development of the bit and the other 50 percent was divided between Sharp and Cullinan. In 1912, at the time of Sharp's death, Hughes was given controlling interest. Not long afterward, the company name was changed to the Hughes Tool Company.

As noted by Hughes in his patent application, the bit removes the rock by quite a different process than the old fishtail bit. According to Hughes (1916).

"The edges, or cone points of the bit, roll in a true circle like a cone bearing, and crumble or chip away the rock. The cone points, being of very hard steel, wear away slowly. Often

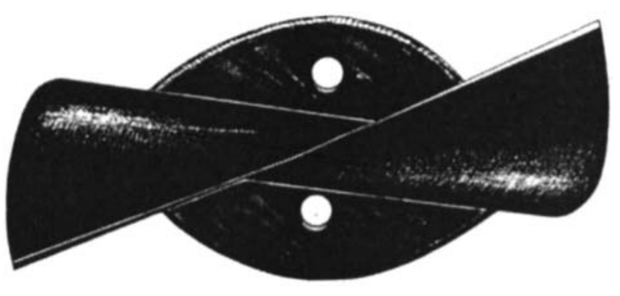


Figure 10.1. A typical fishtail bit design (Hughes, 1916).

Figure 10.2. Frontal view of the Sharp & Hughes rock bit. Hughes (1916).

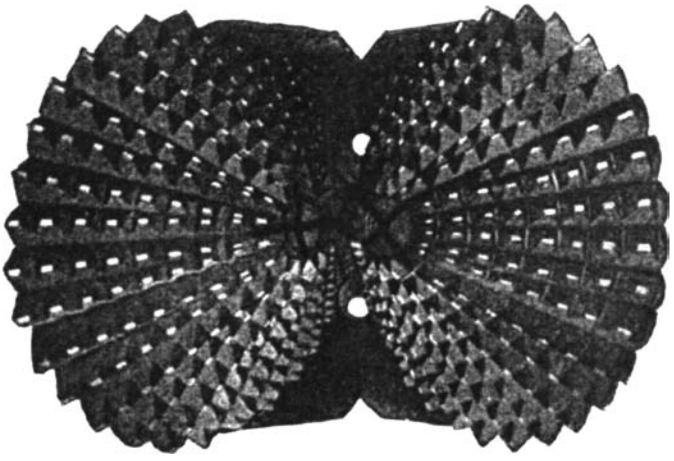


Figure 10.3. Bottom view of the Sharp & Hughes cone bit. Hughes (1916).

they show but slight wear after drilling 50 ft. of rock, a few inches of which would completely dull the ordinary fishtail bit. The rolling motion allows the cutting edges on the cones to chip the rock, one edge after another."

Figure 10.5 shows the Sharp & Hughes cone bit ready for operation.

A number of major improvements have taken place over the years. In 1924, self-cleaning, intermeshing cones were introduced which provided room for larger bearings and greater tooth depth (Fig. 10.6).

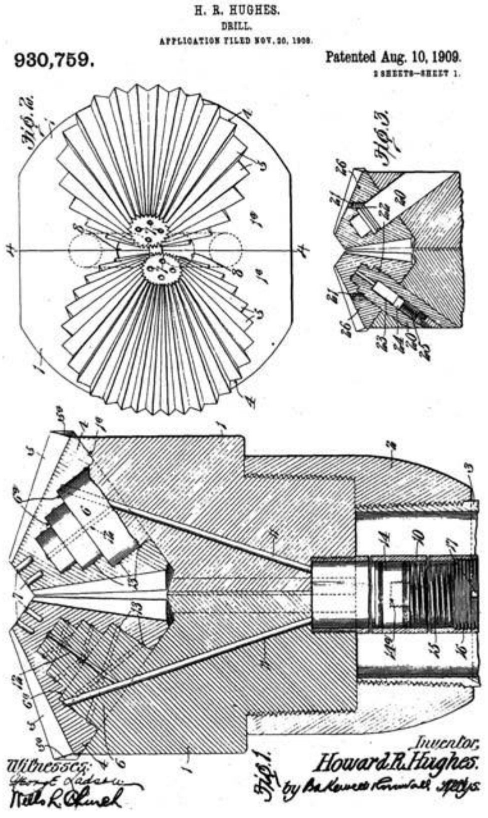


Figure 10.4. Drawings of the rock bit included in U.S. patent 930,759. Hughes (1909).

The introduction of ball and roller antifriction bearings lubricated only by the drilling fluid occurred in 1926. In 1929, bit life was increased by the application of tungsten carbide hard facing to cutter teeth. In 1934, a patent was issued for 3-cone bits (see Fig. 10.7. As noted in the patent application Scott and Garfield, 1934).

"It has heretofore been considered impossible and undesireable to form the teeth on the cutters of a three cone bit so that they may interfit in use, the reason being that it is difficult to make the rows of teeth in one cutter mate with the teeth of two other cutters, the teeth of which must also interfit with each other. We have found that this may be done, and although the rows of teeth of each cutter are necessarily few, by proper arrangement they will together cut the full bottom of the hole and provide a smooth running and fast cutting drill."

Through this advance, the penetration rate and the bit footage essentially doubled.

This patent, in practice, essentially "locked-up" the development of three-cone (tricone) roller cone bits for the Hughes Tool Company until about 1951. In 1948, jet bits by which high velocity streams of drilling fluid were directed against the hole bottom were introduced by the company. This facilitated cuttings removal and transport. In 1949, the use of air as

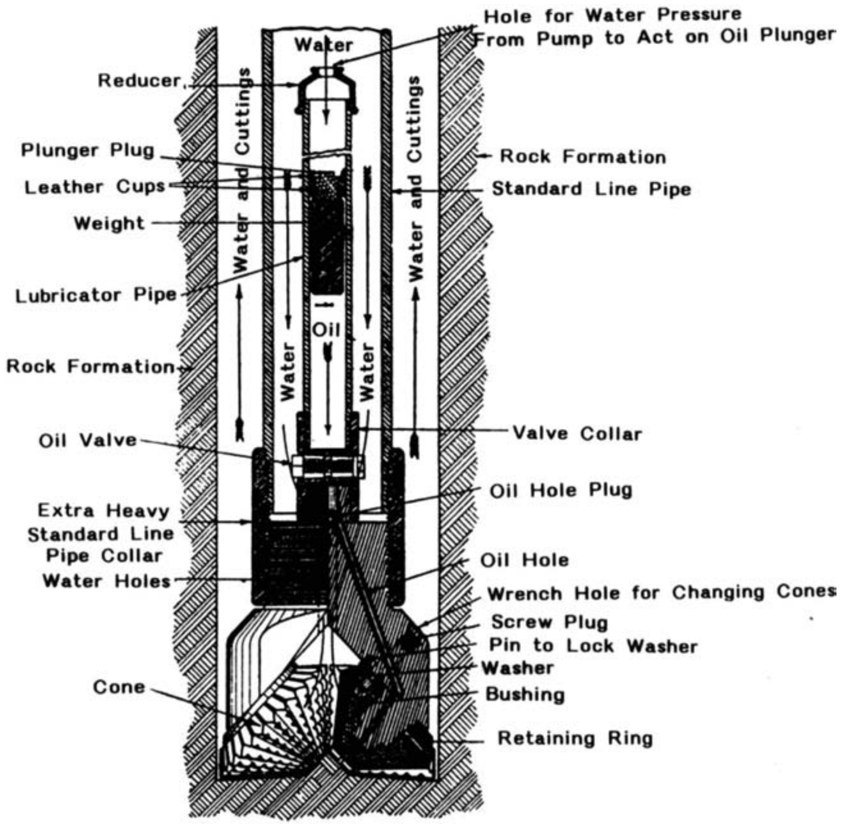


Figure 10.5. The Sharp & Hughes bit ready for operation. Hughes (1916).

the circulating medium for rotary blasthole drilling was introduced. With its introduction, many of the undesirable aspects of water such as

- availability
- freezing
- haulage
- disposal

were eliminated. It was also found that air circulation resulted both in an increased penetration rate and bit life. The use of sintered tungsten carbide as the cutting elements was introduced in 1951. These inserts had the strength and crushing capacity to drill three to ten times more hole than steel tooth bits in hard materials such as chert. Later, in 1959, the first practical sealed, pressure compensated, self-contained system of lubrication for roller bearings was introduced. This was still an inadequate match to the life of the carbide tooth bits. A new journal bearing the life of which matched that of the tungsten carbide cutting structures was introduced in 1969 by the Hughes Tool Company. Bit life was doubled, tripled, or even more. These time-line events were taken from Baker-Hughes (2007). As can be seen, the development process has been one in which cutting element improvements have been followed by bearing improvements, followed by cutting element improvements, etc.

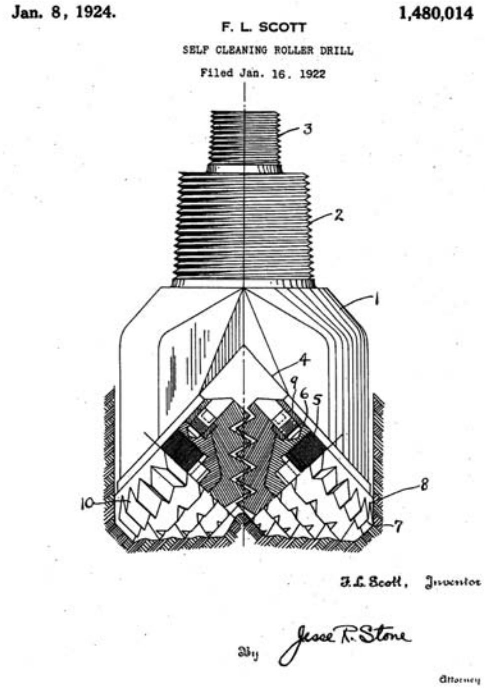


Figure 10.6. Self-cleaning cones as depicted in U.S. patent 1,480,014. Scott (1924).

The history of the use of rotary bits in mining spans a period of almost 70 years. Compact machines designed specifically for rotary blasthole drilling were in use already in 1946. However, because of water circulation systems, their acceptance was very limited. The introduction of sintered tungsten carbide cutting elements in 1951 offered new possibilities for economically cutting even harder rocks. Today rotary bits are used in the hardest formation encountered in mining. Although mining users can still purchase steel tooth cutters, the cutting elements have largely been replaced by sintered tungsten carbide inserts of various shapes and sizes. Figure 10.7 shows a few of the many possible insert designs. The interested reader is also referred to Atlas Copco (2011e).

Figures 10.9–10.11 present pictures showing some modern tricone bits used for blasthole drilling. By comparing Figures 10.9 and 10.10 one can obtain a feeling of how bit design, in this case from the same manufacturer, has changed over a period of about 20 years. By comparing Figure 10.9a to 10.9b and Figure 10.10a to 10.10b one can clearly see the modifications in insert design and pattern needed to accommodate differing rock hardness.

Figure 10.11 clearly shows the required protection for the leg/shirttail of the bit. Figure 10.12 is a top view showing the intermeshing of the cones and inserts to get the proper bottom hole coverage. It is interesting to study this photo with respect to the comments regarding "impossibility" mentioned with respect to the patent of the tricone bit (see Fig. 10.7).

10.2 ROCK REMOVAL ACTION

Figure 10.13 is a diagrammatic view looking from the collar of the hole down on the cutters on the hole bottom. The three cones containing the cutting elements (either steel teeth or

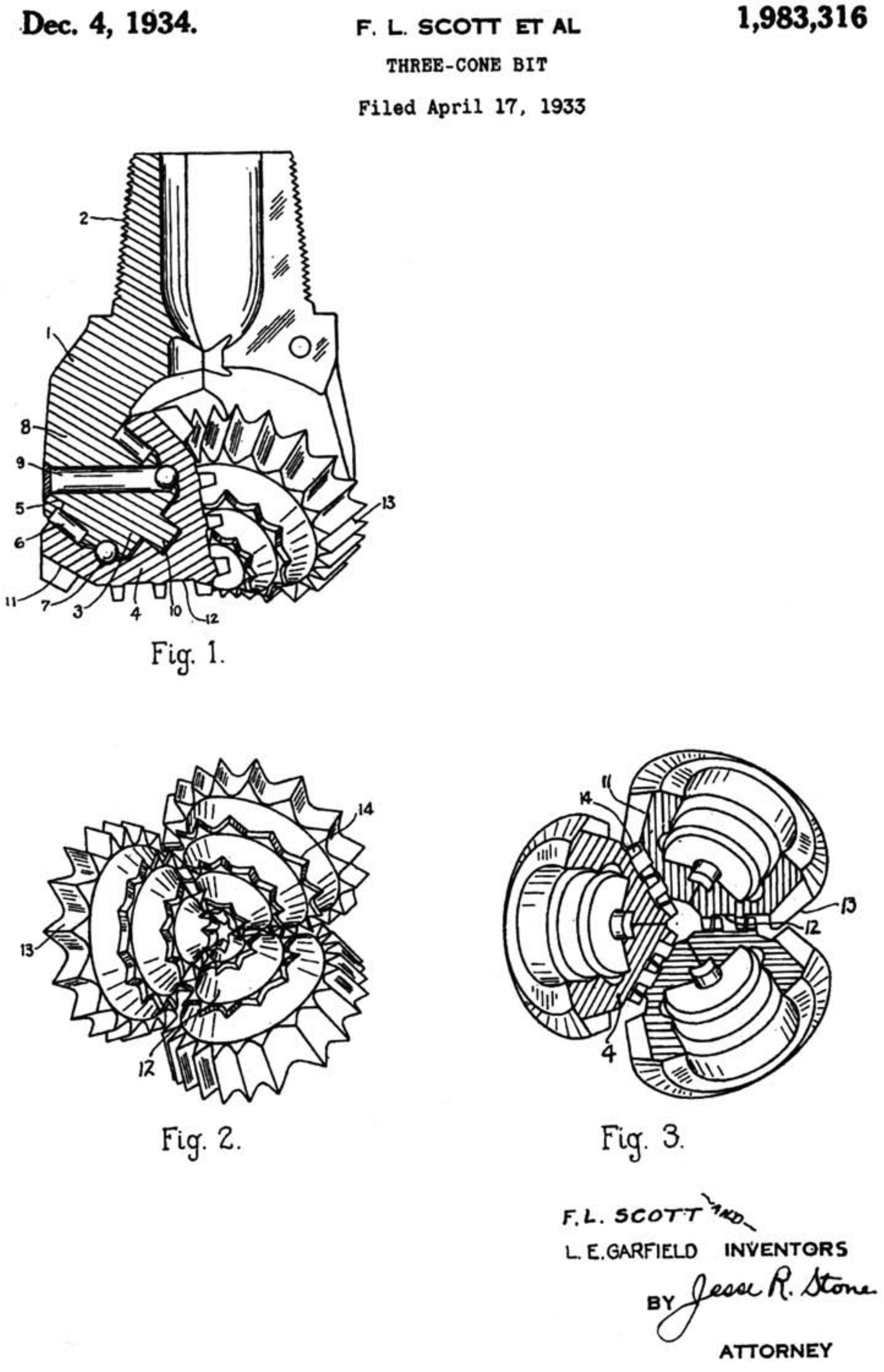


Figure 10.7. The three-cone roller cone bit as depicted in U.S. patent 1,983,316. Scott et al. (1934).

Formation	Insert Shape		Applications	Rock Hardness Range
Soft		Long Tooth-Shaped Carbide Insert	Economically drills most soft formations which can be drilled with steel tooth bits. Maximum penetration rates with minimum bit weight.	0-25000 psi 0-172 MPa
Medium		Tooth-Shaped or Conical Carbide Insert	For medium-hard and slightly more abrasive formations Faster penetration with lower drilling weights than those usually required for harder formation steel tooth bits.	10-50000 psi 69-345 MPa
Medium-Hard		Conical Shaped Carbide Insert	For hard abrasive formations requiring higher drilling weights. Bit is designed for higher penetration rates in high strength formations.	40-85000 psi 276-586 MPa
Hard		Conical or Ovoid Shaped Carbide Insert	For hardest and/or most abrasive formations requiring maximum drilling weights. Durable cutting structure provides long life.	60-100000 psi 414-689 MPa

Figure 10.8. Example of matching formation, insert shape, application and rock hardness. Security Bit Catalog.

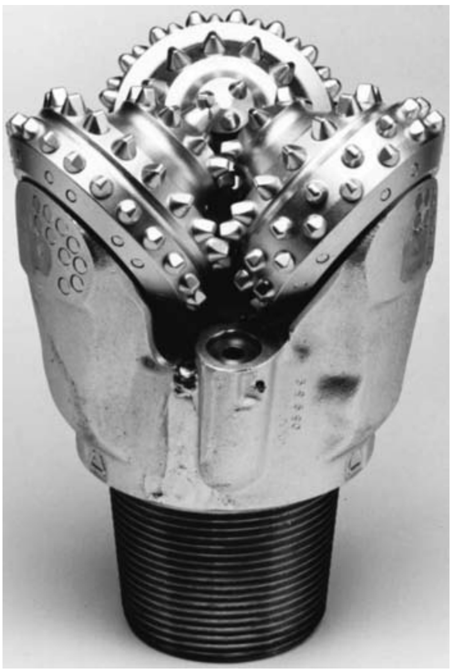


Figure 10.9a. Sandvik medium formation bit. Persson (1993).

tungsten carbide inserts) are made to roll along the hole bottom under the rotation of the drill string. With the addition of an axial force (pulldown or weight) on the bit, the individual cutting elements in contact with the bottom begin to penetrate the surface.

Figure 10.14 is a diagrammatic representation of a section taken through one cone of a tricone insert bit.

As the drill stem rotates, the cutting elements are successively forced into the surface. The idealized indentation on the bottom of the hole due to the penetration of cones A, B and C is shown in Figure 10.15.

Figure 10.9b. Sandvik hard formation bit. Persson (1993).

The actual pattern of indentations developed on the hole bottom by a rotary cone bit drilling in Texas pink granite under laboratory conditions is shown in Figure 10.16. The resemblance to the idealized breakage patterns is quite clear. Through good luck, a slab broke off during the drilling offering a rare cross-sectional view of the bit at the bottom of the hole. The bit and inserts used in the test may be clearly observed in Figure 10.17.

To put these observations into the proper context, it is necessary to examine the rock removal action for a single cutting element. Figure 10.18 shows an idealized view of a single insert being pushed into the rock surface in a laboratory setting.

The first contact of the hemispherical indentor and the rock surface is at a single point. As one can easily imagine, since the contact area is very small, the stresses are very high. In the literature, the stresses developed when such surfaces first come into contact and deform slightly but elastically under the imposed loads are called Hertzian contact stresses. They are named after Heinrich Hertz whose original work "On the contact of elastic solids" was published in 1882 (Hertz, 1882). Since the rock is comparatively much weaker than the insert and has a much lower elastic modulus, the interaction quickly changes from one of elastic bodies in contact to one where rock failure occurs.

A thin crushed zone forms directly beneath the contacting point of the indenter. This is indicated as the "shell of recrystallized material" in Figure 10.18. This thin zone is in triaxial compression (very highly confined) and as a result very strong. It transmits a largely radial pressure from the insert to the surrounding rock, which due to its lower (generally small) confinement, undergoes various degrees of compressive, shear and tensile failure.

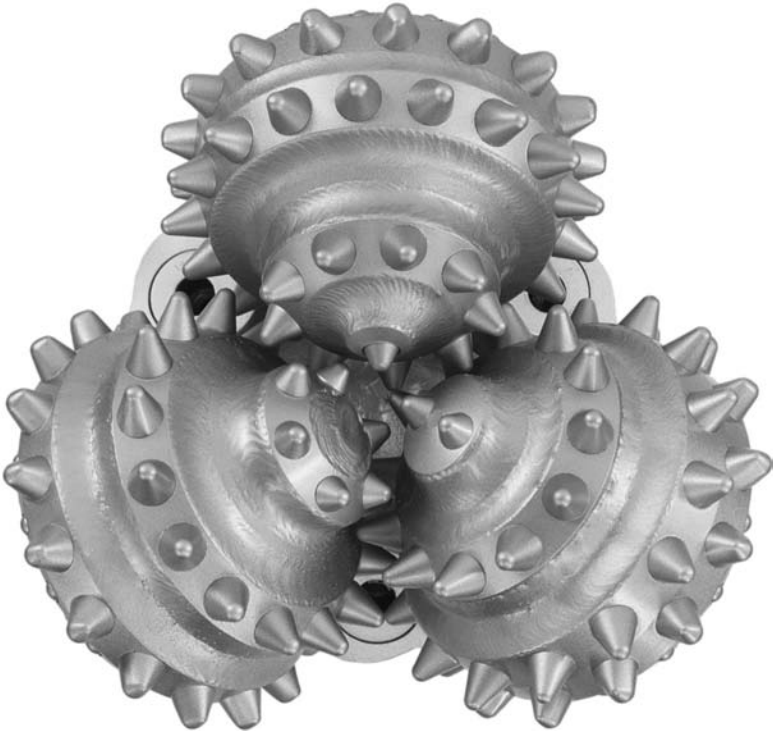


Figure 10.10a. Soft formation bit. Sandvik Mining (2012d).

For example, it is very common to observe a tensile fracture extending downward into the rock along the insert axis.

The zone of relatively high compressive stress and compressive failure is designated as "crushed zone of wedging material" in the figure. This zone grows in extent with increasing penetration and in doing so it exerts a lateral and uplifting force on the adjacent rock. Radial cracks extend outward from the crushed zone. Eventually, the area of contact and the pressure are sufficient to wedge the adjacent material free and create chips. The chipping may occur symmetrical to the insert point or asymmetrically. Chipping is the most efficient phase of the rock removal process and is to be encouraged.

The required depth of penetration for chipping to occur depends upon the cutting element geometry, the rock properties and the proximity of the adjacent craters. These chips often extend in depth to the 'nose' of the false wedge and hence the effective cutting depth can be considerably greater than the 'penetrated' depth. With the removal of the chips, the indenter moves down into the "false wedge" of crushed material below and the process continues. If the depth of penetration is small, little or no chipping occurs and the drilling process is very inefficient. Thus, there is a close relationship between the load on the bit, the insert design, and the proximity of adjacent craters which leads to an optimization of the rock removal process.

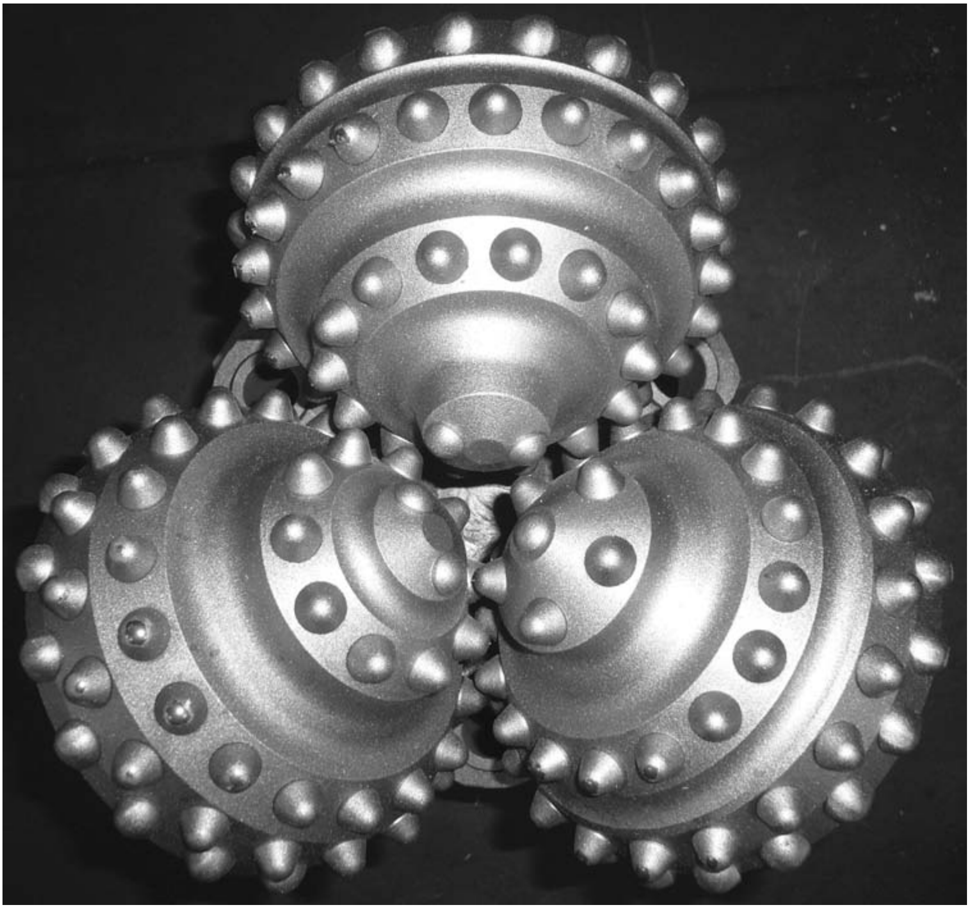


Figure 10.10b. Hard formation bit. Sandvik Mining (2012d).

Figure 10.19 shows a plan view of a rock surface in which several isolated indentations have been made. It is clear that by careful spacing of the indentation locations that interaction will occur and a large area of the surface will be affected.

When the hard metal insert bit first become a reality it was felt that the hemispherical ended or "round top" insert was best suited for drilling hard abrasive formations due to its inherent strength. This shape along with complete bottom hole coverage has proven to be true when drilling formations such as taconite. As the insert bit was used more universally, it became apparent that softer formations could be economically drilled with it. Round top inserts in conjunction with wider spacing were used to effectively do this job. In recent years, the rock bit designer has found that "shaped" carbide inserts when properly utilized will give dramatic improvement in bit performance.

Inserts are made from a mixture of fine grained tungsten carbide powders which are blended with pure cobalt powder. The powders are pressed to shape and then liquid-phase sintered at about 2500°F. At that temperature there is a liquor of cobalt and dissolved tungsten carbide. When the part is cooled, a fully dense part is formed consisting of grains of

Figure 10.11. Medium – hard formation bit. Varel International (2012b).

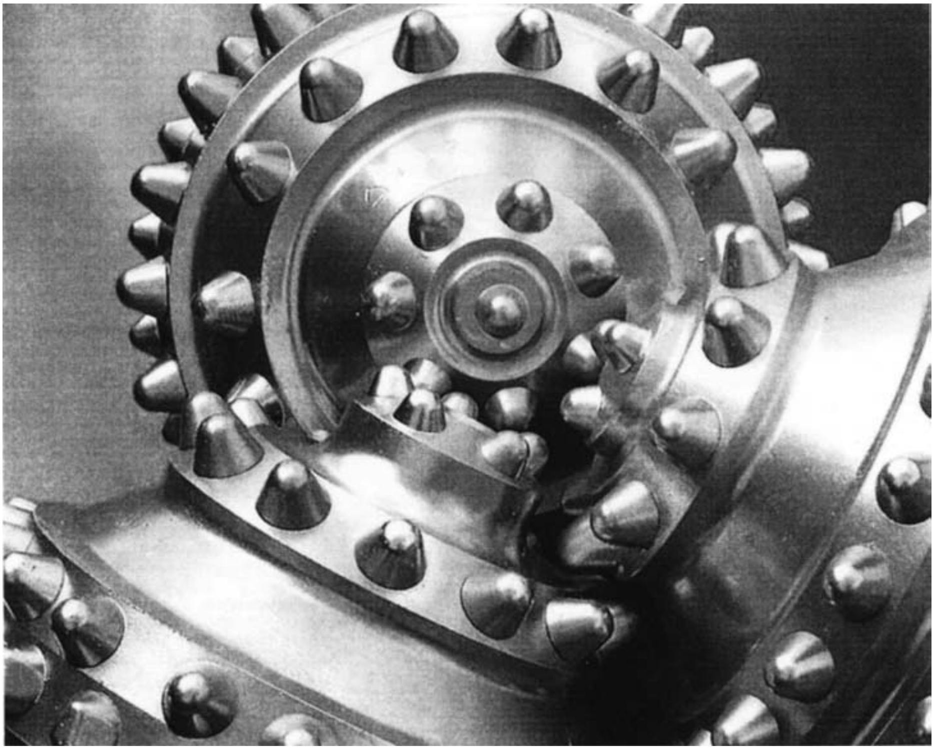


Figure 10.12. Bottom view of a tricone bit showing the interlacing of the cutters. Atlas Copco Secoroc (2005).

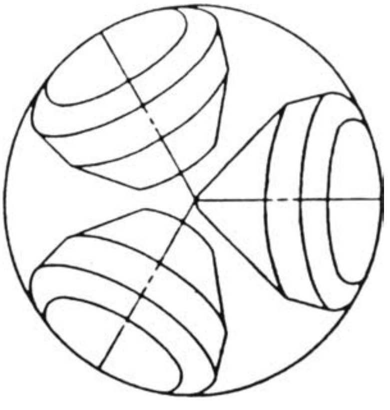


Figure 10.13. View looking down on a three-cone bit. Chitwood (1970).

tungsten carbide cemented together by a binder of cobalt-tungsten-carbon. These materials are referred to as "cemented carbides". There are two basic characteristics of inserts made of cemented carbides; hardness and fracture resistance. As the hardness is increased by decreasing the cobalt content or decreasing the grain size, fracture resistance decreases. As the hardness is decreased by increasing the cobalt content or increasing the grain size,

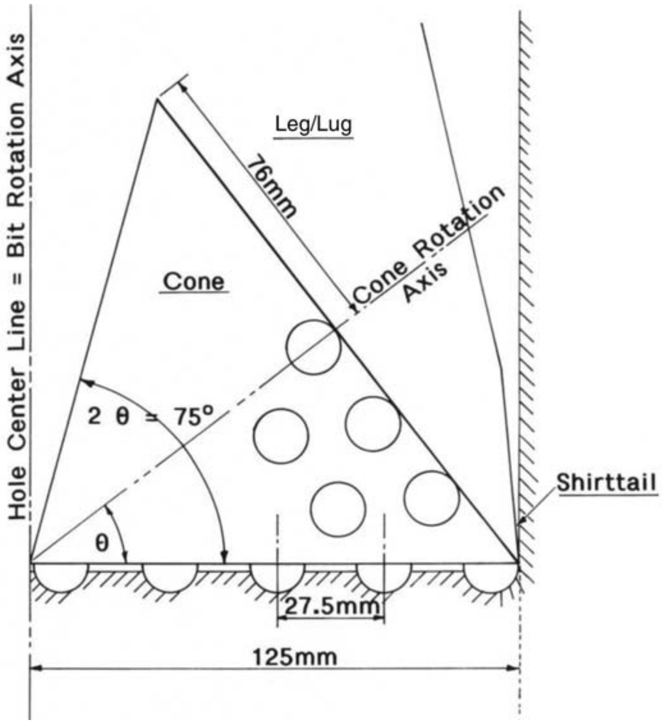


Figure 10.14. Diagrammatic representation of one cone of a tricone bit showing the penetration of the inserts into the rock surface.

fracture resistance increases. Under applications involving high abrasion and low shock loading, inserts would require 90% tungsten carbide or higher. High shock loading applications require grades with greater cobalt content. This is shown in Figure 10.20. Selection of a proper grade for an application must be a compromise of wear and shock resistance qualities.

The cutting elements of the cone type bit are circumferential rows of teeth extending from each cone and inter-fitting between rows of teeth on the adjacent cones. The sintered tungsten carbide teeth are pressed into holes drilled in the cone surfaces. For the steel tooth cutters, the teeth are machined from the basic cone material.

10.3 ROCK BIT COMPONENTS

A rolling cutter rock bits consist of four major components

- the bit body
- the cutters
- the bearings
- the bearing cooling/cuttings removal system

The elements for a typical bit in Figures 10.21 and 10.22.

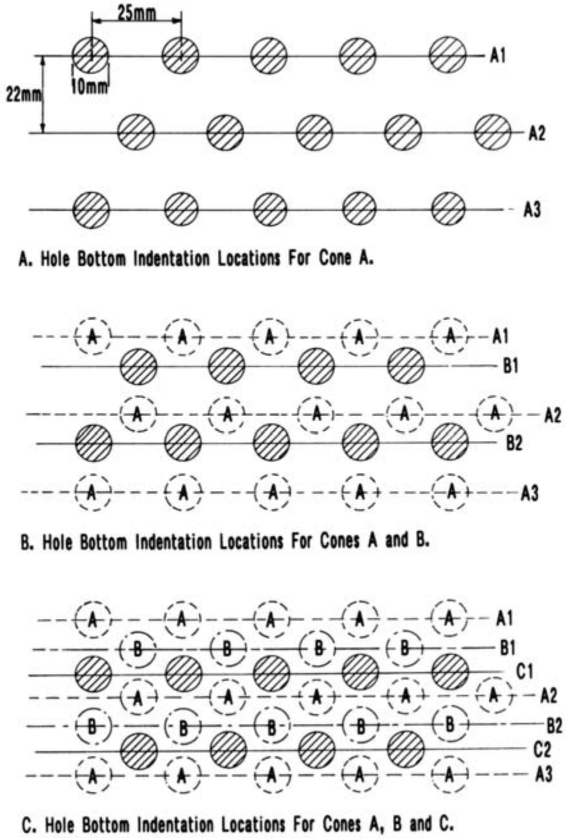


Figure 10.15. Idealized successive indentations on the hole bottom for a tricone bit with cones A, B and C.

Bit design has to start by recognizing that everything must fit into the hole being drilled. As a result it involves a series of trade-offs. In most instances one part can only be made larger at the expense of making some other part or parts smaller.

As can be seen in Figures 10.21 and 10.22, the tricone bit is fabricated from three legs (lugs) each of which carries a bearing journal and cutter. The three legs and various components fit together to form the finished tool. Drilling a hole requires that the cutters work outside the supporting parts. For this reason, rock bit journals are mounted at an angle. This angle is called the "journal angle" and is the angle formed by the center line of the journal intersecting a horizontal plane. The cutters are mounted on bearings which run on pins that are an integral part of the bit body. Radial loads are absorbed primarily by the larger outer bearing element (either a roller bearing or a journal near the cone mouth) and the plain bearing in the cone nose. Ball bearings serve to retain the cones and in some instances to absorb both radial and thrust loads. Additional outward thrust bearing capacity is provided by plain bearing surfaces at the inner end of the bearing pin and at the shoulder between the ball race and nose bearing. The space allotted to the various components depends on the type of formation to be drilled by the bit. Soft formation bits, for instance, which are generally run with less weight, have smaller bearings, thinner cone shells and thinner leg

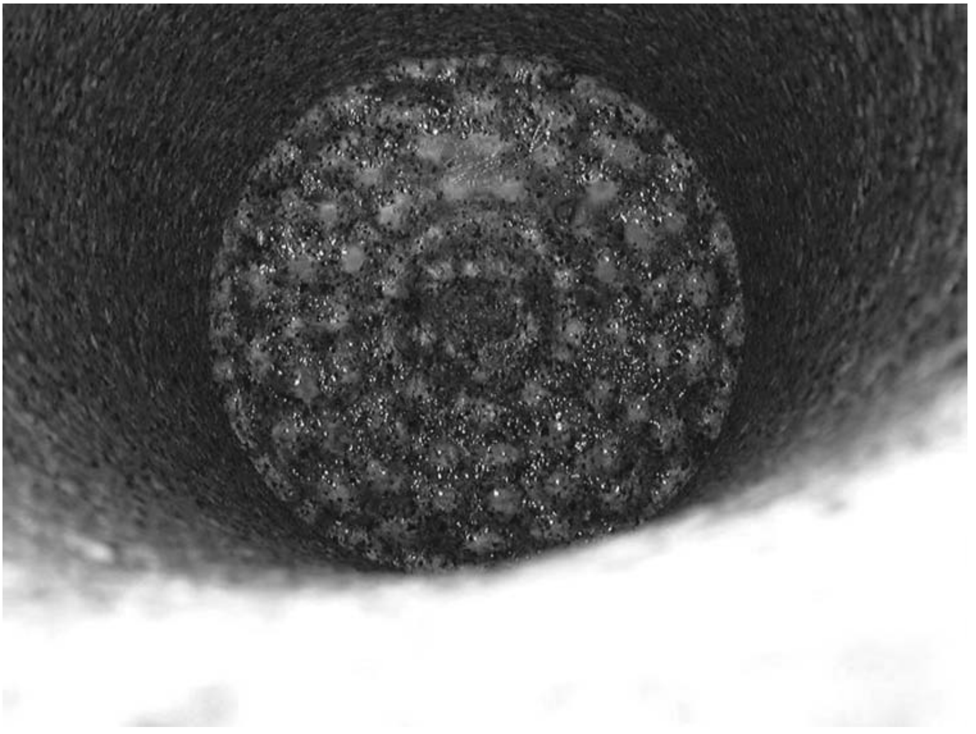


Figure 10.16. Photo of the hole bottom showing the indentation/rock removal pattern. Tricone bit drilling Texas pink granite. Atlas Copco (2012a).

sections than hard formation bits. This allows more space for long toothed cutting elements. Hard formation bits which must be run under much higher weight on the bit to achieve the required penetration, have stubbier cutting elements, larger bearings and sturdier bodies. Shirttail abrasion protection is standard on all blasthole bits. Without this wear resistant material, bit shirttails would wear rapidly allowing the bearings to become exposed and shorten bit life.

10.4 ROLLER BIT NOMENCLATURE

The International Association of Drilling Contractors (IADC) first developed their code for characterizing roller cone bits in 1972. The IADC code is updated periodically with the most recent revision occurring in 1992. Every bit design is designated by a series of three numerals followed by an optional alphabetic letter (123A, for example). The basis for assigning each of these identifiers as described in information provided by Sandvik Mining (2012d) is summarized below:

Character 1 reflects the type of cutter and the general formation hardness characteristic for which the bit is considered best suited. One of eight numbers is assigned.

- Numbers 1 to 3 apply to milled tooth bits
- Numbers 4 to 8 apply to tungsten carbide insert (TCI) bits

Figure 10.17. Cross-section through the hole of Figure 10.16 showing the bit and the cutters. Atlas Copco (2012a).

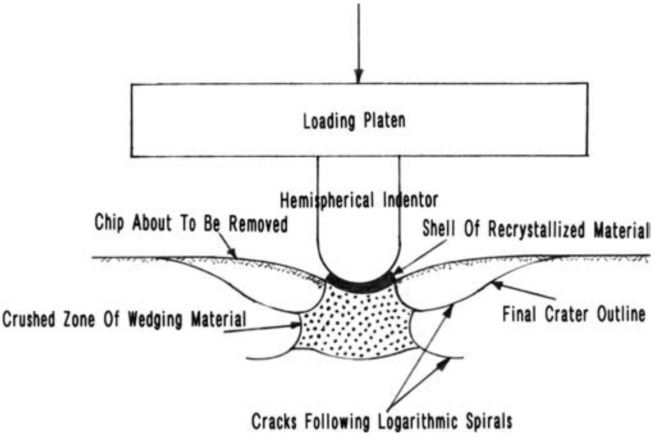


Figure 10.18. Idealized indentor cratering model (Bauer and Calder (1967).

Within each series, the lowest number reflects the softest rock and the greatest number the hardest rock.

Character 2 reflects the rock hardness sub-category. It ranges from 1 to 4 (softest to hardest).

Figure 10.19. Plan view showing several isolated indentations made in a rock surface under laboratory conditions. Atlas Copco (2012a).

Character 3 refers to the bearing design and the gage protection. The seven number assignments are defined below:

1. Standard, non-sealed roller bearing
2. Roller bearing, air cooled
3. Roller bearing, gage protected
4. Sealed roller bearing
5. Sealed roller bearing, gage protected
6. Sealed friction bearing
7. Sealed friction bearing, gage protected

Character 4 is optional and refers to optional available features:

A. Air application
B. Special bearing/seal
C. Center jet
D. Deviation control
E. Extended nozzles
G. Gage/body protection
H. Horizontal application
J. Jet deflection
L. Lug pads

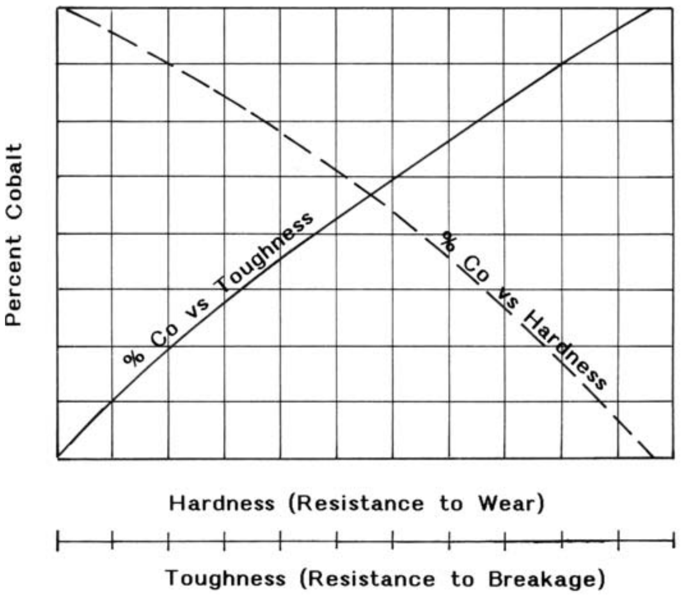


Figure 10.20. Relative hard metal properties as a function of cobalt content. Chitwood (1970).

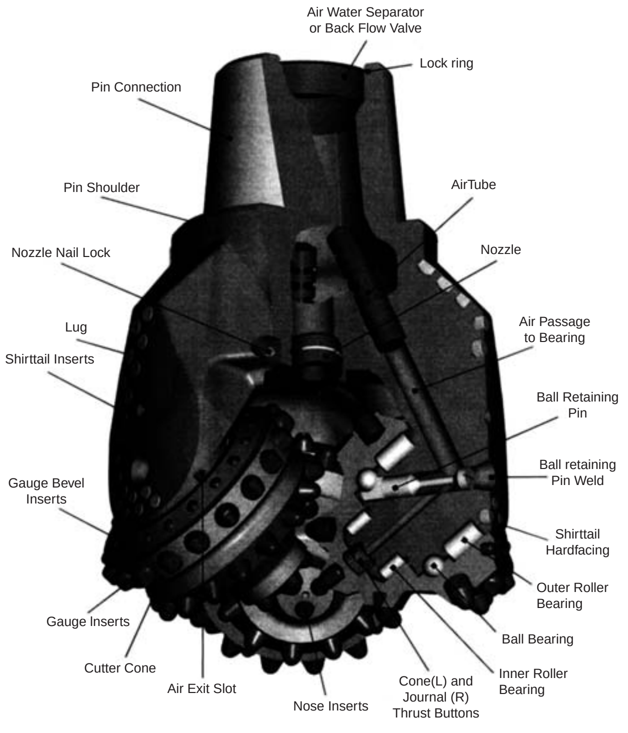


Figure 10.21. Elements of a tricone bit. Atlas Copco (2011b,e).

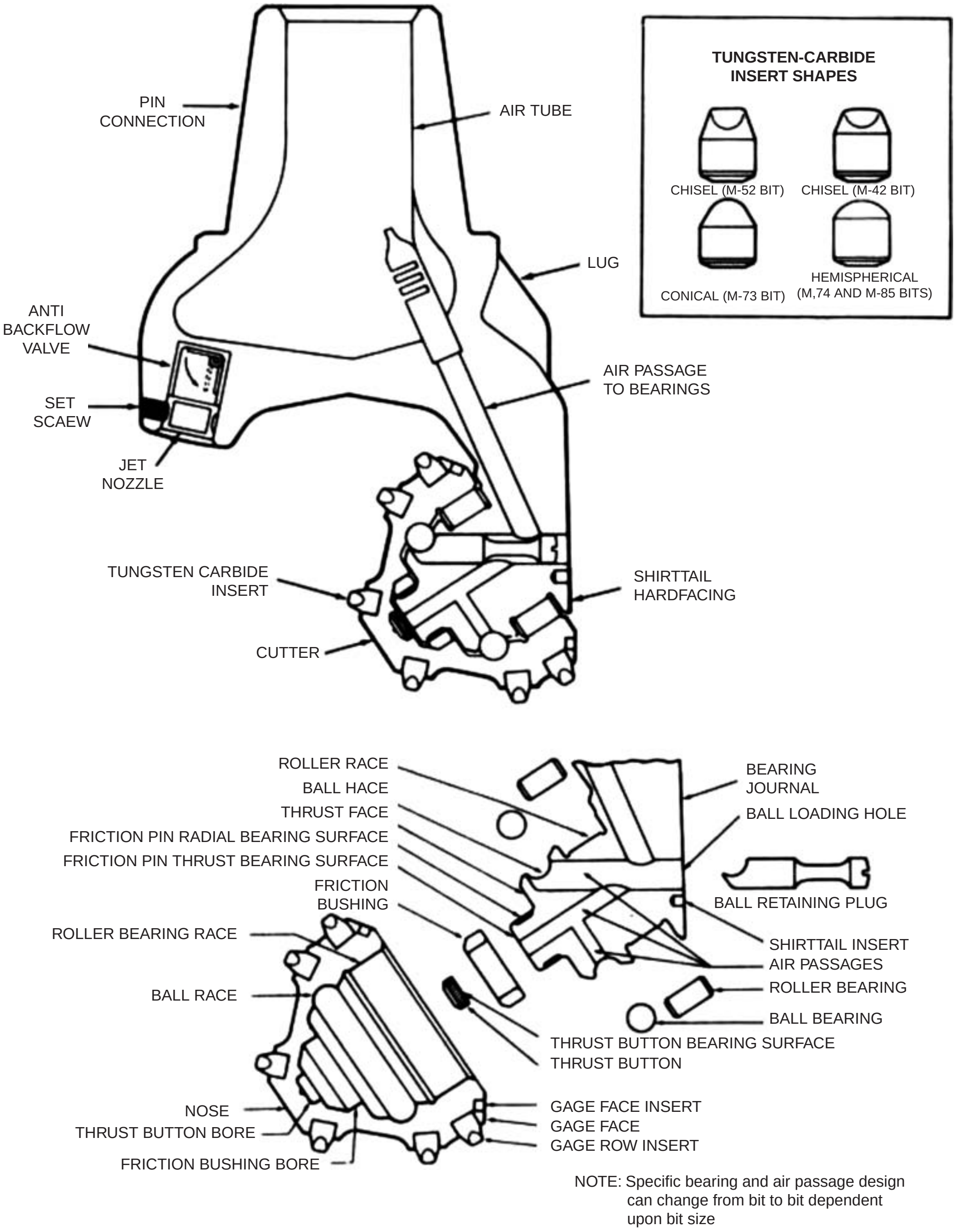


Figure 10.22. Blast hole bit components (Martin, 1982 and Reed Mining Tools, Inc.).

M. Motor application
S. Standard milled tooth
T. Two-cone bit
W. Enhanced C/S

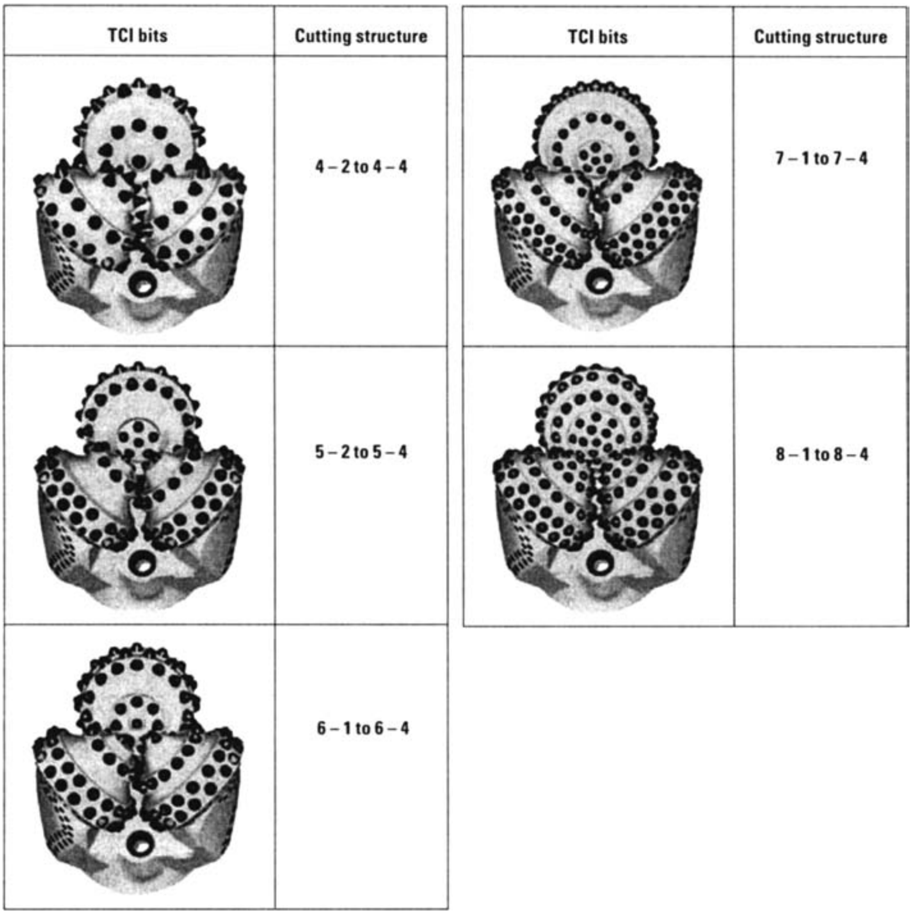


Figure 10.23. TCI Insert configurations corresponding to the IADC categories (Atlas Copco Secoroc, 2012).

X. Chisel tooth insert
Y. Conical tooth insert
Z. Other shape inserts

As seen, there are 16 alphabetic characters. Of these, the feature considered to be the most significant is listed. As an example, consider the designation for:

– milled tooth bit (softest rock)
– softest rock sub-category
– roller bearing – air cooled

The code 1-1-2 would be assigned. For a TCI bit with the same basic conditions except that a sealed roller bearing – gage protected is used, the code would be 4-1-5.

Atlas Copco (2011b) has provided the following practical guidance concerning the application of their Secoroc TCI bits:

a. Code 4-1 to 4-4: Very soft to soft rock (Unconfined compressive strength (UCS) of 1000–10000 psi). Rotation: 50–150 rpm. Pulldown: 1000–5000 lbs/in.

Technical data		
Drilling Method	Rotary - Single pass	
Hole Diameter	10 5/8 in - 16 in	270 mm - 406 mm
Hydraulic Pulldown	120,000 lbf	534 kN
Weight on bit	125,000 lb	56,700 kg
Hydraulic Pullback	60,000 lbf	267 kN
Single pass depth	65 ft	19.8 m
Maximum hole depth	135 ft	41.1 m
Feed speed	127 - 158 ft/min	0.6 - 0.8 m/s
Rotary head, torque	19,000 lbf-ft	25.7 kNm
Estimated weight	385,000 lb - 415,000 lb	175 tonnes - 188 tonnes
Dimensions tower up		
Length	53 ft 10 in	16.4 m
Height	103 ft 9 in	31.6 m
Width	26 ft 8 1/2 in	8.1 m
Dimensions tower down		
Length	98 ft	29.9 m
Height	27 ft 11 in	8.5 m
Compressor range		
Low pressure rotary	3,000 cfm@110psi / 84.9m³/min@7.6 bar	
Low pressure rotary (electric motor)	3,200 cfm@110psi / 90.6m³/min@7.6 bar	
Low pressure rotary	3,800 cfm@110psi / 107.6m³/min@7.6 bar	
Engine (Tier I)		
Caterpillar 3512	1650HP / 1230 kW@1800RPM (LP 3800)	
Cummins QSK45	1500HP / 1119 kW@1800RPM (LP 3000)	
Weg motor 6811	1400HP / 1044kW@ 50 or 60 Hz (LP 3200)	
Drill pipe specification: 35 ft (10.7 m)		
Drill pipe diameter	Suggested bit diameters	Thread
8 5/8" (219 mm)	10 5/8" - 11"	6" BECO
9 1/4" (235 mm)	11" – 12 1/4"	6" BECO
10 3/4" (273 mm)	12 1/4" - 13"	8" BECO
12 3/4" (324 mm)	15" – 16"	8" BECO
13 3/8" (340 mm)	16"	10" BECO

Figure 10.24. View of a large rotary blasthole drill (Pit Viper 351) with some key specifications. Atlas Copco (2011d).

b. Code 5-1 to 5-4: Soft to medium rock (Unconfined compressive strength (UCS) of 6000–28000 psi). Rotation: 50–150 rpm. Pulldown: 3000–6500 lbs/in.
c. Code 6-1 to 6-4: Medium to medium hard rock (Unconfined compressive strength (UCS) of 22000–42000 psi). Rotation: 50–120 rpm. Pulldown: 4000–7000 lbs/in.
d. Code 7-1 to 7-4: Hard to very hard rock (Unconfined compressive strength (UCS) of 36000–56000 psi). Rotation: 50–90 rpm. Pulldown: 4000–8000 lbs/in.
e. Code 8-1 to 8-4: Very hard to extremely hard rock (Unconfined compressive strength (UCS) of 48000–70000 psi). Rotation: 40–80 rpm. Pulldown: 6000–9000 lbs/in.

The bit designs corresponding to these codes are shown in Figure 10.23.

10.5 THE ROTARY BLASTHOLE DRILL MACHINE

A modern rotary blasthole drill is shown in Figure 10.24 together with some of the specifications.

A diagrammatic representation of a crawler drill with key components identified is provided in Figure 10.25.

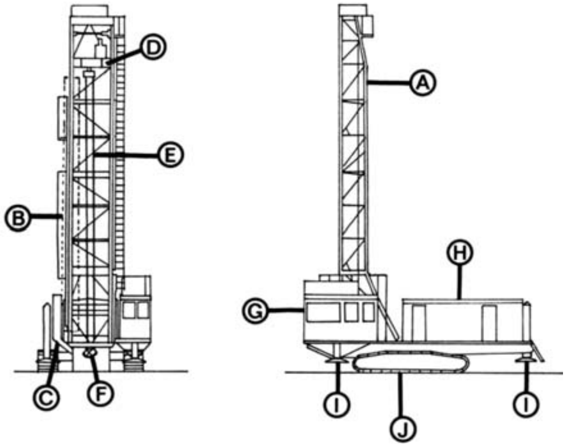


Figure 10.25. Diagrammatic representation indicating key components. Martin (1982).

In Figure 10.25, the following components are denoted:
A: Mast
B: Pipe rack
C: Dust collection system
D: Top drive
E: Drill pipe
F: Bit
G: Cab
H: Machinery house
I: Leveling jacks
J: Crawlers

The functions that are performed by the machine are depicted in Figure 10.26.

The functions and their activation/drive systems have been summarized by Martin (1982) as follows:

- Propel system, motors
- Mast raising and lowering, hydraulic cylinders
- Machine leveling, hydraulic cylinders
- Bit rotation, motors
- Bit pull down pressure, hydraulic cylinders or motors
- Tool and pipe changing, hydraulic cylinders
- Chip removal, compressed air
- Dust control, optional systems

The basic operations in the drilling process as outlined by Martin (1982) are:

- The machine is propelled and maneuvered into position where the hole is to be drilled. This has been previously determined by selection of a drilling pattern.
- The mast is raised into position. For short distance moves, the mast is kept in the raised position. As seen, the mast supports the drilling mechanism.
- The machine is leveled into a horizontal orientation so that holes drilled are consistently vertical (or at a specific angle). While drilling, the wheels (and/or crawlers) are raised off the ground.

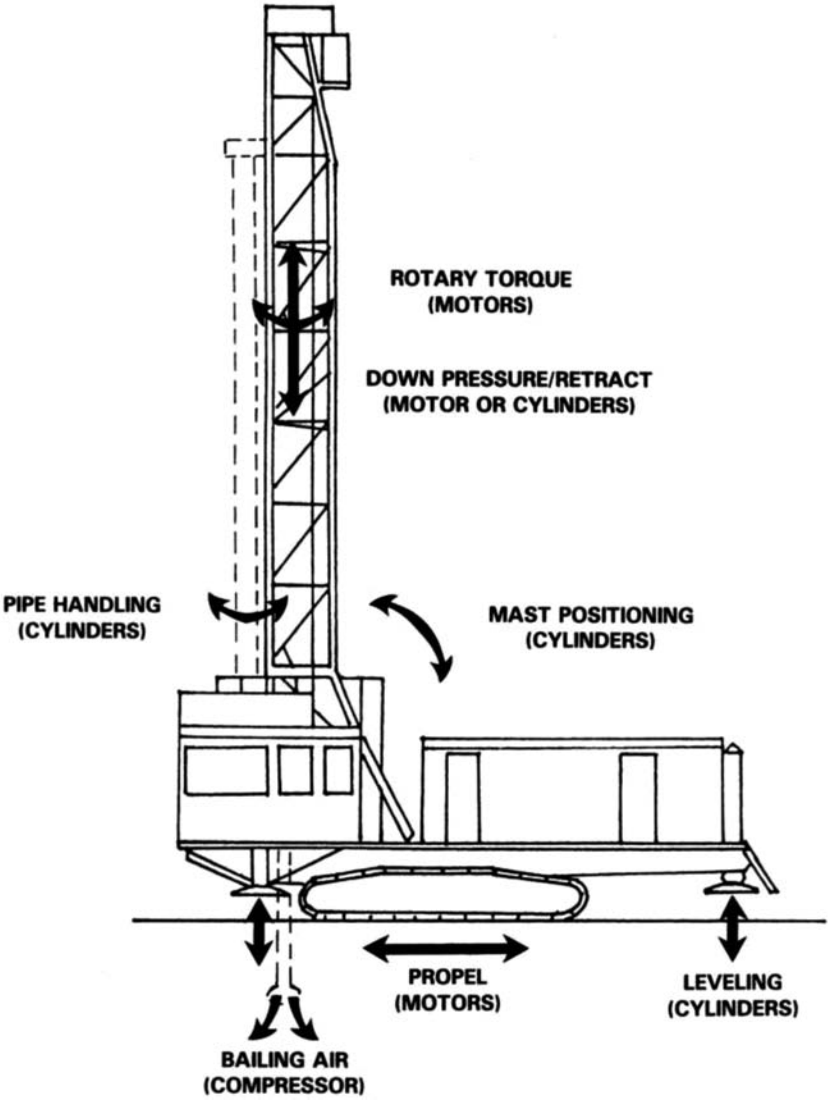


Figure 10.26. Functions performed by the drilling machine.

- Drilling is performed by forcing the rotating drill bit into the ground with a high down pressure and rotary torque.
- With increasing depth, the drill pipe on which the cutting bit is fastened is extended by adding additional lengths of drill pipe.
- Chips formed by the cutting action of the bit are removed from the hole by high velocity air fed down through the center of the drill pipe. The material chips are carried up the circular space between the hole and the pipe.
- Blown out chips and fines are collected on the surface, around the top of the hole with the air borne particles passed through a dust collection system.

- When the hole is completed to the desired depth, the bit is retracted with the drill pipe progressively removed. The machine then starts the relocation sequence.

There are various ways of accomplishing these different activities. The main system components to be discussed are

- mounting
- prime mover
- rotation
- pulldown and hoisting
- circulation
- mast and rod handling

Rotary blasthole rigs may be either crawler or truck mounted. Medium to large drills are always crawler mounted. Rubber tired machines are primarily used when drilling at a number of sites separated by significant distances. Regardless of mounting, all drill rigs are leveled and supported by hydraulic jacks. These can be operated simultaneously or individually to correct for uneven ground conditions. Drill mobility and maneuverability is provided by the mounting. Mobility reflects the time spent going from hole to hole and is called the tramming speed. Maneuverability is the ability to position the drill exactly where it should be. This often involves moving the drill in a very tight area and positioning it over a pre-determined location. For re-drilling holes, the drill must be relocated exactly. The steering of crawler mounted drills is done by controlling the crawlers. In some designs one crawler is locked while the other is engaged. In others, both crawlers can be operated, one in forward and the other in reverse. This gives a high degree of steering control. Of increasing importance is drilling the hole at the position and with the orientation desired, i.e. minimizing the hole deviation. Some sources of hole deviation are:

- accuracy of location
- the type of ground (presence/orientation of rock structures, varying hardness, etc.)
- the drill rig
- the degree of instrumentation
- the skill of the driller

The prime mover for the rotary rig is generally electric or diesel-hydraulic. Large crawler machines use either AC or DC electric systems for the powered systems. Electric drills are cheaper to own, operate and maintain than diesel-hydraulic. The main advantage of the diesel machine is the elimination of the power cable. This allows flexibility to propel to any area without connecting or disconnecting electrical power.

A power cable reel and remote propel features are available which allows one person to operate the drill (without the use of a helper). The power cable reels are electrical/hydraulic or air-operated. These are capable of reeling 2000 ft (600 m of trail cable). Power is required for

- tramming the rig from site to site
- operating the jacks and erecting the mast
- generating the pulldown force on the bit
- rotating the bit
- hoisting the bit
- operating the air compressor(s)

In all cases, the machine leveling, mast raising/lowering and pipe handling systems employ hydraulic cylinders. Many of the pulldown systems also use hydraulic cylinders.

The power required when drilling a hole of a particular diameter in a given material is made up of two parts. The first part is the rotary power (P_R) which is proportional to the rotation rate (R) and the torque (T).

$$P_R = 2\pi RT \tag{10.1}$$

where

P_R = rotary power (in-lbs/min)
R = rotation rate (revolutions/min)
T = torque (in-lbs)

The second part, that associated with the feed (P_F), is proportional to the pulldown force (P_D) and the penetration rate (P).

$$P_F = P_D P = P_d DP \tag{10.2}$$

where

P_F = feed power (in-lbs/min)
P_D = pulldown force (lbs)
P_d = pulldown force/inch of bit diameter
D = hole diameter (in.)
P = penetration rate (in/min)

The total power applied is P_T

$$P_T = P_R + P_L = 2\pi RT + P_d DP \tag{10.3}$$

The required torque depends on the penetration of the inserts. For hard rock formations it will be small whereas for softer rocks it will be high.

During drilling, the bulk of the power is consumed by the air compressor (for hole cleaning) and the rotary drive. On most rigs, a single 'prime mover' drives the compressor off of one end of its drive shaft and a "multi-pump" off of the other. The "multi-pump" drives will have two to four pumps, with two main pumps. One main pump will work the left track and the rotation unit while the other main pump will work the right track and the pulldown system. Other pumps will run the compressor oil and engine coolant radiator fans (Atlas Copco, 2012a,b).

There are several methods for applying the required rotary torque to the drill string. In oil well drilling this is done using a kelly bar system and a rotary table. For blasthole drills, the rotative power is applied by the top head drive directly to the end of the drill string. The power is applied by a gear drive or by a hydraulic or electric motor. The entire drive unit is attached to the drill string and moves up and down on tracks mounted on the mast. When a new length of drill pipe is needed, the drive assembly is detached from the drill string and raised to the top of the mast to permit inserting a new length of pipe. Normally rotary speeds will vary between 30 and 150 rpm. Many "soft" formation drilling operations go up to 150 rpm. Above that point, the bits begin to experience gage row carbide breakage (Atlas Copco, 2012a). Most drills are designed with a torque capability of between 10 and 20 ft-lb per 100 lb of down force.

Table 10.1. Minimum pulldown values used for preliminary drill selection.

Bit Diameter, in.	Pulldown, lb
5	20,000
7	35,000
9	60,000
12	75,000
15	120,000

All pull down systems use the weight of the drill rig to react against the down force applied to the bit. In general the drill rig will be capable of applying about 50 to 60 percent of its total weight. The total down force that can be applied to the bit depends not only on the total rig weight but also how that weight is distributed. Balance of the drill is critical. Sufficient weight at the drill end to obtain rated down pressure without lifting the drill off the jacks is mandatory. There are five different arrangements which are being employed to provide the pulldown force in a top drive system.

- direct rack and pinion
- direct chain and sprocket
- chain and cylinder
- rack and pinion with chain
- cable and cylinder

The rack and pinion and the chain and sprocket systems are the most common. The chain and sprocket system uses a hydraulically driven sprocket gear that engages and moves two continuous roller chains attached to both sides of the top-drive motor mounting. Another system uses a driven pinion gear attached to the rotary drive unit that engages a rack mounted along each side of the mast. This system eliminates the ‘softness’ of the chain or cable connections between the head and the drive unit.

The rig should be capable of applying thrusts of 4000 to 9000 lb/in of bit diameter depending on the size of the bit to be used. Table 10.1 presents some data for preliminary drill selection.

Air is supplied by one or more on-board compressors. Although in the past both

- screw (medium pressure)
- sliding rotary vane (low pressure)

were used, today only screw type compressors are employed. Single stage compressors can supply air at 30 to 110 psi. P&H and Caterpillar/Bucyrus Erie drills are generally set to a maximum of 60 psi whereas Sandvik/Drilltech/Tamrock is set between 80–100 psi and Atlas Copco drills at 110 psi (Atlas Copco, 2012a,b). A two-stage unit can deliver both low pressure air at 30 to 60 psi and high-pressure air at 100 to 350 psi (some drills up to 500 psi).

The heavier cuttings settle around the top of the hole, generally contained by flexible curtains. There are two basic types of dust control packages used on blasthole drills – water injection and dry-type. Water injection relies on a water tank to deliver 0.5 to 2 gpm of water to the bit. Water injection generally will result in 15 to 20% shorter bit life depending on how much water is used.

In a dry-type of unit, a fan is used to suck fine particles into a deck-mounted dust collection system. Cyclones on filter bags may be used for fine particle separation.

Figure 10.27. Sandvik D90KS drill rig with the mast positioned for drilling angled holes in an Australian mine. Sandvik (2012d).

Masts are hinged and connected to hydraulic cylinders. This permits the mast to be lowered, if necessary, when moving the drill rig. If the drill center of gravity is low, the move short, and the terrain suitable, the mast need not be lowered when moving.

Maximum hole depth is a function of mast height, which limits both the length of the drill pipe and the amount of drill pipe that can be efficiently stored and handled.

In the past, drill pipe storage was provided, most commonly, by a vertical, rotatable carousel holding fixture mounted on the front of the mast. Today, single pass drills have no carousel. Rather some manufacturers provide one or two extra pipes in "pots" on the mast.

For most machines there is a choice of mast sizes available. These are generally given in terms of the length of a drill stem (maximum hole depth in one pass) that they will accommodate. The actual mast length is considerably longer than the stem length since it must include drill pipe, bit, stabilizer, rotary head, rotary coupling (swivel), bit basket, shock coupling, etc. The distance from the mast deck to the ground is approximately 5 feet. The bit, stabilizer and rotary coupling each add approximately another 5 feet. Hence a mast length of about 75 ft is needed when drilling 50 ft in a single pass.

The front and rear of the drilling machine are specified with respect to the mast position. The front of the drill is that end which the operator would face when looking towards the mast. Hence the rear of the machine is that closest to the hole.

If the drilling pattern requires angled holes, the mast can be tilted from the vertical. The angles can vary from 0 to 15 or 30 degrees generally in 5 degree increments. The mast is pinned into position during actual drilling. Figure 10.27 shows one drill rig with the mast set for the drilling of angled holes.

Angle hole drilling can be advantageous under some circumstances. Angle drilling is reported to

- minimize back-break
- increase breakage at the toe
- yield better fragmentation
- reduce noise and shock waves

However
- production may decrease since pull down capacity is reduced.
- the drill pipe will impede cuttings outflow as the pipe rubs on the bottom side of the hole.
- pipe connections are also slower.
- maintenance costs may increase with angle hole drilling as well.
- the bit and drill string striking against the bottom side wall of hole may induce vibration to the machine.
- other factors to consider are the higher capital expense for angle hole attachments and auxiliary equipment.

An old rule of thumb is that angle hole drilling reduces production output by about 20% and possibly, because of this, angle drilling is still seldom used in metal mining. Angle drilling is, however, often used in the cast blasting of coal overburden with very good results.

10.6 THE DRILL SELECTION PROCESS

Drill selection is a three step process. The first step involves the identification, in general, as to the type and size of drill required. A number of factors enter into deciding what size of drill to purchase (Nelmark (1983):
- Type of material. A good estimate of drillability is essential from the beginning of a mine study. As many core drilling samples as economically possible should be taken. Rock sample tests in laboratory conditions have correlated quite well with the actual drillability of material. With this type of information and material hardness, the number and size of drill units can be predicted.
- Production requirements. Knowing the volume of material to be moved along with type of material determines size and number of drills as well.
- Size of loading equipment, haulage and plant. This information determines the required ratio of drills to that particular equipment. Pit geometry, haulage restrictions, crushing capacity and fragmentation all enter into the decision making process.
- Bank height. This usually is determined by loading equipment, but the mine plan should take into account the single-pass capabilities of the drill or the depth of drilling if multiple passes are used on blast hole drill specifications.
- Blasting requirements. Drill hole patterns based on hole size, formation, burden, spacing and hole depth all factor into model selection.
- Environmental restrictions. Blast vibration levels and noise restrictions may be important in determining the size of hole to be drilled. Often these restrictions become all encompassing.
- Overall cost. When selecting the drill, all elements of cost must be analyzed and not just the initial capital equipment expenditure. This applies to maintenance, power; blasting, re-usables such as bits, drill pipe and drill collars, lubrication, labor; and depreciation. All of these factors will enter into selecting a drill for lowest overall cost.

The number of drills to be purchased is also a decision which needs to be made. Typically one drill is needed for each shovel. However, where pit layout requires great propel distances between drill patterns, an extra drill may be necessary.

In step two, the potential suppliers of the appropriate drill are identified.

In step three, the factors entering into the selection process become much more specific. Some of the considerations which might be added are (Martin, 1982):

- Machine capabilities (pulldown, rotary torque, etc.) must exceed formation penetration requirements.
- Maximum hole size capability increases with machine size.
- Larger machines are more rugged and can generally drill in harder formations.
- A machine that can handle drill pipe long enough to permit single pass drilling can significantly improve productivity.
- The production rate is dependent both on the actual penetration rate and on the time required for pipe changes and machine repositioning.
- Electric drives have the lowest operating cost, the longest service life and the best track record for reliability.
- Electric drives require an in-pit power distribution system.
- Three levels of pit and area mobility are available; low speed crawlers (electric machines), medium speed crawlers (diesel machines) and roadable high speed carriers (wheel mounted units).
- Dust control requirements are dictated by regulations.
- Optional equipment such as powered cable reels, automatic lubrication, automated controls, etc., can increase the efficiency of the drilling operations.
- Long term productivity is dependent on the ruggedness, reliability and maintainability of the design.

Special consideration must also be given to mine life, ruggedness of the terrain, the frequency and distance of moves between drilling locations, the availability of skilled operating and maintenance personnel, and maintenance and service support facilities. These are all examined and compared with the information contained in specification sheets (Martin, 1982).

Figures 10.28 through 10.30 present the specification sheets associated with a large rotary drill.

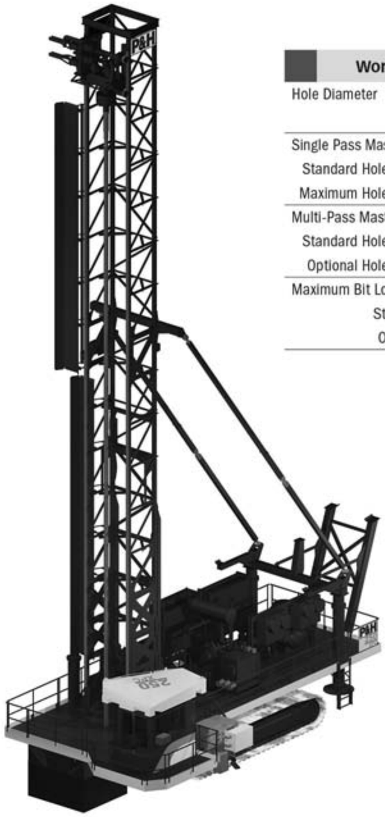
10.7 THE DRILL STRING

The pulldown force as well as the rotational torque are transmitted from the drill rig on the surface through the drill stem to the bit. The parts which make up the drill string are shown in Figure 10.31.

Drill stems vary in length from 15 to 55 ft. The current trend is to provide a mast/derrick high enough to permit use of only one drill rod for the depth of the hole selected. Some rigs, however, provide a semi-automatic feed rack, which simplifies adding and removing sections of drill stem. Drill stem connections are called "tool joints" and are usually of the BECO thread type for drill pipe 5″ and larger in diameter.

The blast hole drill stem is normally a flush OD member with a threaded connection at each end (male end = pin, female end = box). It has wrench slots at one or both ends for "make-up" or "break-out" with another threaded connection. Table 10.2 provides the dimensions for standard drill pipe.

Drill stems are usually chosen of sufficient size to give an adequate annulus velocity for cuttings removal. That size is generally sufficient to provide the necessary torque capacity. On air circulation rigs, the diameter of the drill stem should, at a minimum,



Working Ranges		
Hole Diameter	200 mm to 349 mm	7 7/8 in. to 13 3/4 in.
Single Pass Mast		
Standard Hole Depth:	19.8 m	65 ft.
Maximum Hole Depth:	59.4 m	195 ft.
Multi-Pass Mast		
Standard Hole Depth:	73.1 m	240 ft.
Optional Hole Depth:	85.3 m	280 ft.
Maximum Bit Loading		
Standard:	40,823 kg	90,000 lbs.
Optional:	47,627 kg	105,000 lbs.

Mast			
Construction	Lattice type using alloy steel structural shapes		
Single Pass Mast Multipass Mast		19.8 m 12.2 m	65 ft. 40 ft.
Raising and Lowering	Two hydraulic cylinders Diameter (each):	267 mm	10.5 in.
Angle Drilling	Up to 30° in 5° increments (only available with multipass mast)		

Hoist/Pulldown			
Design	Dual hydrostatic/direct drive chainless rack and pinion		
Bit Loading (maximum)	Standard: Hard Rock Carriage:	40,823 kg 47,627 kg	90,000 lbs. 105,000 lbs.
Drill Feed Rate	To	13.72 m/min	45 ft/min
Hoist Rate	To	27.43 m/min	90 ft/min

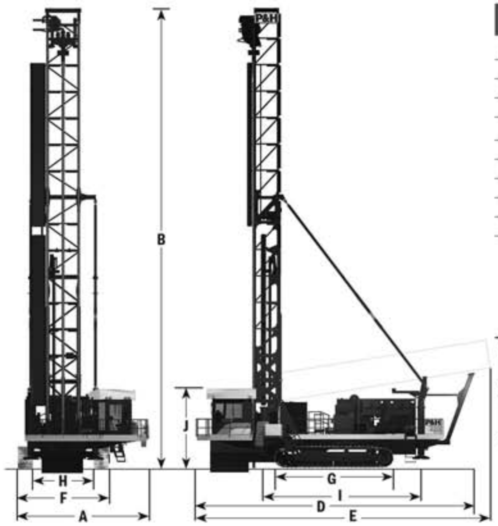
Rotary Machinery	
Design	Dual hydrostatic/helical gear
Rotation Speed	0-130 RPM
Maximum Torque	Up to 16,270 Nm (12,000 ft-lbs.) at 80 RPM

Pipe Handling			
Type Single Pass: Multipass:	Parallelogram style pipe rack, one rack is standard 5 pipe rotary carousel for 73.1 m (240 ft.) total depth		
Pipe Size	Diameter:	158 mm - 273 mm	6.25 in. - 10.25 in.
Options	Additional pipe rack, up to two total 6 pipe carousel for 85.3 m (280 ft.) total depth		
Auxiliary Equipment	Standard deck wrench, optional slide wrench		
Auxiliary Winch	Capacity:	5,443 kg	12,000 lbs.
Breakout Wrench	P&H SureWrench™		

Figure 10.28. Specification sheet (1) for a P&H 250 XPC electric rotary blasthole drill. P&H (2012a).

be 1-1/2 to two inches smaller than the hole diameter to (1) allow for cuttings transport, and (2) reduce the chance for sticking in the hole. Stems 2 to 3 inches smaller in diameter than the bit diameter are to be preferred since this provides a nice open annulus facilitating cuttings removal. Table 10.3 provides some typical bit diameter – pipe outer diameter (O.D.) combinations.

A bent stem will cause excessive vibration, undue stress in the drive train and usually causes uneven wear on rock bit cutters. For long steel life and overall economy, the thickest wall should be selected which is consistent with the limitations of the drill or the operation.



	Overall Dimensions		
A	Width, overall	7.83 m	25 ft. 8 in.
B	Height, mast up	27.77 m	91 ft. 2 in.
C	Height, mast down	8.38 m	27 ft. 5 in.
D	Length, mast up	16.83 m	55 ft. 3 in.
E	Length, mast down	28.72 m	94 ft. 3 in.
F	Overall width of crawlers	5.52 m	18 ft. 1 in.
G	Overall length of crawlers	7.04 m	23 ft. 1 in.
H	Width of jacks	3.91 m	12 ft. 10 in.
I	Length between jacks	9.51 m	31 ft. 2.5 in.
J	Height to top of op. cab	4.81 m	15 ft. 10 in.

Electrical Control Systems

GUI (Graphical User Interface):

Standard Touch screen including operating parameters

Standard operator error protection systems available (over-temperature shut-downs, over-tilt protection, pipe rack interlock protection, pipe protection software, etc.)

Lighting

Standard: LED floodlights for work and area lighting

Operator's Cab	
Type	Rear mounted with vibration and noise suppression
FOPS Rating	Level II
Noise	< 80 dB while drilling
Controls	PLC controlled, backlit for night operation
Glazing	Shatter-resistant, laminated glass on all sides, roof window with guard
Climate Control	Mine Air Systems or Bergstrom* HVAC unit available, providing pressurization and filtration

**Bergstrom is not a registered trademarks of Joy Global Inc. or any of its affiliates.*

Weights - Approximate			
Operating Weight (maximum)		113,400 kg	250,000 lbs.
Shipping Weight with Mast		105,929 kg	233,533 lbs.
Ground Bearing Pressure for Track Pads		101 kPa	14.7 psi
Ground Bearing Pressure for Jacks			
	Standard size:	607 kPa	88 psi
	Optional size:	10 kPa	45 psi

Leveling Jacks			
Cylinders	Four (4) 203 mm (8 in.) diameter x 1,676 mm (66 in.) stroke		
Jack Pads		Ground Pressure	
Standard:	Diameter: 762 mm (30 in.)	607 kPa	88 psi
Optional:	Diameter: 1,067 mm (42 in.)	310 kPa	45 psi
Auto Level	Standard feature		

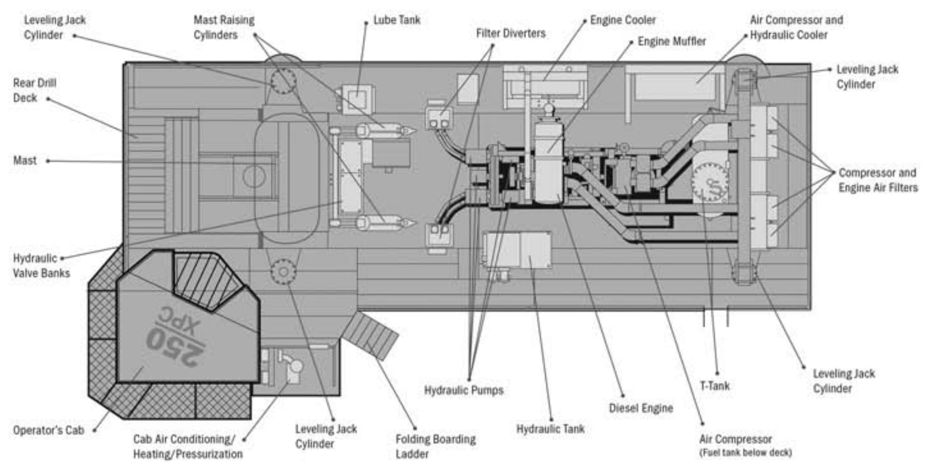
Lower/Propel			
Crawler Type	Heavy-duty B-8 excavator type with B-9 drive components, Intertractor design		
Shoe Width	902 mm (35.5 in.) width, double grousers		
Propel Machinery	Dual hydrostatic planetary drive with spring set, hydraulic release brake, 149 kw (200 hp) per crawler, towing release		
Propel Speed (maximum)	High:	3.06 kph	1.90 mph
	Low:	1.53 kph	.95 mph
Maximum Grade	41%		
Take-Up Adjustment	High pressure grease gun for hydraulic tensioning cylinder		

Figure 10.29. Specification sheet (2) for a P&H 250 XPC electric rotary blasthole drill. P&H (2012a).

The resistance to buckling of steel used in compression is a function of the wall thickness and diameter of the tube. Table 10.4 provides some common dimensions for drill pipe.

Whenever practical, the drill string should be made up of two identical lengths of drill steel with a 36″–48″ long "stabilizer sub" (see Fig. 10.31) located between the lower drill

Machinery Deck Plan



Air System

Compressor
Gardner-Denver SSY Series oil-flooded screw type

Available Volumes	Standard:	85 m^3/min	3,000 cfm*
	Optional:	97 m^3/min requires 783 kW / 1050 HP engine	3,450 cfm*
Sullair oil-flooded screw type			
Available Volumes	Optional:	102 m^3/min requires 783 kW / 1050 HP engine	3600 cfm*
Operating Pressure		448 kPa	65 psi
Air Filters	Dual Donaldson 2-stage, dry type (Gardner Denver)		
	Three Donaldson 2-stage, dry type (Sullair)		

**Nominal air volume. Actual values may vary according to application.*

Power Unit

Diesel Engine		
Power Rating Cummins QST 30		
	Standard:	634 kW (850 hp) @ 1800 rpm (Tier I/II)
	Optional:	783 kW (1,050 hp) @ 1800 rpm (Tier I/II)
MTU/Detroit Series 4000		
	Optional:	1193 kW (1,600 hp) @ 1800 rpm (Tier I/II) (Used in High Altitude Applications)
Fuel Consumption		115 liter/hr (30 gallon/hr) at 90% duty cycle
Fuel Capacity		4,542 liters (1,200 U.S. gallons)
Air Filtration		Dual SRG Series, 2-stage dry type
Electrical System		24 volt DC, standard 260 Amp alternator

Hydraulic System

Main System	Closed loop design utilizing dual variable-displacement piston pumps for propel, rotary, and pulldown
Auxiliary System	Open loop design utilizing a dual vane pump for mast raising, machine leveling, and pipe handling
Control Valves	Manifold mounted electro-hydraulic, PLC controlled Discrete 24V DC Solenoid
Hydraulic Lines	Extensive use of high pressure steel tubing
Filtration	3-micron return filters, 5-micron high pressure filters, 3-micron charge filters, suction strainers

Optional Equipment

- Bit Lubrication
- Cold Weather Package
- Deck Bushing (roller type)
- Fire Suppression
- High Altitude Package
- Remote Propel Control
- Shock Sub Adaptor
- Wiggins 'Fastfil' System

Figure 10.30. Specification sheet (3) for a P&H 250 XPC electric rotary blasthole drill. P&H (2012a).

stem and the bit. The "sub" takes 95% of the erosion and is much cheaper to replace than the lower pipe. Once the lower pipe is worn, it can be swapped with the upper pipe. It is normal to replace 5–10 bit saver subs before you need to switch pipe pieces (Atlas Copco, 2012a).

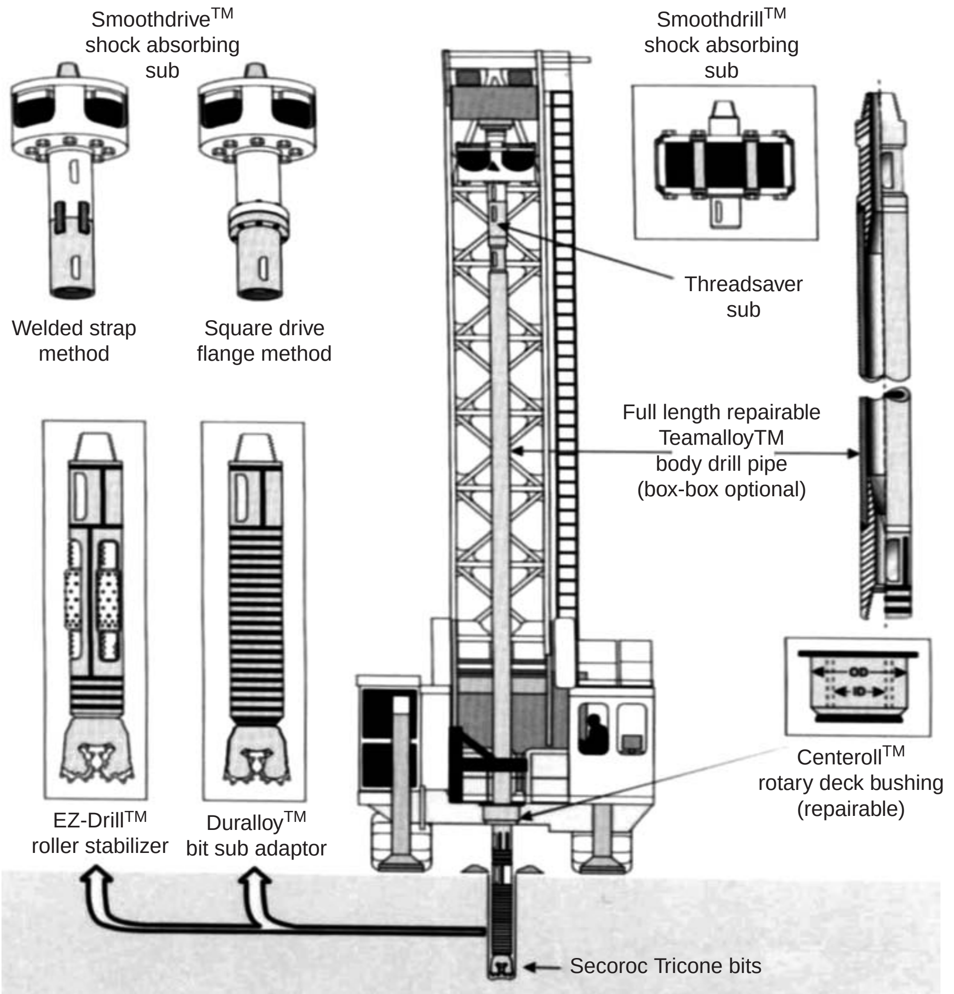


Figure 10.31. Components of a typical drill stem and drill stem stabilization system. Atlas Copco (2011c).

The following should be considered in the selection of drill steel (Nelmark, 1983), Toll (1981):

1. Drill hoisting capacity
2. Size and depth of hole to be drilled
3. Volume of circulation media available
4. Torsion, compression or tension loading
5. Makeup and breakout wrenching available

When drilling blastholes with a top drive rotary head configuration, the drill string tends to behave as a slender column and can be quite susceptible to buckling. The bit effectively acts as a bottom pivot point for the column. Any column buckling and hence rotation about this pivot point causes angular misalignment of the bottom hole assembly and gives

Table 10.2. Standard blasthole drill pipe sizes for use in surface mining applications. (Sandvik, 2012c).

Outer diameter O.D. (ins)	Outer diameter O.D. (mm)	Nominal wall Thickness (ins)	Nominal wall Thickness (mm)	Nominal weight (lbs/ft)	Nominal weight (kg/m)
3	76.2	0.2	5.16	6	9
		0.28	7.01	8	12
3 1/2	88.9	0.22	5.49	7	10
		0.3	7.62	10	15
4	101.6	0.28	7.15	11	16
		1/2	12.7	19	28
4 1/2	114.30	0.29	7.33	13	19
		0.34	8.56	15	22
		1/2	12.7	21	31
		3/4	19.05	30	45
		1	25.4	37	55
5	127	1/2	12.7	24	36
		9/16	14.29	26	39
		3/4	19.05	34	51
		1	25.4	43	64
5 1/2	139.7	1/2	12.7	27	40
		3/4	19.05	38	57
6	152.4	3/4	19.05	42	63
		1	25.4	53	79
6 1/4	158.8	1/2	12.7	31	46
		3/4	19.05	44	65
6 1/2	165.1	3/4	19.05	46	68
		1	25.4	59	88
7	177.8	3/4	19.05	50	74
		1	25.4	64	95
7 5/8	193.7	7/8	22.23	63	94
		1	25.4	71	106
8 5/8	219.1	1	25.4	82	122
		1 1/2	38.1	114	170
9 1/4	235	1	25.4	88	131
		1 1/2	38.1	124	185
10 3/4	273.1	1	25.4	104	155
		1 1/4	31.75	127	189
		1 1/2	38.1	148	220
12 3/4	323.9	1	25.4	127	189
13 3/8	339.7	1 1/4	31.75	163	243
		1 1/2	38.1	192	286

off-center load concentrations on the bit. This off-center distribution of down thrust on the bit causes three undesirable effects (Toll, 1981).

1. The full thrust is shifted from one cone to another as the bit rotates. This overloads the cone teeth or carbide buttons which rapidly shatter and break up.
2. The overload down thrust on the cone is transferred to cyclic thrust along the axis of the cone bearing pin as the bit rotates. This thrust can increase to the point where the ball and nose bearings are over stressed and the ball flange breaks down with the overall result of premature bearing failure.

Table 10.3. Various combinations of hole diameter and pipe diameter (Atlas Copco 2011f).

Hole diameter (ins)	Pipe O.D. (ins)	Hole diameter (ins)	Pipe O.D. (ins)
5 5/8	2 7/8	9 7/8	7
	3 1/2		7 3/4
	4		8 5/8
5 7/8	2 7/8		9
	3 1/2	10 5/8	7
	4		7 3/4
6	2 7/8		8 5/8
	3 1/2		9
	4	11	7
6 1/4	3 1/2		7 3/4
	4 1/2		8 5/8
	5		9
6 3/4	3 1/2	12 1/4	7
	4		7 3/4
	4 1/2		8 5/8
	5		9
7 3/8	3 1/2	13 5/8	10
	4 1/2		10 3/4
	5 1/2	15	10
7 7/8	3 1/2		10 3/4
	4 1/2		12
	5 1/2		13
	6 1/2	16	10
	6 5/8		10 3/4
	7		12
9	4 1/2		13
	5 1/2	17 1/2	10
	6 5/8		14
	7		16
	7 3/4		

3. The outside gauge cutting teeth or carbide inserts are the most heavily overloaded of all cone teeth. These break up and the cone assumes an “apple shape”. Bit life is severely reduced and bits worn in this manner tend to drill undersize holes.

The same overall effect is evident when the bit exhibits a tendency to follow structural planes and transfers a bending moment to the drill string. This, once again, results in angular misalignment. The common expression for this type of behavior is “coffee-grinding” (Toll, 1981). To prevent the bit acting as the rotation point, a stabilizer is used just above the bit (see Fig. 10.31). Theoretically the guiding elements should have the same diameter as the bit. Unfortunately this is not practical because of the normal attrition of rock bit gage wear surfaces. The stabilizer should therefore be held at the largest diameter practicable. Concentricity of the guiding elements is important. Eccentricity in these guiding elements is as detrimental to drill performance, rock bit life, and drill pipe life as adequate stabilization is beneficial. There are two types of stabilizers

- bar or fixed blade stabilizers
- roller stabilizers

Table 10.4. Common drill pipe dimensions. Sandvik (2012c).

Outer diameter O.D. (ins)	O.D. (mm)	Nominal wall Thickness (ins)	Nominal wall Thickness (mm)	Nominal weight (lbs/ft)	Nominal weight (kg/m)
3	76.2	0.2	5.16	6	9
		0.28	7.01	8	12
3 1/2	88.9	0.22	5.49	7	10
		0.3	7.62	10	15
4	101.6	0.28	7.15	11	16
		1/2	12.7	19	28
4 1/2	114.30	0.29	7.33	13	19
		0.34	8.56	15	22
		1/2	12.7	21	31
		3/4	19.05	30	45
		1	25.4	37	55
5	127	1/2	12.7	24	36
		9/16	14.29	26	39
		3/4	19.05	34	51
		1	25.4	43	64
5 1/2	139.7	1/2	12.7	27	40
		3/4	19.05	38	57
6	152.4	3/4	19.05	42	63
		1	25.4	53	79
6 1/4	158.8	1/2	12.7	31	46
		3/4	19.05	44	65
6 1/2	165.1	3/4	19.05	46	68
		1	25.4	59	88
7	177.8	3/4	19.05	50	74
		1	25.4	64	95
7 5/8	193.7	7/8	22.23	63	94
		1	25.4	71	106
8 5/8	219.1	1	25.4	82	122
		1 1/2	38.1	114	170
9 1/4	235	1	25.4	88	131
		1 1/2	38.1	124	185
10 3/4	273.1	1	25.4	104	155
		1 1/4	31.75	127	189
		1 1/2	38.1	148	220
12 3/4	323.9	1	25.4	127	189
13 3/8	339.7	1 1/4	31.75	163	243
		1 1/2	38.1	192	286

The bar (or fixed blade) type of stabilizer consists of a sub the same diameter as the drill stem and approximately three to four feet long. Three to four blades are welded to the stabilizer body so that they will be the same diameter as the hole being drilled. Pelletized tungsten carbide can be applied to the blades to resist abrasion. The wear rate can be excessively high in hard, abrasive formations due to the scraping action against the hole wall. This necessitates frequent rebuilding. Blade stabilizers should be changed every time a new bit is put on the string. In practice blade stabilizers are usually changed every second or third bit to reduce lost time due to changes and to defray the purchase price of the stabilizer. Blade stabilizers can be rebuilt or the rings can be replaced.

In roller stabilizers, the rollers are mounted on and freely rotate about large diameter tool steel bearing pins which are cooled by the circulating air. The guiding elements in this tool roll against the bore wall, providing close-to-hole wall stabilization while minimizing both attrition and torque requirements. Normally the rollers contain tungsten carbide inserts to resist wear. This assembly maintains gauge and effective stabilization for 15,000 to 20,000 m of drilling depending on tool size and ground conditions.

Stabilizers help to maintain a straight hole and produce a beneficially smooth bore. By acting as a guide, a properly sized stabilizer forces the drill bit to rotate about its own center, thereby utilizing the energy directed to it in the most efficient manner. Figure 10.31 shows a stabilizer assembly. As indicated, the blades or rollers acting on the hole walls effectively act as constraints on movement in the horizontal plane. The bottom hole assembly and stem column is stiffened by this constraint. The top of the stabilizer becomes the pivot point rather than the bit. The advantages are:

- if buckling were to occur it would not misalign the bit
- rod buckling would require a higher load because of the extra stiffness
- the blades or rollers act on the hole wall to resist any bending moment which might result from the bit tending to follow structural planes
- it allows higher down thrusts to be applied to the bit which means higher penetration rates while bit life remains relatively constant.

The major point of drill stem constraint apart from in-hole stabilizers and the point of attachment to the rotary drive head is the point at which the drill stem passes through the working deck of the drill rig (see Fig. 10.31). Constraint provided by deck bushing must be firm and concentric while allowing free stem rotation. Earlier, the most successful bushes for this point were made of hard cast iron (Toll, 1981). Manufacturers are now producing roller bearing equipped rotary deck bushes.

A vibration dampener is a resilient coupling used between the drill and the drill pipe on top drive machines to dampen shock loads created by the bit action (see Fig. 10.31). It is most beneficial when drilling in fractured formations, intermittent hard and soft layers, or hard formations (Toll, 1981). The following benefits occur

- reduction in drilling machine maintenance by dampening torsional and axial shock loads.
- increases drilling rates by keeping bit in more uniform contact with the formation allows the use of more weight and higher rotary speeds in rough drilling areas.
- increases bit life by dampening cyclical shock loading normally transmitted to the bit bearings and cutting structure
- decreases operator noise by eliminating the metal to metal contact between rotary drive and the drill pipe.

10.8 PENETRATION RATE – EARLY FUNDAMENTAL STUDIES (Morris 1969)

In 1969, Morris (1969) reported the results of a series of studies aimed at the development of a drillability index for 'hard' rock mining roller cone rotary bits. A 90 degree conical tungsten carbide compact with a 1/8 inch radius at the apex (Fig. 10.32) was mounted in a simple hydraulic press (Fig. 10.33).

This tungsten carbide compact was chosen as a test element to duplicate the drill bit element of a hard rock mining bit. The compact was pressed into a smooth, flat surface of a

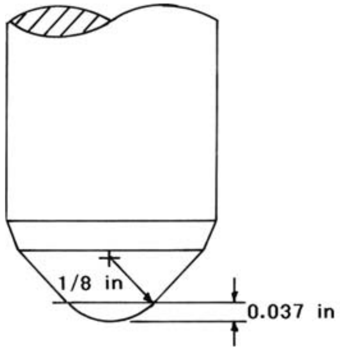


Figure 10.32. Bit element used in the laboratory experiments conducted by Morris (1969).

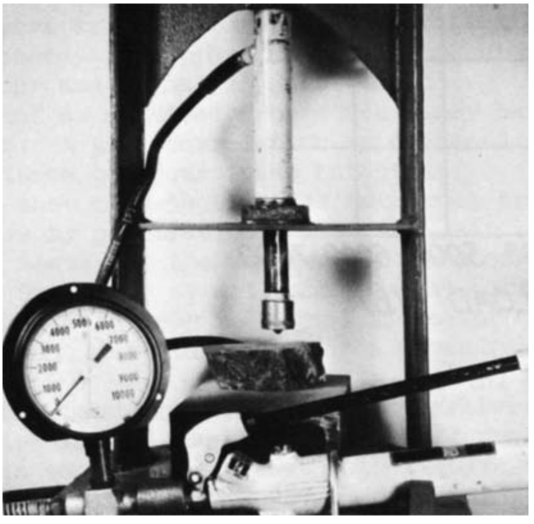


Figure 10.33. Hydraulic pump and ram with single carbide bit element. Morris (1969).

number of control samples with the hydraulic ram. The penetration (P) was recorded at load increments of 500 lbs up to a maximum load of 5000 to 6000 lbs which was considered to represent the maximum load per insert experienced in actual drilling situations.

A typical force-penetration curve is plotted in Figure 10.34. As can be seen, the penetration is a direct function of load up to the threshold force, E. There, due to extrusion of the crushed material from under the bit element, the penetration increases sharply.

The slope of the force-penetration curve to point E may be taken as a measure of rock strength under a hemispherical bit element. However, the relationship between actual crater depth (P′) and the threshold force (E) was thought to be more directly related to the drillability of the roller-cone bit. As indicated by Morris (1969), roller-cone bits drill most efficiently at a bit load in excess of a certain minimum threshold force (That load is considered to be related to the single bit element threshold force of the static penetrator test). In practice, Morris (1969) found that adequate penetration rates can be maintained at weights below

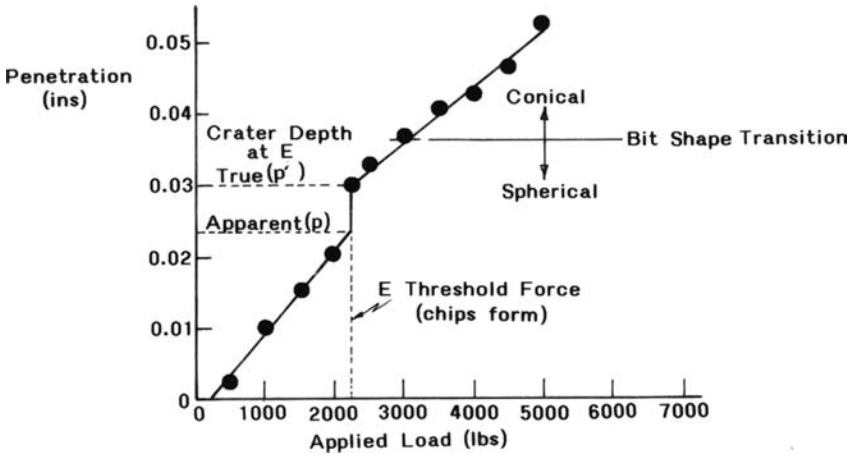


Figure 10.34. Typical penetration versus applied load curve for the 1/8 in. sphere. Morris (1969).

the static threshold weight. Dynamic loading, a relatively rough hole bottom, and the fact that all bit elements are not equally loaded lowers the effective threshold weight in actual drilling. For these reasons, the ratio P′/E, rather than the slope of the force-penetration curve (P/E) as the drillability index.

$$\text{Drillability index} = P'/E \tag{10.4}$$

Experience (Morris (1969)) has indicated that ground with a drillability index greater than 0.00002 in/lb can be successfully drilled with steel tooth roller bits or cutters whereas drillability index values less than 0.00002 in/lb dictate the use of tungsten carbide insert bits. By multiplying the static threshold force (E) by the average number of bit elements working (I), one arrives at an effective drilling weight.

$$W = EI \tag{10.5}$$

where

E = static threshold force
I = average number of bit elements in contact

The number of bit elements in contact with the rock surface at any one time depends upon the

- bit-type (design)
- size
- average bit penetration per revolution

Observations such as shown in Figure 10.35 reveal that

$$I = 0.08C \tag{10.6}$$

where

C = total number of bit elements

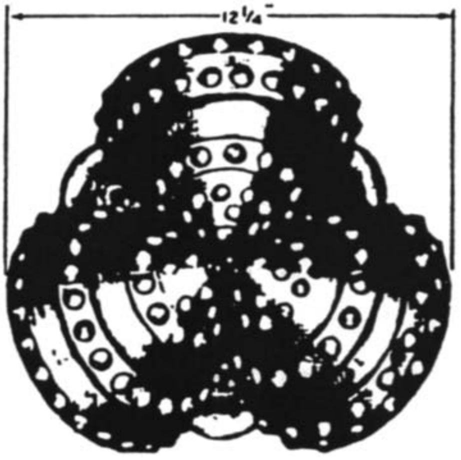


Figure 10.35. The inserts in contact with the bottom of the hole (Bauer and Calder (1967)).

Table 10.5. Number of bit elements working at any time (After Morris (1969)).

	Insert bits		Tooth bits	
Size	Type	I	Type	I
6-3/4″	H8M	10	H7M	14
	H10M	14		
7-7/8″	H8M	10	H7M	14
	H10M	17		
9″	H8M	12	H7M	18
	H10M	18		
9-7/8″	H8M	16	H7M	22
	H10M	19		
10-5/8″	H8M	17	H7M	22
	H10M	21		
12-1/4″	H8M	18	H7M	27
	H10M	25		

The average number of bit elements in contact with the rock is given in Table 10.5 for a number of different Dresser-Security bits.

Thus

$$W = 0.08EC \tag{10.7}$$

Through the use of this relationship which considers both bit size and type, one can avoid the concept of bit weight per inch of bit diameter. This is important since it has been found that bit weight should not be a direct function of diameter.

The penetration rate of a roller-cone rock bit may be expressed by

$$R = NP \tag{10.8}$$

Table 10.6. Comparison of the predicted and actual penetration rates (Morris (1969)).

Sample Number	Weight on bit (lb)	Rotary Speed (RPM)	Penetration Rate (ft/hr)		Drillability index, p′/E (in/lb)
			Actual	Test	
1	62,000	40	16.4	12.6	.0000108
2	68,500	41	17.4	14.0	.0000106
3	86,000	80	34.5	36.8	.0000113
4	85,000	60	19.3	20.0	.0000078
5	88,000	60	31.7	30.2	.0000122
6	60,000	60	48.0	52.6	.0000308
7	80,000	60	37.2	35.4	.0000156
8	65,000	60	50.0	44.5	.0000203
9	45,000	50	7.4	8.1	.0000067

where

R = penetration rate (in/min)
N = rotary speed (rpm)
P = bit penetration per revolution (ins.)

It will be assumed that the actual bit penetration per revolution is proportional to the bit penetration observed in the laboratory.

$$P = K_1 P' \tag{10.9}$$

where

K_1 = constant

Combining equations (10.5) through (10.9) one finds that

$$R = NK_1\left[\frac{P'}{E}\right]\left[\frac{W}{I}\right] \tag{10.10}$$

By comparing the predicted value from equation (10.10) with known penetration rates, it is found that $K_1 = 1.8$. This factor may reflect dynamic loading, interaction between adjacent craters, damage from previous impacts, etc.

The penetration rate in feet/hour is obtained using

$$R' = 5NK_1\left[\frac{P'}{E}\right]\left[\frac{W}{I}\right] \tag{10.11}$$

Table 10.6 presents a comparison between actual and predicted penetration rates. As can be seen, the average difference is of the order of 10 percent.

The samples listed in Table 10.6 include magnetic taconite, oxidized taconite, gabbro, diabase, and massive sulfide ore. These samples were taken from 7 different mines. Samples 3 and 4 were taken from two locations in a Minnesota taconite mine. Sample 9 (a very homogeneous pink quartzite) was drilled by a laboratory drill rig with water circulation at ambient pressure.

Test results obtained with the spherical indentor have been shown accurate in determining penetration rates for tooth bits as well as for button bits. This is probably due to a number

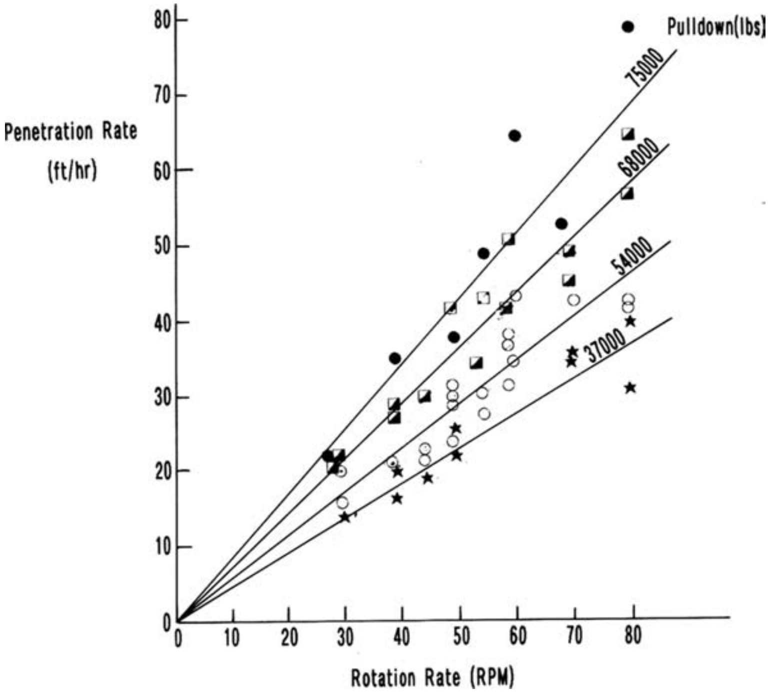


Figure 10.36. Penetration rate as a function of RPM and pulldown force. After Bauer (1967).

of reasons. One, however, is that the basic design criteria for hard formation steel tooth and tungsten carbide button bits is the same.

10.9 PENETRATION RATE – FIELD EXPERIENCE

Some of the earliest and still most relevant work regarding the effect of different operating variables on the penetration rates achieved in roller cone rotary drilling of blast holes in mining was done by Bauer and coworkers in Canada (Bauer (1965), Bauer (1967), Bauer and Calder (1967), Bauer (1971)). Their attention was focused on quantifying field experience primarily gained when drilling in iron formations. As has been discussed earlier, two of the important parameters which affect the penetration rate are the rotation rate and the force acting on the bit. The actual force acting on the bit, the so-called weight on the bit (WOB), is actually a combination of two components.

$$\text{WOB} = \text{pulldown} + \text{weight of the drill string including the bit} \tag{10.12}$$

Typically, only the pulldown term is evaluated in the field tests since it is the part controlled by the operator and it will simply be the pulldown referred to in the rest of this section.

Figure 10.36 shows the effect of the rotation rate on the penetration rate in one iron formation Bauer (1967).

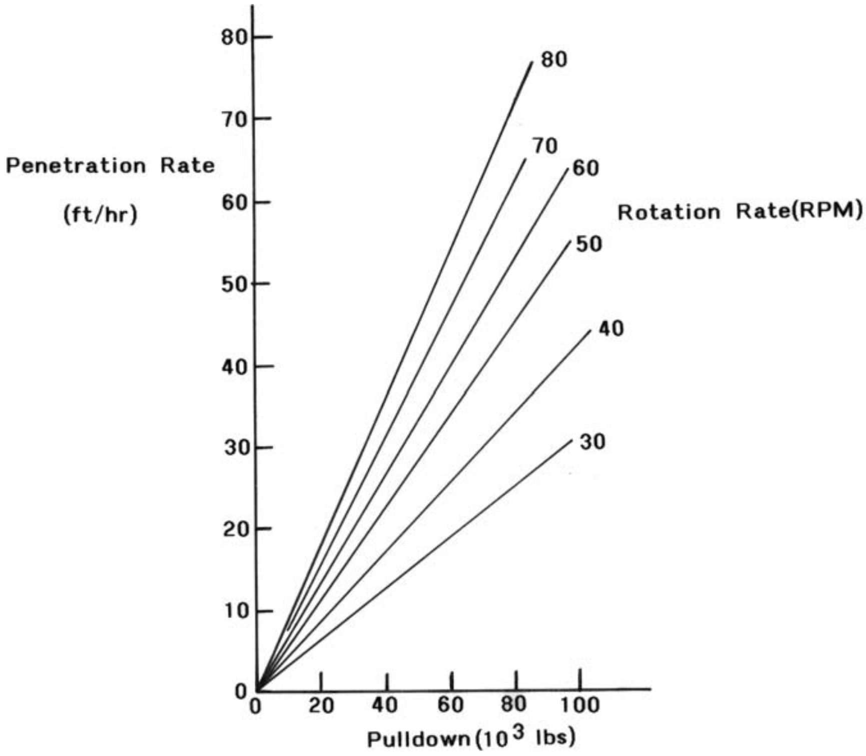


Figure 10.37. Effect of pulldown on the penetration rate as a function of rpm. After Bauer (1967).

As can be seen, the penetration rate is approximately proportional to the RPM at all pulldown values.

$$P_R \propto RPM \tag{10.13}$$

These same basic data have been re-plotted in Figure 10.37 to reveal the effect of pulldown on penetration rate for a constant rotation rate.

As can be seen, the apparent dependence of the penetration rate on the pulldown is also linear.

$$P_R \propto W \tag{10.14}$$

The pulldown (W) can be normalized with respect to the bit diameter φ. Thus,

$$P_R \propto \frac{W}{\phi} \tag{10.15}$$

An overall expression for the penetration rate can be given as

$$P_R = K\left[\frac{W}{\phi}\right] RPM \tag{10.16}$$

where

K = constant reflecting the drillability of the rock.

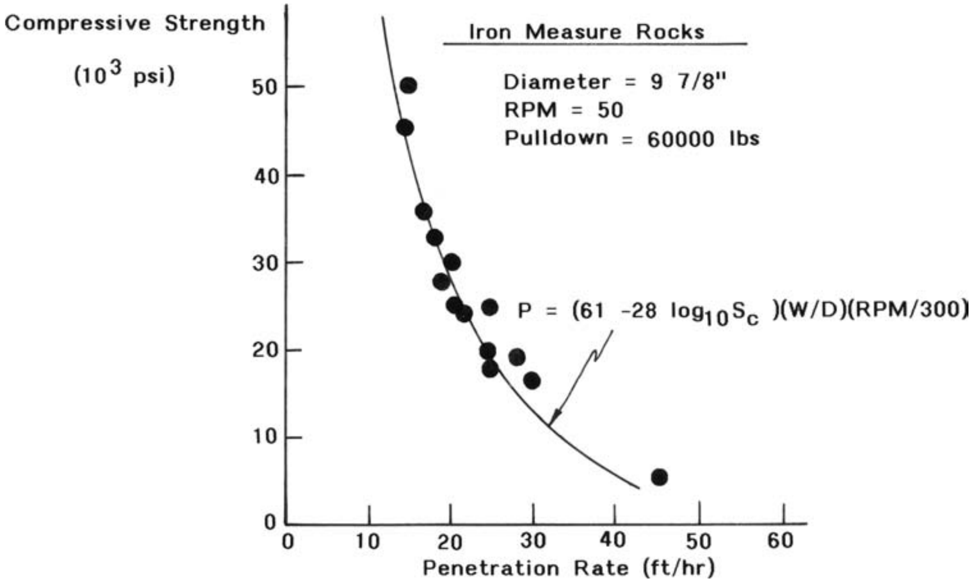


Figure 10.38. Penetration rate as a function of compressive strength for the 50R drill. After Bauer (1967).

Figure 10.38 is a plot of the penetration rate obtained when using a Bucyrus-Erie 50-R (50 rpm, pulldown of 60,000 lbs) machine to drill 9-7/8″ diameter holes in iron ore as a function of the compressive strength. The different points are the production figures from different operating mines and represent hundreds of thousands of feet of drilling.

The uniaxial compressive strength was determined by testing AX-drill core samples with a length to diameter ratio of 2.5:1. Sufficient samples were run to get a statistically meaningful result and the samples were considered to be representative of the different rock types in the pit. A plot of

$$P_R \text{ vs } \log_{10} S_c$$

where

$$S_c = \text{uniaxial compressive strength expressed in } 10^3 \text{ psi}$$

was found to yield a straight line described by the equation

$$P_R = 69.1 - 33.0 \log_{10} S_c \qquad (10.17)$$

This can be converted into the more generalized form

$$P_R = [63.2 - 30.2 \log_{10} S_c] \left[\frac{W}{\phi}\right] \left[\frac{RPM}{250}\right] \qquad (10.18)$$

where

W = pulldown expressed in 1000’s of lbs.

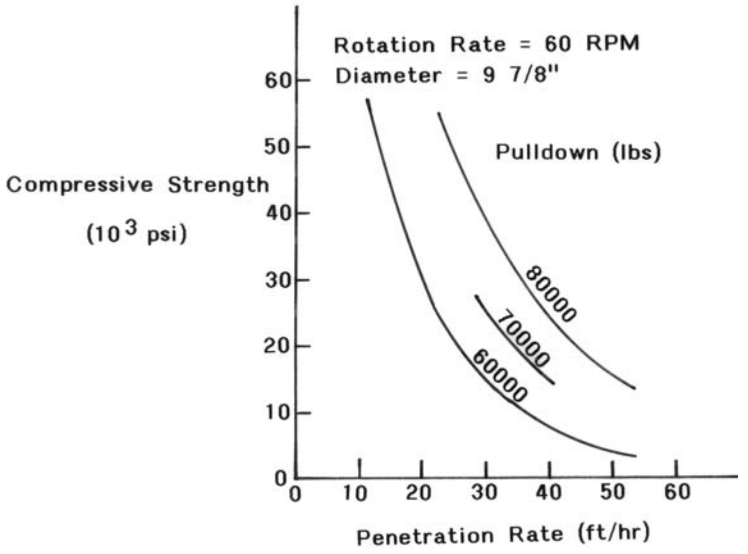


Figure 10.39. Penetration rate as a function of compressive strength for different rotary machines drilling 9-7/8″ diameter holes. Bauer (1967).

Additional data were collected in the same way for the Bucyrus-Erie 60-R drill (60 rpm and 80,000 lbs of pulldown) and the Bucyrus-Erie 45-R drill (60 rpm and 70,000 lbs of pulldown) drilling 9-7/8″ diameter holes in this iron formation. The curves expressing the results are given in Figure 10.39.

For the 60-R results the equation is

$$P_R = [60.2 - 28.1 \log_{10} S_c] \left[\frac{W}{\phi}\right] \left[\frac{RPM}{250}\right] \tag{10.19}$$

and for the 45-R it is

$$P_R = [53.7 - 25.7 \log_{10} S_c] \left[\frac{W}{\phi}\right] \left[\frac{RPM}{250}\right] \tag{10.20}$$

Combining all results one obtains the familiar formula

$$P_R = [61 - 28 \log_{10} S_c] \left[\frac{W}{\phi}\right] \left[\frac{RPM}{250}\right] \tag{10.21}$$

which first appears in the paper by Bauer and Calder (1967). This relationship is shown in nomograph form in Figure 10.40 for a rotation rate of 60 rpm and a recommended weight per inch of bit as shown in Figure 10.41.

In two later papers (Bauer (1971) and Bauer and Crosby (1990)), the formula has been given as

$$P_R = [61 - 28 \log_{10} S_c] \left[\frac{W}{\phi}\right] \left[\frac{RPM}{300}\right] \tag{10.22}$$

The nomograph included in the papers is, however, based upon the use of the 250 constant.

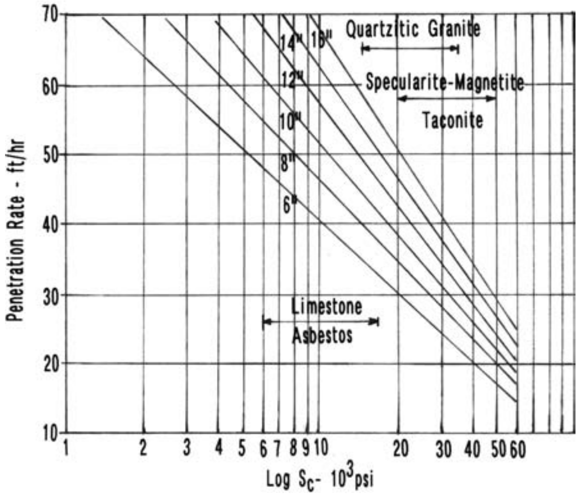


Figure 10.40. Penetration rate versus rock compressive strength for various hole diameters at the recommended weight per inch of bit (Fig. 10.36) and at 60 rpm. After Bauer and Calder (1967).

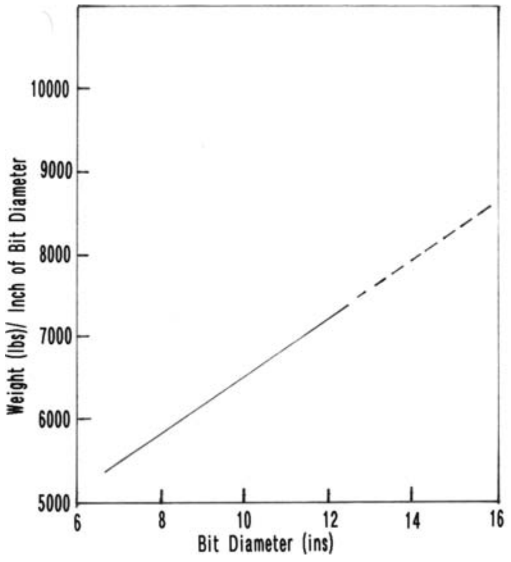


Figure 10.41. Recommended pulldown weight per inch of bit diameter versus bit diameter. Bauer and Calder (1967).

The data upon which formula (10.19) is based were collected in the mid- to late 1960's. Since that time considerable development has occurred in both bits and machines. Figure 10.42 is a more recent (approximately early 1980) nomograph developed by the Smith-Gruner Division of Smith International (Steinke (1983), Lebel (1984)).

The central trend for this relationship can be expressed by a series of straight line segments over a given interval with curves of the same form as given by Bauer.

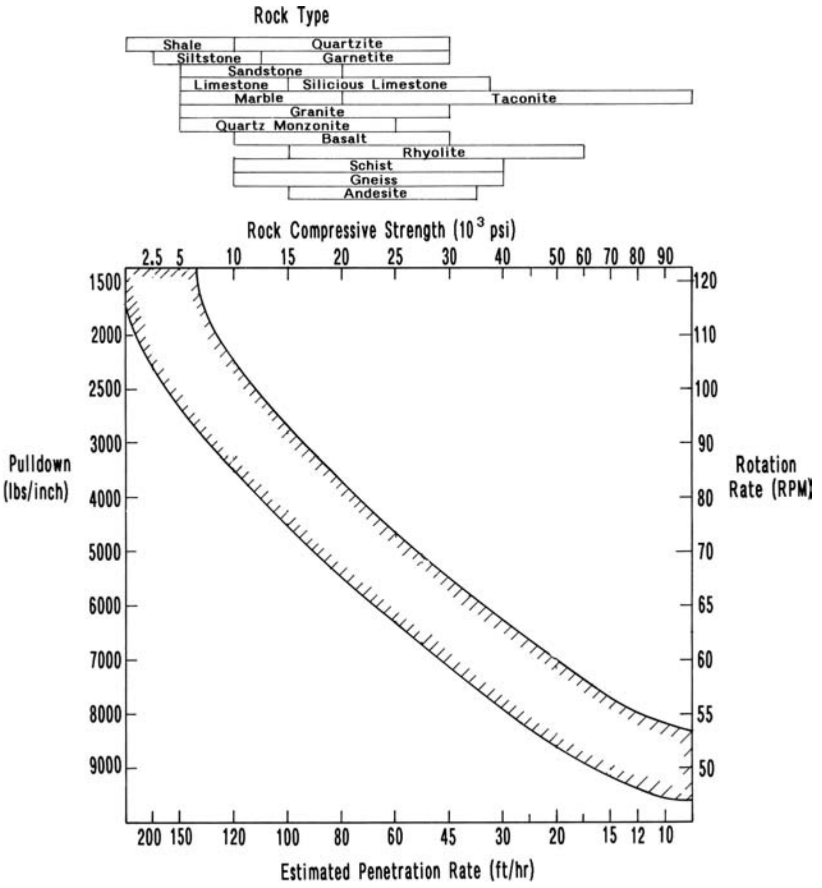


Figure 10.42. Estimating nomograph developed by Smith-Gruner. Steinke (1983), Lebel (1984).

Segment 1: $5000 \leq S_c \leq 15000$ psi

$$P_R = [356 - 217\log_{10} S_c]\left[\frac{W}{\phi}\right]\left[\frac{RPM}{300}\right] \tag{10.23}$$

Segment 2: $15000 \leq S_c \leq 25000$ psi

$$P_R = [378 - 236\log_{10} S_c]\left[\frac{W}{\phi}\right]\left[\frac{RPM}{300}\right] \tag{10.24}$$

Segment 3: $25000 \leq S_c \leq 50000$ psi

$$P_R = [210 - 115\log_{10} S_c]\left[\frac{W}{\phi}\right]\left[\frac{RPM}{300}\right] \tag{10.25}$$

Segment 4: $50000 \leq S_c \leq 100{,}000$ psi

$$P_R = [61 - 28\log_{10} S_c]\left[\frac{W}{\phi}\right]\left[\frac{RPM}{300}\right] \tag{10.26}$$

The equation for segment 4 is identical to that given by Bauer in his later papers.

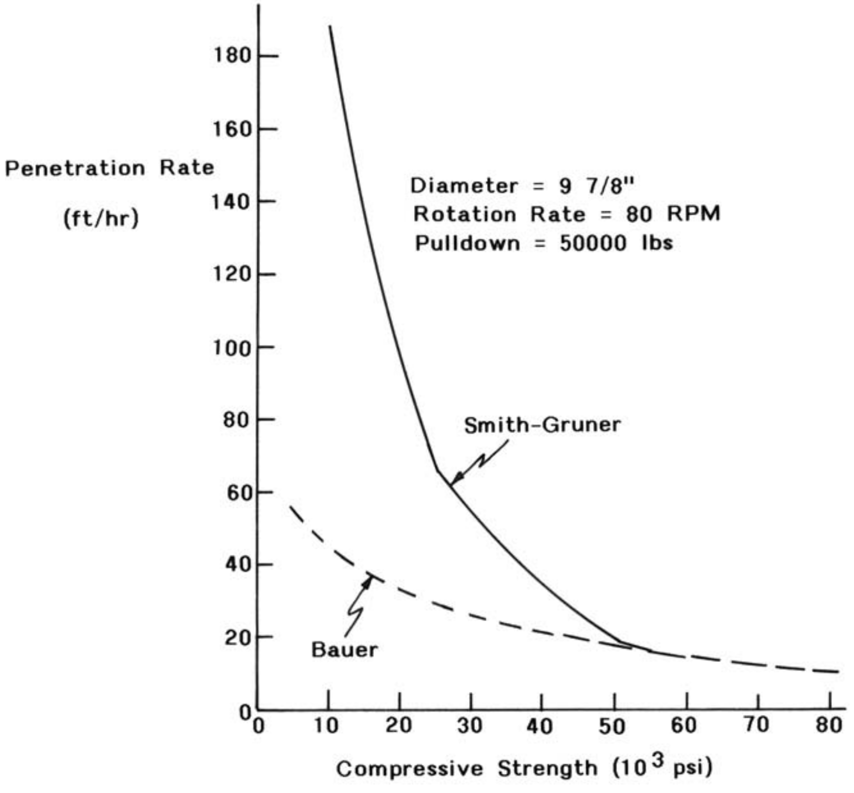


Figure 10.43. Comparison of the Bauer et al. (1967) and Smith-Gruner (1983) curves.

A comparison of the Bauer et al. (1967) and Smith-Gruner (1983) penetration rate – compressive strength curves for the case of

W = 50,000 lbs
$\varphi = 9\text{-}7/8''$
RPM = 80

is given in Figure 10.43.

As can be seen, there is a considerable difference between the results undoubtedly reflecting the change in technology over the intervening 15 years.

In 1996, Workman and Szumanski (1996, 1997) began with the original Bauer (1971) equation

$$P = (61 - 28\log_{10} S_c)\left(\frac{W}{\varphi} \times \frac{RPM}{300}\right) \tag{10.22}$$

where

P = penetration rate (ft/hr)
S_c = uniaxial compressive strength expressed in thousands of psi

Table 10.7. Recommended values of the rock factor for different rock strengths. After Workman and Szumanski (1996, 1997).

Rock	Uniaxial Compressive Strength (MPa)	Rock Factor (RF)
Very hard	>207	84.5
Hard	103–207	104.0
Moderate	69–103	123.5
Soft	34–69	158.0
Very soft	7–34	224.0
Extremely weak	<7	323.5

W/φ = weight per inch of bit diameter expressed in thousands of lb.
RPM = revolutions of drill pipe/minute.

They then wrote equation (10.22) in the following form based on the introduction of SI units

$$P = (84.5 - 28\log_{10} S_c)\left(\frac{W}{\varphi} \times \frac{RPM}{17.6}\right) \tag{10.27}$$

where

P = penetration rate (mm/hr)
S_c = uniaxial compressive strength expressed in MPa
W/φ = pulldown weight expressed in kg/mm of bit diameter.
RPM = revolutions of drill pipe/minute

Workman and Szumanski (1996, 1997) suggest writing equation (10.27) as

$$P = (RF - 28\log_{10} S_c)\left(\frac{W}{\varphi} \times \frac{RPM}{17.6}\right) \tag{10.28}$$

where

RF = rock factor

They note (Workman et al., 1997)

> "This equation (equation (10.27)) has worked well over the years but it is based on studies in hard iron ore. In weaker formations, it predicts penetration rates much lower than those achieved in practice and the deviation is substantial when the compressive strengths are less than 69 MPa. This is believed due to the changing nature of rock breakage under the indentors as the rock strength decreases. But, whatever the formation, penetration rate remains linear with the pulldown and the rotational speed. It is necessary to use different rock factors for different formations. Table 10.7 provides recommended values of the empirical rock factor for different rock strengths instead of the 84.5 factor given in equation (10.27). These values provide reasonable predictions for the rock strengths listed, although it should be noted that variations may occur in specific geological environments."

Unfortunately, the data base has not been included.

Recent discussions with Varel International (2012b) indicate:

1. They recommend beginning with pulldown = 5500 lb/in, RPM = 65–70, and a minimum air pressure drop across the nozzles of 35–45 psi. This applies for all bit diameters and in formations ranging from medium strength to very hard. At a given operation, the rotation rate and weight/in are then adjusted with experience.
2. The Smith-Gruner curves still provide reasonable penetration rate results.

Until new data and curves become available, it appears to the authors that the Smith-Gruner curves may be used to provide an upper limit and the Workman-Szumanski curves a lower limit for the penetration rate.

10.10 PULLDOWN FORCE

From the penetration rate equations given in section 10.9, it can be seen that the penetration rate in a given rock type with a given bit can be increased by increasing both the pulldown and the rotation rate. The appropriate mix of these two is the subject for some discussion. One can first examine the maximum pulldown capacity of a rotary drill. Maximum pulldown force is derived from the working weight distribution of the machine. Design engineers as a rule of thumb attempt to layout machine components such that the maximum pulldown force will equal 50% of the working weight (see Martin (1982), for example). A 10 to 15% weight reserve factor should be added to ensure stability while operating and propelling (Nelmark, 1981).

The minimum pulldown value, as discussed by Morris (1969), could be that given by the threshold value for chipping multiplied by the number of bit elements in contact at any time. Other limiting values between these maximum and minimum pulldown values are imposed by wear considerations for the cutting elements and the bearings.

For a bit of given design, constraints introduced by physical size may impose differences in the maximum load which can be carried as the bit diameter is changed. Obviously, this effect will also be reflected in the permissible load per inch of bit diameter. In Figure 10.41, one observes that the allowable weight/in increases with the bit diameter. Figure 10.42 constructed from data supplied by bit manufacturers in the early 1980's depicts the same basic effect.

In the way of providing an explanation, consider the fact that a 6-inch tricone bit weighs approximately 40 lbs whereas a 12-inch bit weighs 240 lbs (Hughes-Christensen (2012)). It is known that the overall weight of a bit is approximately proportional to the cube of the gauge diameter of a cone. Using this approximation, one would expect that the ratio of the gauge diameters to be 0.55. In actual fact, the gauge diameter of a 6-in bit is about 1/2 that of a 12-in. bit. This ratio also applies to the bit components such as the individual cone assemblies complete with bearings and spindle. As a result, the areal space for the bearings in the smaller (6″) bit is only about 1/4 that of the larger (12″) bit. From a practical point of view, this means that for a given bit type, the allowable amount of pulldown per inch of diameter increases with the diameter. If applicable, Figure 10.44 can be used together with the Smith-Gruner curve for making penetration rate estimations. The question, obviously, is whether this same type of relationship applies today. In this regard, data included in sales material prepared by three major suppliers of TCI roller cone bits for the mining industry (Atlas Copco Secoroc, Sandvik Mining and Varel) have been examined. The general observation

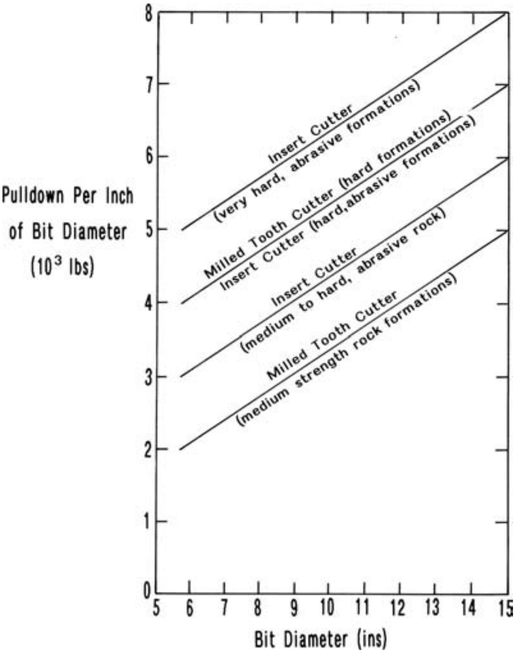


Figure 10.44. Manufacturers recommended maximum bit weights as a function of diameter. Applicable in the early 1980's.

was that there was little if any variation in the suggested bit weight/diameter with diameter for a bit of a particular design. This was most clearly seen in the Atlas Copco Secoroc data which are summarized in Table 10.8.

It is clearly shown that (1) the WOB/in for a given bit design when calculated using the average suggested WOB value is independent of the diameter, and (2) the WOB/in value increases, as expected, with the rock strength application category.

In discussing the issue of pulldown and rotation rate with a number of bit suppliers, the overall procedure appears to begin with the "suggestion" of a WOB range in which to start. Each mine (driller) is then encouraged to explore a range of bit weight settings and rpm values and to then select the one that gives the best overall rate of penetration (ROP) with a reasonable bit life. Obviously good record keeping is essential in this regard. In some cases, mines may end up using 10000 lbs/in of bit diameter because less loading does not give a high enough ROP. In this case, the customer desires a high ROP. Over the years, bit loading has increased by 20–25% over the values provided in Table 10.8 in response to the mine's request for higher ROP.

Whereas the limiting design factor in the early 1980's was the bearings, it appears to be something else today – possibly the insert design and insert strength. The penetration rate for insert-type bits will generally increase with an increase in WOB. But this only continues up to the point where the bit "bogs-down" (the insert is at full penetration and further load is taken by the bit body) or when the hole is not being properly cleaned (Toll, 1981). With regard to the latter, today much higher compressor capacities are being used than earlier and cuttings removal is much improved.

Table 10.8. Summary of average WOB/(in. of bit diameter) for a selection of Atlas Copco Secoroc TCI bits. After Atlas Copco Secoroc (2012).

IADC Code	Diameter (ins)	Diameter (mm)	Operating suggestions		Average WOB (lbs)	Average WOB/in (lbs/in)
			WOB (lbs)	RPM		
4-2-2	6 3/4	171	6750-33750	50-150	20250	3000
	9	229	9000-45000	50-150	27000	3000
	10 5/8	270	10625-53125	50-150	31875	3000
	12 1/4	311	12250-61250	50-150	36750	3000
5-2-2	5 1/8	130	15375-33313	50-150	24344	4750
	6 1/4	159	18750-40625	50-150	29687	4750
	6 3/4	171	20250-43875	50-150	32062	4750
	7 3/8	187	22125-47938	50-150	35032	4750
	7 7/8	200	23625-51188	50-150	37406	4750
	9	229	27000-58500	50-150	42750	4750
	9 7/8	251	29625-64188	50-150	46906	4750
	10 5/8	270	31875-69063	50-150	50469	4750
6-2-2	5 5/8	143	22500-39375	50-120	30938	5500
	6 1/4	159	25000-43750	50-120	34375	5500
	7 3/8	187	29500-51625	50-120	40562	5500
	7 7/8	200	31500-55125	50-120	43312	5500
	8 1/2	216	34000-59500	50-120	46750	5500
	9	229	36000-63000	50-120	49500	5500
	9 7/8	251	39500-69125	50-120	54312	5500
	11	279	44000-77000	50-120	60500	5500
	12 1/4	311	49000-85750	50-120	67375	5500
	13 3/4	349	55000-96250	50-120	75625	5500
	15	381	60000-105000	50-120	82500	5500
	16	406	64000-112000	50-120	88000	5500
7-2-2	4 3/4	121	19000-38000	50-90	28500	6000
	10 5/8	270	42500-85000	50-90	63750	6000
	12 1/4	311	49000-98000	50-90	73500	6000
	13 3/4	349	82500-123750	40-80	103125	7500
	15	381	60000-120000	50-90	90000	6000

10.11 ROTATION RATE (Toll, 1981)

With regard to selecting a rotation rate, it is known that rock has to be given time to react to the indentation. If the indentation episode is too short (too high rotation rate), the cracks do not have time to extend very far before the load is removed. Hence, deep cracking will not occur and little, if any, crack propagation from the current crater to previous, adjacent craters will not take place. To achieve high penetration rates, the creation of large chips is desired/required. Thus, sufficient dwell time is a prerequisite. It has been found that "elastic" rock requires more indentor time whereas "brittle" rock requires less time. In high strength, high modulus rock, a higher WOB and lower RPM generally gives a better ROP (Atlas Copco, 2012a).

Outside of rig limitations, the upper limit of rotary speed is determined by two factors

- bearing wear
- insert breakage

Table 10.9. Penetration rate values from the 1980's used in conjunction with the bit life estimation rule.

Rock Hardness	Penetration Rate (ft/hour)				
	9″	9 7/8″	10 5/8″	12 1/4″	15″
Soft	60	70	90	100	120
Medium	50	60	70	80	90
Hard	35	45	55	65	70
Very Hard	20	25	30	40	50

Bearing wear varies with load, hole cleaning, temperature, and the total number of revolutions. At higher rpm, bearing temperatures would be expected to be higher and, if bit life is measured in units of time, the life in revolutions would be reached sooner at faster rotating speeds. Insert or tooth breakage generally results from the impact magnitude and the impact forces generally go up as the square of the velocity. Thus, doubling the rotary speed tends to increase the impact effects by a factor of four. When using TCI bits, the rotary speed is especially critical. Although tungsten carbide is a very hard, wear resistant material, it is also somewhat brittle. High rotary speeds usually cause excessive insert breakage.

10.12 BIT LIFE ESTIMATES

The authors, unfortunately, have been unable to find anything of a quantitative nature which can be used to estimate bit life in the current literature. Hopefully by including the following information from the 1980's, companies/researchers will be encouraged to assemble their information in such a form that it can be included in the next edition of this book. The rule is simply

> *"A rough estimate of the bit life (distance drilled) can be obtained by using a life factor of 80 hours."*

The average penetration rates as a function of bit diameter and rock hardness applicable from the 1980's which were intended to be used together with this rule are provided in Table 10.9.

One experience point from today suggests that in the sandstone layers (10,000–15,000 psi compressive strength) overlying eastern U.S. coal, the 6-3/4″, 7-7/8″ and 9″ diameter bits being used typically drill 10,000–30,000 ft of hole and fail from cone erosion. They achieve a penetration rate in the range of 150–450 ft/hr (Atlas Copco, 2012a). Table 10.9 would suggest a ROP of 50 ft/hr in medium strength rock when using a 9″ diameter bit. The corresponding life would be 4000 ft when applying the 80 hr rule. In actual fact the average ROP is of the order of 300 ft/hr which is greater than the table value by a factor of 6. Applying the 80 hr life factor to the actual ROP, one would estimate the life at 24,000 ft which is of the order observed.

Another experience factor from today suggests a life of about 1600 ft for a 9″ diameter bit drilling very hard rock. This is similar to what would be predicted using the table (Atlas Copco, 2012a).

Since there has been a major improvement in bits over the intervening years and operating conditions have changed (trend to higher RPMs, bailing velocities and air volumes), it is clear that new rules are in place. This would appear to be a fruitful area of applied research.

10.13 TECHNICAL TIPS FOR BEST BIT PERFORMANCE

Some technical tips for best bit performance are provided below (Atlas Copco, 2011g):

1. Exercise care in making-up and breaking out the bit to avoid damage to threads of bit and drill steel.
2. When a new bit is installed, drill at a reduced weight for a short break-in period.
3. Provide adequate air to the bit to insure trouble-free bearing performance and reduced abrasion wear on cone teeth and shirttails.
4. Always open the air valve before lowering the bit to collar the hole and keep air on until bit is out of the hole. Always rotate when moving in or out of hole.
5. In wet holes or where water injection is used for dust control, maintain as high a pressure drop across the bit air courses as possible and minimum amount of water.
6. Periodically inspect the bit and feel the cones to be sure that all are about the same temperature. One hot cone generally indicates that the air passages to that particular bearing have become obstructed.
7. Never drop the bit while on the end of the drill steel.
8. When a partially dull bit sits idle for a shift or longer, rotate the cones by hand to insure that they turn freely before drilling. If the bit is laid up for a period of time, bearings should be coated with a light oil.
9. Occasionally check the air pressure with the bit off to insure that there are no obstructions in the hose, swivel or steel.
10. Properly maintain the drill steel and its thread connections. A bent steel will often cause early bit failure.

Because of the complexity of factors influencing bit life, estimates based on prior experience in similar ground are usually good.

10.14 CUTTINGS REMOVAL AND BEARING COOLING

In rotary drilling, air of sufficient volume and at the proper pressure must be delivered to the bit to achieve the desired objective, i.e. optimum bit life and/or maximum penetration rate and/or some combination.

The air from the compressor goes to a swivel at the top of a drill string. The air travels down the inside of the drill pipe to the bit. At the bit there are two main passages for the air to travel prior to exhausting to the atmosphere. One path, as shown in Figure 10.21, is through air tubes which carry a portion of the air through channels cut in the bit to the different bearings. This air is exhausted to the atmosphere once it passes the bearings.

The other path for the air to take is through the nozzles. By these nozzles, the air is directed along the cutter path helping to clean the teeth and to remove the broken material thereby allowing the cutters to attack a fresh rock surface. The spent air is deflected up the outside of the bit carrying its load of cuttings in the annulus between the drill pipe and the hole wall. It is obvious in viewing Figure 10.21 that the most difficult path for the air to take in trying to reach atmospheric pressure is by way of the bearings. By varying the diameter of the nozzles used, more or less air can be forced into the bearings for optimal bearing cooling and cleaning.

In order to insure maximum bearing life, approximately 25% of the circulating or bailing air is diverted to the bearings to keep cuttings flushed out and to provide cooling to critical bearing surfaces such as the thrust bearing. Without this important cooling air, blasthole bit bearings would overheat and seize almost immediately. Also highly abrasive material entering the bearings would accelerate bearing wear and cause premature failure. In order to protect the air bleeder holes to the bearings, a check valve is sometimes installed in the pin of the bit or check values in the jets themselves. These prevent plugging in the event that the air is turned off between drilling operations such as when making connections (Chitwood, 1970).

The air passages through rolling-cutter bits to the bottom of the hole take three forms

- single air nozzles
- drilled air courses
- jet nozzles

Those with a single air passage usually have a special nozzle which distributes the air over the hole bottom. Drilled air courses consist of three holes drilled through the bit body near the center of the bottom of the bit. Jet air courses also have three passages but these are located between the cones near the outer diameter of the bit body.

One simple expression for the rate at which air will flow through a nozzle is given by

$$\mathrm{Q_n = 14.72aP} \tag{10.29}$$

where

$\mathrm{Q_n}$ = volume of free air passing through a nozzle (cfm)
a = orifice area (in.^{2})
P = upstream total pressure (psia)

Since the compressor is located at the surface, there is a pressure loss ($\mathrm{P_L}$) between the compressor and the bit. Thus,

$$\mathrm{P = P_a - P_L} \tag{10.30}$$

where

$\mathrm{P_a}$ = absolute pressure measured at the compressor (psia)
$\mathrm{P_L}$ = pressure loss (psi)

As indicated, at the bit a percentage of the total air will be directed through the bearings rather than through the nozzles (presuming an open rather a sealed bearing design). This amount will vary but is usually between 20 to 35% depending on the bit bearing design and the bearing condition. Worn bearings pass 20–30% more air than new bearings.

Thus,

$$Q_n = (1 - x)Q_b \tag{10.31}$$

where

x = the ratio of the volume directed to the bearings
Q_b = total volume of free air going to the bit (cfm)

The conversion between the absolute pressure (psia) and the gage pressure (psig) is

$$P_a(\text{psia}) = P_g(\text{psig}) + 14.7 \tag{10.32}$$

Substituting equations (10.30), (10.31) and (10.32) into equation (10.29) one finds that

$$Q_n = (1 - x)Q_b = 14.72\left(\frac{\pi}{4}\right)n^2(P_g + 14.7 - P_L) \tag{10.33}$$

Simplifying yields

$$n^2 = \frac{4}{\pi}\frac{(1 - x)Q_b}{14.72(P_g + 14.7 - P_L)} \tag{10.34}$$

Assuming 30% of the air volume is directed through the bearings and a 10 psi line loss, one finds that for a single nozzle

$$n^2 = \frac{Q_b}{16.36(P_g + 4.7)} \tag{10.35}$$

For 3 nozzles or jets, one obtains

$$n^2 = \frac{Q_b}{49.08(P_g + 4.7)} \tag{10.36}$$

where

n = diameter of nozzle (ins)
Q_b = volume of free air going to the bit (cfm)
P_g = pressure at the compressor (psig)

Table 10.10 gives the flow through a single nozzle evaluated using equation (10.36) for common compressor pressures and nozzle sizes.

With bits having three air courses or jets, the total volume would be three times that shown. It must be kept in mind that additional air also goes through 3 sets of bearings. Therefore Table 10.10 will not give correct "compression gage pressure readings." It will give higher values than the actual. It will, however, be correct for sealed bearings where there is no air going through the bearings. The different bit manufacturers present tables similar to Table 10.10 for their bits (see for example reference Varel (2012a) for Varel bits and references Sandvik (2012a and 2012b) for Sandvik bits.

Table 10.11 presents some logical combinations of bits and nozzles.

Although the simplified calculations used to develop Table 10.10 are interesting, the actual values depend upon the bit design. Table 10.12 presents the air pressure drop table as a function of air volume for Atlas Copco Secoroc roller cone blasthole bits (Atlas Copco

Table 10.10. Nozzle flows. Smith-Gruner (1981).

Nozzle Diameter		Diameter squared	Nozzle Area	CFM per nozzle – Compressor Gage Pressure (psig)					
(n)	(decimal)	n*n	(in)^2	40	50	60	90	100	110
5/16	0.31	0.098	0.077	71	87	103	151	167	183
3/8	0.38	0.141	0.111	103	126	149	218	241	264
13/32	0.41	0.165	0.130	121	148	175	256	283	310
7/16	0.44	0.191	0.151	140	171	203	297	328	359
1/2	0.50	0.250	0.197	183	224	265	387	428	469
9/16	0.56	0.316	0.249	231	283	335	490	542	594
5/8	0.63	0.391	0.307	286	350	413	605	669	733
3/4	0.75	0.563	0.442	411	503	595	871	964	1056
13/16	0.81	0.660	0.519	483	591	699	1023	1131	1239
7/8	0.88	0.766	0.602	560	685	810	1186	1311	1437
1	1.00	1.000	0.787	731	895	1058	1549	1713	1876
1 1/8	1.13	1.266	0.995	926	1133	1340	1961	2168	2375
1 1/4	1.25	1.563	1.229	1143	1398	1654	2421	2676	2932
1 3/8	1.38	1.891	1.487	1383	1692	2001	2929	3238	3548
1 1/2	1.50	2.250	1.770	1645	2014	2382	3486	3854	4222
2	2.00	4.000	3.146	2925	3580	4234	6197	6852	7506
2 1/8	2.13	4.516	3.552	3302	4041	4780	6996	7735	8474
2 1/4	2.25	5.063	3.982	3702	4530	5359	7843	8672	9500
2 1/2	2.50	6.250	4.916	4571	5593	6616	9683	10706	11728

Secoroc, 2012). Tables such as these are used for selecting the appropriate nozzle size. Note that this table compensates for the air going to the bearings.

The following steps are used when selecting nozzles.

1. Establish the approximate air delivery volume and operating pressure for the air compressor used. The condition of the compressor, efficiency, and altitude should be taken into consideration when estimating these values.
2. Ten (10) psi should be subtracted from the above operating pressure for pressure loss through the surface equipment and drill steel between the compressor and bit. This becomes the corrected air pressure.
3. From Table, choose the "Air Volume Delivered" column nearest the volume established in Step 1.
4. Proceed down the correct "Air Volume Delivered" column to the proper "Bit Size Range" for the bit being used.
5. Select the smallest nozzle size within a given bit size range that can be used without exceeding the corrected air pressure delivered to the bit.

Two examples are given to illustrate the procedure:

Example 1: a. Bit size: 12-1/4″. b. Air volume delivered: 1800 cfm. c. Compressor operating pressure: 75 psi. d. Corrected air pressure: 65 psi. (75-10). From Table, select three 9/16″ jet nozzles (59 psi).

Example 2: a. Bit size: 9-7/8″. b. Air volume delivered: 1600 cfm. c. Compressor operating pressure: 60 psi. d. Corrected air pressure: 50 psi (60-10). From Table, select three 3/4″ jet nozzles (47 psi).

The second purpose of the compressed air introduced into the drill hole is to clean the hole bottom and to remove the cuttings. The correct bailing or flushing velocity depends on hole conditions and rock density. It should be the highest when

- the rock is dense and/or
- cuttings are wet and/or
- the penetration rate is high

To determine the return air velocity required to carry cuttings of a particular size and density out of the borehole, the following relationship can be used:

$$V = 54600\left(\frac{\rho}{\rho + 62.4}\right)D^{0.6} \tag{10.37}$$

Table 10.11. Nozzle selection chart. Atlas Copco (2011f).

Bit size range (ins)	API pin size (ins)	Air course size (ins) – 3 each
5 to 6	2 7/8 to 3 1/2	5/16 3/8 1/2 9/16
6 1/4 to 7 3/8	3 1/2	5/16 3/8 7/16 1/2 9/16
7 7/8 to 9	4 1/2	3/8 7/16 1/2 9/16 5/8 11/16 3/4
9 7/8 to 11	6 5/8	3/8 7/16 1/2 9/16 5/8 11/16 3/4 7/8 1
12 1/4 to 15	6 5/8 to 7 5/8	7/16 1/2 9/16 5/8 11/16 3/4 7/8 1 1 1/8 1 1/4

where

V = minimum air velocity (fpm)
ρ = rock density (lb/ft^3)
D = chip diameter, ft

Although this formula must be considered as giving only approximate values, it does show how the required air velocity varies with rock density and chip size.

Table 10.12. Air requirements and nozzle selection table for Atlas Copco Secoroc bits. Atlas Copco 2011f.

Bit size range (ins)	API Pin size (ins)	Air course size (ins) – 3 each	Pressure drop as a function of nozzle size and delivered air volume										
			400	500	600	700	800	900	1000	1100	1200	1300	1400
5 to 6	2 7/8 to 3 1/2	5/16	47	62	77								
		3/8	35	47	59	71							
		7/16	25	35	45	55	65	75					
		1/2	18	26	34	42	50	58	66	74			
		9/16	11	18	24	31	38	44	58	58	64	71	
6 1/4 to 7 3/8	3 1/2	5/16	42	52	62	72	81						
		3/8	33	43	51	61	69	78					
		7/16	27	34	41	48	57	65	73	79			
		1/2	23	29	33	41	48	54	61	67	73	79	
		9/16	18	23	29	34	41	47	51	56	62	67	73
7 7/8 to 9	4 1/2	3/8	27	36	45	55	66	75	83				
		7/16	21	28	35	42	49	55	63	69	75	81	
		1/2		21	27	33	39	45	51	59	67	76	84
		9/16			20	26	32	37	43	49	55	61	67
		5/8				21	26	31	36	41	47	52	57
		11/16					20	25	29	34	39	44	50
		3/4						21	25	29	34	37	41
9 7/8 to 11	6 5/8	3/8	26	36	46	54	62	70	77				
		7/16	19	27	35	42	50	58	65	72	79		
		1/2		21	27	33	39	45	53	60	66	71	77
		9/16			20	26	32	38	43	49	54	59	64
		5/8				19	25	32	36	41	46	49	53
		11/16					20	24	29	34	39	43	47
		3/4						19	22	26	31	36	40
		7/8								20	24	26	30
		1											19
12 1/4 to 15		7/16			19	25	30	35	41	46	53	58	63
		1/2				18	23	27	33	38	43	47	52
		9/16					19	23	27	31	34	38	42
		5/8						19	22	25	27	31	34
		11/16								20	23	26	29
		3/4									19	22	25
		7/8										17	19
		1											
		1 1/8											
		1 1/4											

(*Continued*)

Table 10.12. (Continued).

Bit size range (ins)	API Pin size (ins)	Air course size (ins) – 3 each	Pressure drop as a function of nozzle size and delivered air volume										
			1500	1600	1700	1800	1900	2000	2200	2400	2600	2800	3000
5 to 6	2 7/8 to	5/16											
	3 1/2	3/8											
		7/16											
		1/2											
		9/16											
6 1/4 to	3 1/2	5/16											
7 3/8		3/8											
		7/16											
		1/2											
		9/16	79										
7 7/8 to	4 1/2	3/8											
9		7/16											
		1/2											
		9/16	73	80									
		5/8	62	69	73	79							
		11/16	55	60	65	71	77						
		3/4	47	51	55	60	65	70	79				
9 7/8 to	6 5/8	3/8											
11		7/16											
		1/2											
		9/16	68	73	78								
		5/8	58	62	66	70	74	78					
		11/16	51	54	58	62	66	70	78				
		3/4	43	47	50	54	57	61	68	75	79		
		7/8	32	35	38	41	44	46	52	59	63	69	75
		1	21	23	25	28	30	33	38	42	47	52	57
12 1/4 to		7/16	69	75									
15		1/2	56	60	65	70	75						
		9/16	46	50	55	59	63	67					
		5/8	38	42	46	49	53	57	64	72			
		11/16	32	35	39	42	45	48	55	62	66	70	
		3/4	28	31	34	37	40	42	48	53	57	61	65
		7/8	21	23	25	27	28	30	35	40	42	44	47
		1				17	19	21	25	29	33	37	41
		1 1/8							19	25	27	29	31
		1 1/4								17	19	23	25

The air volume necessary to provide the required rising velocity in the annular area between the drill stem and the hole wall should be

$$Q_c = VA \tag{10.38}$$

where

Q_c = required compressor capacity (cfm of free air)
V = velocity in fpm
A = annular area

Table 10.13. Required annular velocities as a function of material type.

Material type	Annual Velocity (ft/min)
light	3500
average	5000
heavy	7000
west and/or heavy, and/or high penetration rates	9000

Using a round a drill stem equation (10.38) becomes

$$Q_c = 0.0054V(D^2 - d^2) \tag{10.39}$$

where

D = diameter of hole (ins)
d = outer diameter of drill pipe (in.)

For a square drill stem equation (10.38) becomes

$$Q_c = \frac{V(0.785D^2 - y^2)}{144} \tag{10.40}$$

where

y = width of square drill stem (ins.)

As a first approximation, sufficient air volume should be provided to produce the annular velocities given in Table 10.13.

Tables 10.14 and 10.15 give the bailing air volumes required to produce annular velocities of 5000 ft/min and 7000 ft/min, respectively, for common bit and drill pipe combinations. Velocities higher than 9000 fpm may cause sandblasting of drill pipe.

Figure 10.45 presents a simple nomograph which can be used as part of the selection process. The example shown is for a 9-7/8″ diameter bit, a 7″ OD drill stem and an assumed 5000 fpm. As read, the required air volume is about 1330 cfm. As expected, this is in good agreement with the value of 1323 cfm given in Table 10.14.

As the cuttings reach the surface, the heavier chips normally fall out around the drill hole and the lighter particles are collected for disposal. If a primary dust collection system is used, a flexible shroud surrounds the drill hole and most of the smaller airborne particles are collected by a vacuum system and deposited in hoppers. If a secondary system is used, the dust-laden air is run through a cyclone separator and all but the finest particles are separated out.

Water injection into the air stream is another way to reduce dust. Unfortunately it also reduces roller bit bearing life and can cause buildup and damage to the exposed parts of the bit.

This section has focused on the most common blasthole bit designs in which the journal bearings are cooled and maintained clean via the passage of the compressed air stream. Sandvik (2012d) supplies sealed journal bearing bits for blasthole drilling. These bits have been designed exclusively for the surface mining industry. Sandvik (2012d) suggests that "when bearings are the limiting factor for rotary air bearing bits, the use of sealed bearing bits can be the key to improved bit life, increased productivity and lower costs. Sealed bearing bits can deliver more than two times the bearing hours than air bearing bits. However

Table 10.14. Bailing air volumes required to produce 5000 fpm for common bit and drill pipe combinations. Atlas Copco 2011f.

Hole diameter (ins)	Pipe diameter (ins)	Air volume (CFM)	Hole diameter (ins)	Pipe diameter (ins)	Air volume (CFM)
4 1/2	2 7/8	327	9	4 1/2	1665
	3 1/2	218		5 1/2	1383
	4	116		6 5/8	1063
4 3/4	2 7/8	390		7	873
	3 1/2	282		7 3/4	570
	4	178	9 7/8	7	1323
5 1/8	2 7/8	491		7 3/4	1022
	3 1/2	382		8 5/8	627
	4	280		9	450
5 5/8	2 7/8	637	11	7	1964
	3 1/2	530		7 3/4	1662
	4	426		8 5/8	1272
6 1/4	3 1/2	732		9	1090
	4 1/2	513	12 1/4	8 5/8	2063
	5	382		9	1882
6 3/4	3 1/2	908		10	1365
	4	805		10 3/4	941
	4 1/2	690	13 3/4	10	2429
	5	560		10 3/4	2004
7 3/8	3 1/2	1358	15	10	3409
	4 1/2	932		10 3/4	2985
	5 1/2	658		12	2209
7 7/8	3 1/2	1358		13	1527
	4 1/2	1138	17 1/2	10	3743
	5 1/2	867		14	3007
	6 1/2	625		16	1370
	6 5/8	493			
	7	355			

to realize the benefits of using a sealed bearing bit, there must be sufficient bit structure remaining on an air bearing bit to realize the increased performance."

This design may offer certain special possibilities for the surface miner and should be seriously considered.

10.15 PRODUCTION TIME FACTORS

To estimate the production which might be expected from a drill rig, the following factors are of use (Atlas Copco, 2012a):

- Propel speed ranges from about 0.7 to 1.0 mph (60 to 90 feet per minute)
- The time to move from one hole to another if the mast must be lowered and set up is of the order of 9–11 minutes. It is broken down as follows:
  - Lower mast = 3–4 minutes
  - Jacks up = 1 minute
  - Tram and locate collar = 2–3 minutes
  - Jacks down and leveling = 1 minute
  - Raise mast and lock = 2–3 minutes

Table 10.15. Bailing air volumes required to produce 7000 fpm for common bit and drill pipe combinations. Atlas Copco 2011f.

Hole diameter (ins)	Pipe diameter (ins)	Air volume (CFM)	Hole diameter (ins)	Pipe diameter (ins)	Air volume (CFM)
4 1/2	2 7/8	458	9	4 1/2	2331
	3 1/2	305		5 1/2	1936
	4	162		6 5/8	1488
4 3/4	2 7/8	546		7	1222
	3 1/2	395		7 3/4	798
	4	249	9 7/8	7	1852
5 1/8	2 7/8	687		7 3/4	1431
	3 1/2	535		8 5/8	878
	4	392		9	630
5 5/8	2 7/8	892	11	7	2749
	3 1/2	742		7 3/4	2323
	4	596		8 5/8	1779
6 1/4	3 1/2	1025		9	1526
	4 1/2	718	12 1/4	8 5/8	2888
	5	535		9	2635
6 3/4	3 1/2	1271		10	1911
	4	1127		10 3/4	1317
	4 1/2	966	13 3/4	10	3400
	5	784		10 3/4	2806
7 3/8	3 1/2	1900	15	10	4772
	4 1/2	1305		10 3/4	4179
	5 1/2	921		12	3093
7 7/8	3 1/2	1900		13	2138
	4 1/2	1503	17 1/2	10	5240
	5 1/2	1214		14	4210
	6 1/2	875		16	1918
	6 5/8	690			
	7	497			

- Time to make a pipe change. A skilled driller adds a single pipe in 1.5 minutes and removes a pipe in 1.5 minutes.
- Hoist speeds generally are limited to 60 to 100 feet per minute
- When leveling the machine, the hydraulic systems are sized to do this in 30 seconds to 1 minute.

The penetration rates discussed earlier represent the footage drilled during the time actually spent drilling. Many operators keep track of drilling rates on a shift basis. Normally only 65–70 per cent of the time a drill is manned will be spent drilling. The rest of the time will be spent moving, changing bits, etc.

10.16 COST CALCULATIONS

The following formula (Atlas Copco, 2011f) can be used to calculate the operating cost per unit length of hole

$$C = \frac{B}{F} + \frac{D}{P_R} \tag{10.41}$$

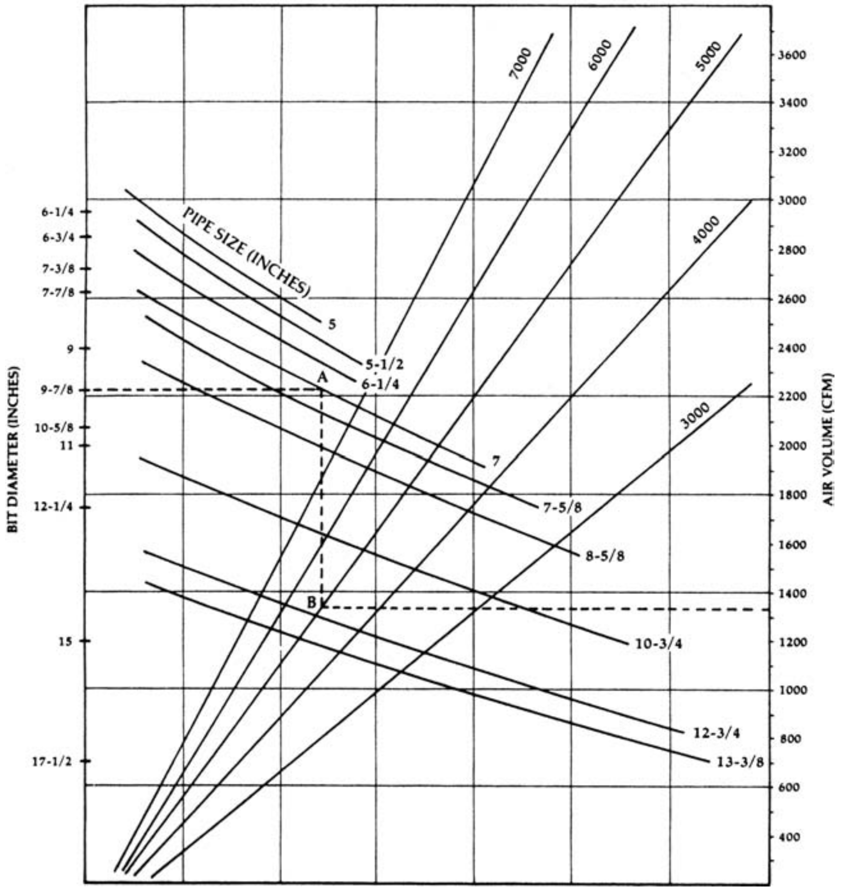


Figure 10.45. Nomograph for use in calculating air volume. Martin (1982).

where

C = operating cost in dollars per foot
B = bit cost ($)
F = bit life (ft)
D = Drill operating cost in dollars per hour
P_R = penetration rate (ft/hr)

Consider the following example

B = $6500 (9 7/8″ diameter TCI bit)
F = 3000 ft
D = $400/hour
P_R = 80 feet/hour

The cost per foot of hole becomes

$$C = \frac{B}{F} + \frac{D}{P_R} = \frac{6500}{3000} + \frac{400}{80} = 2.16 + 5 = \$7.16/\text{ft}$$

In this particular case, the drill operating cost per foot of hole is more than twice that associated with the bit. One can write equation (10.41) in the form

$$C = \frac{B}{F} + \frac{D}{P_R} = C_1 + C_2 \tag{10.42}$$

where

C_1 = bit cost component
C_2 = operating cost component

Both the bit life (F) and penetration rate (P_R) depend upon operating parameters such as the pulldown and rotary speed. With increasing pulldown and rotary speed, the penetration rate increases thereby decreasing cost component C_2. With increasing pulldown and sometimes with increasing rotary speed, the bit life goes down, thereby increasing cost component C_1. There is obviously an optimum set of operating parameters which provide an overall minimum cost per length drilled. To do that one needs to know about the factors involved in bit life and penetration rate. It must be pointed out that if ownership costs of the drill are included in cost C_2, the penetration rate then becomes a very important consideration. It is much more important than the bit life since the ownership costs are often the same order of magnitude as the operating costs.

10.17 DRILL AUTOMATION

Manufacturers are adding many computer-assisted, computer-monitored, and computer-controlled functions to the drill rigs (Atlas Copco, 2011a). One such system records key parameters and monitors changes in material hardness by correlating penetration rate, torque, speed and pulldown. Results can be printed, stored or relayed by radio. Others can be used to

- automatically level the machine
- control startup, operation and shutdown
- monitor key maintenance parameters

As part of their "Mine of the Future" project, Rio Tinto (Rio Tinto, 2012) has indicated that they expect that all vehicles and equipment at the mines will be self-guiding and self-controlling with minimal direct human intervention (Grad, 2010). With respect to the drills, an autonomous equipped drill will move smoothly from one hole location to the next. They will combine GPS navigation and track encoders to position themselves, tilt-meters to jack and level themselves, and drilling sensors. A position accuracy of half a hole diameter and a depth accuracy of 10 mm is to be achieved (Grad, 2010).

The future offers many interesting possibilities and realities (Fiscor, 2009).

REFERENCES AND BIBLIOGRAPHY

Aboujaoude, C.E. 1991. Modeling simulation and control of rotary blasthole drills. M.E. Thesis, Dept of Electrical Eng., McGill University, Montreal, 116 pp.

Altindag, R. 2002. The evaluation of rock brittleness concept on rotary blasthole drills. JSAIMM. 102:61–66. Jan/Feb.
Altindag, R. 2010. Assessment of some brittleness indexes in rock-drilling efficiency. Rock Mech. and Rock Eng. 43(3):361–370.
Amundsen,C.O. 1968. Rotasjonsboring i Syd-Varanger. Ekamensarbetet. Institute for Gruvedrift, NTH, Trondheim. In Norwegian.
Amundsen, C.O. 1974. Rotasjons-og full profil- boring. Teknisk rapport nr 16/2. Bergverkenes Landssammenslutnings Industrigruppe, Trondheim. In Norwegian.
Anonymous. 1964. Excavation – Scientific and Technical Applications Forecast. Office of the Chief of Research and Development, Department of the Army, Washington, D.C. (Department of Commerce).
Anonymous. 1980. Rotary blast-hole drills. Mining Magazine. 142(3):222–239.
Anonymous. 1982. Drilling for profit. World Mining. 35(2):40–41.
Anonymous. 1982. Drills: Manufacturers are focusing on larger blasthole drills to reduce unit drilling costs. E/MJ. 183(6:126–131A).
Anonymous. 1987. Large diameter blasthole drills face the future. World Mining Equipment. 12(1): 8–10, 12, 61–62.
Anonymous. 1988. Blast hole drills for surface mines. E&MJ. 189(1):58–65.
Anonymous. 1988. Drilling large-diameter inclined blast holes. E&MJ. 189(1):56–57.
Anonymous. 1990. Designed for productivity. International Mining. 7(Mar):12–18.
Anonymous. 1992. International equipment guide: blasthole drills. World Mining. 45(2):33–39.
Anonymous. 1994. Advanced new P&H blasthole drill. Mining Journal, London. 322(Sept 30):239.
Anonymous. 1995. Drilling/blasting forum. E&.M/J. 196(3):ww95.
Anonymous. 2006. Drill monitor systems tell the hole story. E/MJ. 207(10):44–47. December.
Anonymous. 2006. Equipment – Surface drilling: The appliance of drilling. Mining Magazine. September. pp. 45–54.
Anonymous. 2007. Equipment – Surface drilling: Rotary versus crawler. Mining Magazine. September. pp. 39–51.
Anonymous. 2007. What's new in blasthole drilling. E/MJ. 208(6):60–63. July/August.
Appelgren, J., L.M. Eriksson, & D. Penn. 2010. Rotary drilling automation increasing operator productivity and drilling quality. Paper 10-143. SME Annual Meeting. Feb 28–Mar 3. Phoenix, AZ.
Atlas Copco Secoroc. 2012. Blasthole Drilling Handbook. 40 pp.
Atlas Copco. 1989. Surface Drilling: Know-how and Equipment. pp. 98.
Atlas Copco. 2011a. Automated surface blasthole drilling. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 29–33.
Atlas Copco. 2011b. Tricone rotary blasthole drilling. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 35–37.
Atlas Copco. 2011c. Optimizing the rotary drill string. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 39–40.
Atlas Copco. 2011d. Rotary blasthole drills. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 147–169.
Atlas Copco. 2011e. Tricone rotary blasthole drilling. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 190–209.
Atlas Copco. 2011f. Air practices. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 210–219.
Atlas Copco. 2011g. Guides for best bit performance. In Blasthole Drilling in Open Pit Mining. 2nd Edition. Atlas Copco Drilling Solutions LLC, Garland, TX. pp. 223–225.
Atlas Copco. 2012a. Personal communication with Clarence Zink. June–July.
Atlas Copco. 2012b. Personal communication with Ulf Linder. July.
Atlas Copco. 2012c. Atlas Copco Blasthole Drills: PV-271/PV-275 Pit Viper Series.
Baker Hughes Mining Tools. 1988. Blasthole Bit Handbook. 37 pp.
Baker Hughes Mining Tools. 1989. Blasthole Bit Handbook. 37 pp.
Baker Hughes Mining Tools. 1991. Product Catalog. 17 pp.
Baker-Hughes INTEQ. 1995. Drilling Engineering Workbook – A Distributed Learning Course. Dec.
Baker-Hughes. 2004. Rock Bit Catalog.
Baker-Hughes. 2007. In-Depth. 100 years of service. Special edition. July, Volume 13, No. 2.

Bauer, A. 1967. Open pit drilling. Proceedings of the 28th Annual Mining Symposium and Meeting of Minnesota Section AIME Duluth, Minn. January. pp. 175–180.
Bauer, A. 1971. Open pit drilling and blasting. Journal of the South African Inst. of Min. and Met., 71(1): 115–121.
Bauer, A. & P.N. Calder. 1967. Open pit drilling – factors influencing drilling rates. In Proceedings of the 4th Canadian Symposium on Rock Mechanics, Ottawa, March. pp. 1–33.
Bauer, A. & W.A. Crosby. 1990. Drilling application. Section 6.1.2 Surface Mining, 2nd Edition. (B.A. Kennedy, editor) Society for Mining, Metallurgy and Exploration, Inc. Port City Press, Baltimore, pp. 524–539.
Bauer, A., Carr, N.H., Calder, P. & G.R. Harris. 1966. Drilling and blasting at the Smallwood Mine. Trans. Soc. Mining Engineers. Vol. 235. March. pp. 39–45.
Bauer, A., Carr, N.H., Calder, P.N. & G.R. Harris. 1965. Drilling and blasting at Smallwood Mine. In Proceedings of the 7th U.S. Symposium on Rock Mechanics, Penn State University, June. pp. 218–241.
Baur, G.W. 1956. Rotary bits in taconite and other hard rocks. Proceedings of the 6th Annual Drilling and Blasting Symposium. Univ. of Minn. Oct 11–13. pp. 26–31.
Bielstein, W.J. & G.E. Cannon. 1950. Factors affecting the rate of penetration of rock bits. Drilling and Production Practice, API.
Biggs, M.D. & J.B. Cheatham, Jr. 1969. Theoretical forces for prescribed motion of a roller bit. SPE Paper No. 2391. Preprints of the 4th Conference on Drilling and Rock Mechanics, Austin, Texas, Jan 14–15. pp. 97–108.
Bilgin, N. & S. Kahraman. 2003. Drillability prediction in rotary blasthole drilling. 1st Int. Mining Congress and Exhibition in Turkey, Istanbul. pp. 177–182.
Bodsberg, E. 1969. Rotasjonsboring ved titania. Ekamensarbete. Institut for Gruvedrift, NTH, Trondheim. In Norwegian.
Borquez, G.V. 1981. Estimating drilling and blasting costs – An analysis and prediction model. E&MJ. Jan. pp. 83–89.
Bradbury, P.N. 1986. Common sense approach to cutting drill and blasting costs. Proceedings, Twelfth Conference on Explosives and Blasting Technique. ISEE. pp. 103–106.
Brezovec, D. 1983. Expensive bits are cheaper to use. Coal Age. 88(3):69–72.
Brown, J. W. 1998. Maximizing rotary blast hole drills. Proceedings, 24th Annual Conference on Explosives and Blasting Technique. ISEE. pp. 461–465.
Capp, F.M. 1962. Factors in rotary drill evaluation. Mining Congress Journal. 26(12):20–23.
Carter, R.A. 1992. Blasthole drills: Better by design. Coal, 97(7):40–43.
Carter, R.A. 2011. Cost effective drilling comes at a price – but pays big dividends. E&MJ. Dec. pp. 52–61.
Carter, R.A. 2011. Remote possibilities. E&MJ. Mar. pp. 42–48.
Casteel, K. 2008. Atlas Copco's new Texas snake: Smaller-scale suits larger market. E/MJ. 209(10):40–45. December.
Caterpillar. 2012. Rotary Blasthole Drills – Product Line.
Cheatham, J.B. & P.F. Gnirk.1967. Chapter 17. Mechanics of rock failure associated with drilling at depth. Eighth U.S. Symposium on Rock Mechanics. pp. 4l0–439.
Cheatham, J.B., Jr. 1958. An analytical study of rock penetration by a single bit tooth. In the Proceedings of the 8th Annual Drilling and Blasting Symposium, Univ. of Minn. 25 pp.
Chironis, N.P. 1981. Wider choice of large drills offered. Coal Age. 86(1):95–108.
Chitwood, B. & N.E. Norman. 1977. Blasthole drilling economics: A look at the costs behind the costs. E&MJ. pp. 83–89. Jan.
Chitwood, B.W. 1970. Rotary blast hole bits for drilling taconite. Proceedings of the 31st Annual Mining Symposium, Univ. of Minnesota. Duluth, Minn. pp. 191–194.
Clark, G.B. 1982a. Principles of Rock Drilling and Bit Wear, Part 1. Quarterly of the Colorado School of Mines, vol 77, No 1, Jan. 118pp.
Clark, G.B. 1982b. Principles of Rock Drilling and Bit Wear, Part 2. Quarterly of the Colorado School of Mines, vol 77, No 2, April, 42pp.
Clark, G.E. 1987. Principles of Rock Fragmentation. Wiley Interscience. New York. 610 pp.
Coffman, K.W. & J. Connors. 1974. Rolling cutter bit development and application in the mining industry. In the symposium proceedings Materials for the Mining Industry, Vail, Colorado. July 30–31. pp. 8–1 to 8–14.

Compressed Air and Gas Institute. 1965. Rotary Drilling Information. 5 pp.
Cotterill, S.R. & J.D. Nelmark. 1984. Inclined-hole drilling with large blasthole drills can sometimes be used. Mining Engineering. 36(9):1295–1298.
Crosson, C.C., M.J.H. Tonking & G. Paterson. 1976. The development of drilling and blasting practice at Palabora Mining Company, Limited. JSAIMM. 76(2):325–336.
Cunningham, R.A. 1978. An empirical approach for relating drilling parameters. Journal of Petroleum Technology. 30(7):987–991.
Cunningham, R.A.& W.C. Goins, Jr. 1960. Laboratory studies on the effect of rotary speed on rock bit performance. Journal of Petroleum Technology. 12(7):91–96.
Daly, S. & D. Assmus. 1991. Blast optimization and stability enhancement through geotechnical mapping at Highland Valley Copper. 93rd CIMM Annual General Meeting, Vancouver, April.
Danell, R.E. 1989. Estimation of insitu material strength. Proceedings of the 30th U.S. Symposium on Rock Mechanics, (editor). Balkema, Rotterdam. pp. 571–578.
Danell, R.E. 1990. Drill monitoring for the estimation of in-situ material strength. Proceedings, Advances in Mining Equipment Performance Monitoring, McGill University, Montreal, March 21–23.
Del Greco, P. 1956. Churn drilling in taconite at Pilotac Mine. Proceedings of the 6th Annual Drilling and Blasting Symposium. Univ. of Minn. Oct 11–13. pp. 66–71.
Dollinger, G.L. 1977. Choosing cutters for the best boreability. Compressed Air Magazine. 82(9):15–19.
Drake, R. 1990. Bench Drilling Techniques and Equipment Selection Manual. Ingersoll-Rand, Rock Drill Division, Roanoke, VA.
Drake, R. 1991. Better blasting can cost less. Pit and Quarry. Jan. pp. 37–40.
Dresser Mining Equipment. Technical Report- Blasthole Drilling Economics.
Drilco Industrial. 1976. The Drilco Roller Stabilizer.
Drilco Industrial. 1978. Blast Hole Drilling Tools.
Drilco Industrial. 1990. Blast Hole Drilling Tools. 14 pp.
Drilco Industrial. Bit Buddy Stabilizer.
Drilltech Mission. Introduction to Blasthole Drilling Using Driltech Mission Products and Accessories.
Dunn, A.R. & P.J. Hoddinott. 1994. Mining annual review-surface mining. Mining Journal. 170:19–29.
Ergin, H., C. Kuzu, C. B N. Bilgin. 2000. Optimum bit selection and operation for the rotary blasthole drilling through horizontal drilling rig (HDR) – a case study at KBI Nurgul Copper Mine. Int. J. of Mining, Reclamation and Environment. 14(4):295–304.
Eronini, I.E., W.H. Somerton & D.M. Auslander. 1981. A dynamic model for rotary rock drilling. Trans. ASME. 104(June):108–120.
Estes, J.C. 1971. Selecting the proper rotary bit. Society of Petroleum Engineers Preprint 3717. AIME. pp. 158–166.
Evans, I. & S.A.F. Murrell. 1958. The forces required to penetrate a brittle material with a wedge-shaped tool. In Mechanical Properties of Non-Metallic Brittle Materials. Interscience, N.Y. pp. 432–449.
Fairhurst, C. & W.D. Lacabanne. 1956. Some principles and developments in hard rock drilling techniques. In: Proceedings, 6th Annual Drilling and Blasting Symposium, Univ. of Minnesota, Oct 11–13. pp. 15–25.
Fairhurst, C. & W.D. Lacabanne. 1957. Hardrock drilling techniques. Mine and Quarry Engineering. 23(4):157 and 23(5):194.
Fiscor, S. 1993. Maintaining a detailed bit program reduces costs. Coal. 98(7):58–59.
Fiscor, S. 2009. Fully automated blasthole drilling becomes a reality. E/MJ. 210(10):30–35. December.
Fiscor, S. 2009. Fully automated blasthole drilling becomes a reality. E&MJ. Dec. pp. 30–35.
Fiscor, S. 2011. New system manages main compressor on rotary drills. E/MJ. 212(4):30–34. May.
Galle, E.M.& H.B. Woods.1960. How to calculate bit weight and rotary speed for lowest cost drilling. Oil and Gas Journal 58(46):167–176.
Galle, P.M. & H.B. Woods. 1963. Best constant weight and rotary speed for rotary rock bits. Drilling and Production Practice, API, pp. 48–73.
Gauthier, G. & G. Baron. 1965. Classification of rocks previous to any drillability investigation. Revue IFP 18(3):343.
Gillette, H.P., R.T. Dana & A.P. Ackerman. 1927. Rock excavation. Section 5: Mining Engineers' Handbook, 2nd Edition (Robert Peele, editor). Wiley, New York. pp. 187–198.
Gilmore, P.M. 1966. Rotary drilling in taconite. Proceedings of the 26th Annual Mining Symposium, Duluth, Minnesota, Jan.

Gnirk, P.F. & J.B. Cheatham, Jr. 1969. A theoretical description of rotary drilling for idealized down-hole bit/rock conditions. Soc. of Pet. Eng. J. 9:443–450.

Grad, P.S. 2010. Running with robotics: Step by step, Rio Tinto's concept of highly autonomous mining is materializing in Western Australia's Pilbara iron ore district. www.womp-int.com/story/2010vol01/story026.htm.

Gstalder, S. & J. Raynal. 1966. Measurement of some mechanical properties of rocks and their relationship to rock drillability. J. Pet. Tech. 18(8):991–996.

Hagan, T.N. & I.W. Reid. 1983. Performance monitoring of production blasthole drills- a means of increasing blasting efficiency. Trans. Inst. Mi & Met. 92: A173–A179.

Hartman, H.L. 1959. Basic studies of percussion drilling. Trans SME TP4785A or Mining Engineering 11(1):68–75.

Hartman, H.L. 1990. Drilling principles. Section 6.1.1. Surface Mining, 2nd Edition (B.A. Kennedy, editor). Society for Mining, Metallurgy and Exploration, Inc. Port City Press, Baltimore. pp. 513–523.

Heinen, R.H. 1979. Drill Evaluation. Section 18 in Open Pit Mine Planning and Design (J.T. Crawford, III and W.A. Hustrulid, editors). AIME. New York. pp. 245–256.

Hendricks, C. J. Peck & M. Scoble. 1990. Integrated drill and shovel performance monitoring towards blast optimization. Fragblast '90. Proceedings of the 3rd Int Symp. on Rock Fragmentation by Blasting. Australian Inst. of Min & Met., Brisbane. August 26–31. pp. 9–20.

Hertz, H. 1882. Űber die Berűhrung Elastischer Kõrper (On the contact between elastic bodies). In Gesammelte Werke (Collected works). Vol. 1. Leipzig. 1895.

Hissem, W.D. 1986. Drilling and cost analysis using a personal computer and spread sheet. Proceedings, Twelfth Conference on Explosives and Blasting Technique. ISEE. pp. 284–293.

Hoover, J., P. Conrad, S. Parrow, S. Rosenthal, & J. Downey. 2012. Forecasting drill bit consumption in surface drilling and blasting operations. Paper 12-011. SME Annual Meeting. Feb 19–22. Seattle, WA.

Hopkins, M.L. & S.D. Duggan. 1993. Automated drilling of surface blast holes using computer control. Proceedings, 2nd Int. Conf. on Mine Mech. & Automation, (Almgren, Kumar & Vagenas, eds). pp. 153–160.

Hoseinie, S.H. H. Aghababaei & Y. Pourrahimian. 2008. Development of a new classification system for assessing of rock mass drillability index (RDi). Int. J. Rock Mech. & Min. Sci. 45(2008):1–10.

Hughes Drilling Equipment. 1985. Smoothdrill.

Hughes Rigway. 1976. Hughes Tool Company. 14(3):25 pp.

Hughes Tool Company. 1959. Rotary Drilling Bits.

Hughes Tool Company. 1972. Rotary Drilling Bits, 37 pp.

Hughes Tool Company. 1975. Blast Hole Bit Handbook.28 pp.

Hughes Tool Company. 1976. Tri-Cone Bit Handbook. 48 pp.

Hughes, H.R. 1909. Drill. U.S. Patents 930758 and 930759. August 10.

Hughes, H.R. 1916. A modern rotary drill. Trans. AIME. Volume 51. pp. 620–627. Containing the papers and discussion of the New York Meeting (Feb 1915).

Hughes-Christensen. 2012. Drill Bit Catalog.

Huntly, M. 1999. Benefits of drill improvement teams. Proceedings, 19th Annual Conference on Explosives and Blasting Technique. ISEE. pp. 29–34.

Hustrulid, W. 1977. Blasthole Underreamers for Surface and Insitu Mining. Final Report on Contract JO265042 Submitted to the USBM. 144 pp.

Hustrulid, W.A. 1968. Theoretical and experimental study of percussive drilling of rock. PhD Thesis, University of Minnesota. pp. 310.

Hustrulid, W.A. 1971. The percussive drilling of rock (Parts 1–3). JSAIMM., 71(12) July: 245–268.

Hustrulid, W.A. 1974. A review of rotary drilling principles. U.S. Bureau of Mines Contract Report No. 110242–062.

Hustrulid, W.A. & C. Fairhurst. 1972. A theoretical and experimental study of the percussive drilling of rock (parts I–IV). Int. J Rock Mech & Min Sci., 9:431–449.

Ingmarsson, K. 1998. Drilling productivity and total drilling cost. Proceedings, 24th Annual Conference on Explosives and Blasting Technique. ISEE. pp. 451–461.

Jackson, R.A. 1976. Rock bit technology – The effect on drilling costs. J. Canadian Pet. Tech. Jan–Mar. pp. 64–68.

Joy Global. 2012. P&H MinePro Services. Personal communication with Tony Muhovich.

Kahraman, S. 1999. Rotary and percussive drilling prediction using regression analysis. Int. J. Rock Mech. Min. Sci. 36:981–989.
Kahraman, S. 2003. Performance analysis of drilling machines using rock modulus ratio. JSAIMM. 103(8):515–522.
Kahraman, S., C. Balchi, S. Yazichi & N. Bilgin. 2000. Prediction of the penetration rate of rotary blasthole drills using a new drillability index. Int. J. Rock Mech. Min. Sci. 37:729–43.
Kahraman, S., Develi, K., & E. Yasar. 2004. Predicting the penetration rate of percussive blasthole drills using coarseness index and median particle size. CIM Magazine. October. pp. 97–100.
Kelessidis, V.C. & H. Ergin. 2011. Analysis and optimization of drilling operating parameters in coring and drilling operations. Proceedings, 22nd World Mining Congress and Expo 2011. Istanbul, Turkey, Sept 11–16. 10 pp.
Kennedy, A. 1994. Improved operator conditions. Mining Magazine. 170:212–218.
Langford, J. 2011. Development of a percussion assisted rotary drilling system for blasthole drilling. Paper 11-108. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.
Larsen-Basse, J. 1973. Wear of hardmetals in rock drilling: a survey of the literature. Powder Metallurgy, 16(31):1–32.
Laswell, G.W. & B.J. Laswell. 1978. A supermarket of rotary drills. Mining Engineering. 30(7):773–779.
Laswell, G.W. & B.J. Laswell. 1978. Blasthole drilling doesn't have to be bad. Mining Engineering. 30(8):1162–1166.
Laswell, G.W. & B.J. Laswell. 1978. The pro's and con's of rotary blasthole design. Mining Engineering. 30(6):625–627.
Lebel, J.R. Guy, 1984. An Investigation to Evaluate the Relationship Between Rock Quality Index (RQI) and Powder Factor for Surface Mining. M.Sc. Thesis. University of British Columbia, Vancouver, Canada.
Leighton, J.C. 1986. Correlating rotary drill performance to powder factors for improved blasting control. Proceedings of the First Int. Symp. on Geotechnical Stability in Surface Mining, Calgary, Canada. pp. 121–132.
Leighton, J.C., C.O. Brawner & D. Stewart. 1982. Development of a correlation between rotary drill performance and controlled blasting powder factors. CIM Bull. 75(844):67–73.
Li, T.M. 1974. Rotary drills with automated controls-new force in open-pit blasthole production. Coal Age Operating Handbook of Coal Surface Mining and Reclamation, vol 2. (N.P. Chironis, ed). McGraw Hill, Inc. New York pp. 212–219.
Liu, F. & K.K. Yin. 2001. Analysis and interpretation of monitored rotary blasthole drill data. Int. J. of Surface Mining, Reclamation and Environment. 15(3):177–203.
Lovejoy, C. 2012. Surface drill and blast: Rotary rigs evolve. Mining Magazine. April. pp. 57–72.
Lownds, C.M. & P.C. Seligmann. 1976. Discussion of the paper "The development of drilling and blasting practice at Palabora Mining Company Limited". JSAIMM, 76(2):336–340.
Ludwiczak, J. 1989. Size selection of blast hole drilling equipment – Bigger is not always better. The Journal of Explosives Engineering. 6(5):27.
Lundquist, R.G. 1968. Rock Drilling Characteristics of Hemispherical Insert Bits. M.S. Thesis, University of Minnesota.
Lusk, B. & P. Worsley. 2011. Chapter 7.3 Explosives and Blasting. In SME Mining Engineering Handbook 3rd Edition (P. Darling, editor). SME, pp. 443–459.
MacGregor, K. 1967. The Drilling of Rock. C.R. Books. 306 pp.
Martin, J.W., T.J. Martin, T.P. Bennett & K.M. Martin. 1982. Surface Mining Equipment (First Edition). Martin Associates. 453 pages.
Maurer, W.C. 1962. The 'perfect cleaning' theory of rotary drilling. J. Pet. Tech. 14(11):1270–1274.
Maurer, W.C. 1967. Chapter 15. The state of rock mechanics knowledge in drilling. Failure and Breakage of Rock (C. Fairhurst, editor). Proceedings of the Eighth U.S. Symposium on Rock Mechanics. Held at the University of Minnesota, Sept 15–17, 1966. AIME, New York. pp. 355–395.
Maurer, W.C. 1968. Novel Drilling Techniques. Pergamon Press, New York, pp. 114.
Maurer, W.C. 1970. Potential application of novel rock disintegration techniques in mining. Proceedings of the 31st Annual Mining Symposium, University of Minnesota, Duluth. pp. 181–190.
Maurer, W.C. 1980. Advanced Drilling Techniques. Petroleum Publishing, Tulsa. pp. 698.
McIntyre, R.L. 1985. Surface mine rotary drilling. Chapter 1. In Proceedings of the Australian Mineral Foundation Workshop – Drilling and Blasting in Surface Coal Mines. Brisbane, Qld, Australia. Oct 28–Nov 1.

Medlock, J.D. 1960. Laboratory testing of rotary rock bits. Proceedings of the 10th Drilling and Blasting Symposium, Colorado School of Mines, Golden, Colorado.

Meyer, R. 2009. Optimizing the rotary drill string. E/MJ. 210(10):36–37. December.

Meyer, R. 2009. Optimizing the rotary drill string. E&MJ. Dec. pp. 36–37.

Mikkelborg, J.K. 1969. Rotasjonsboring ved Rana Gruber. Ekamsarbetet. Institut för Gruvedrift, NTH, Trondheim. In Norwegian.

Mining Technologies International, Inc. 2012. MTI Blasthole Drilling Tools.

Mishnaevsky, L.L., Jr. 1995. Physical mechanisms of hard rock fragmentation under mechanical loading: A review. Int. J. Rock Mech Min. Sci. & Geomech, 32(8):763–766.

Mishnaevsky, L.L., Jr. 1996. A new approach to the design of drilling tools. Int. J. Rock Mech Min. Sci. & Geomech, 33(1):97–102.

Modular Mining Systems, Inc. 1990. Drill monitor system(DMS) Sales Brochure MMS-DMS-9012. Tucson, Arizona.

Mol, O., R. Danell & L. Leung. 1987. Studies of rock fragmentation by drilling and blasting in open cut mines. Proceedings 2nd Int. Symp. on Rock Fragmentation by Blasting, Keystone, Colorado. pp. 381–392.

Montgomery, R.S. 1968. The mechanism of percussion wear of tungsten-carbide composites. Wear. 12: 309–329.

Moore, P.L. 1958. Five factors that affect drilling rate. Oil and Gas J. 56(Oct 6).

Moore, R.N. 1982. Drilling for profit. World Mining. 35(2):40–41.

Morey, G.B. & K.J. Ashley. 1984. Open pit mine design. Northwest Mining Association Short Course, Spokane, WA. Nov.

Morrell, R.J. & H.F. Unger. 1973. Drilling Machines, Surface. Section 11.4. SME Mining Engineering handbook, Vol. 1 (A.B. Givens and I.A. Given, editors). pp. 11–24 through 11–60.

Morris, R.I. 1969. Rock drillability related to a roller cone bit. Preprint SPE 2389. Preprints of the Fourth Conference on Drilling and Rock Mechanics (AIME), Univ of Texas, Austin. Jan. pp. 79–86.

Nelmark, J. 1981. 26 Most-asked questions about Bucyrus-Erie drills. Bucyrus-Erie Bulletin No 3007. 2.5 MFIB 781.

Nelmark, J.D. 1980a. Large diameter blasthole drilling: past, present & future. Proceedings of the 53rd Annual Meeting of the Minnesota Section AIME and the 41st Annual Mining Symposium (compiled by P.J. Sanders), Duluth, Minnesota.

Nelmark, J.D. 1980b. Large diameter blast hole drills. Mining Congress Journal. 66(4):30–35.

Nelmark, J.D. 1983. Blasthole drill design and application. Drilling and Blasting in Open Pits and Quarries (W.A. Crosby, editor). Australian Mineral Foundation Workshop, Brisbane, Australia (Feb 21–25).

Nelson, D.K. 1961. Recent developments and trends in production blast-hole drilling on the Mesabi Range. Proceedings of the 10th Annual Drilling and Blasting Symposium. Quarterly of the Colorado School of Mines. 56(1):135–145.

Norman, P. & B.W. Chitwood. Blasthole Drilling Economics. Reed Tool Company Technical Bulletin.

Outmans, H.D. 1960. The effect of some drilling variables on the instantaneous rate of penetration. Trans., Soc. of Pet. Engrs. (AIME). 219:137–149.

P&H MinePro Services. 2012a. P&H 250XPC Electric Rotary Blasthole Drill.

P&H MinePro Services. 2012b. P&H 350XPC Electric Rotary Blasthole Drill.

Paone J. & W.E Bruce. 1963. Drillability Studies. Bureau of Mines Report of Investigation 6324. 32 pp.

Paone, J. & S. Tandanand. 1966. Inelastic deformation of rock under a hemispherical drill bit. Trans SME. 235 (June):113–123.

Paterson, L.B. 1999. Productive and cost effective drilling. Paper 99-078. SME Annual Meeting. Mar 1–3. Denver, CO.

Paul, B. & D.L. Sikarskie. 1965. A preliminary study of static penetration by a rigid wedge into a brittle material. Trans. Soc. of Mining Engineers of AIME. 232:372–383.

Paul, B. & M.D. Gangal. 1969. Why compressive loads on drill bits produce tensile splitting in rock. SPE Preprint 2392. Fourth Conference on Drilling and Rock Mechanics, Univ. of Texas, Austin, Texas. Jan 14–15.

Peck, J. & D. Pollitt. 1990. Lithological recognition based on drill performance data. Proceedings of the Advances in Mining Equipment Performance Monitoring Symposium. McGill University, Montreal. March 21–23.

Peck, J. & J. Vynne. 1993. Current and future status of drill monitoring technology. 95th Canadian Inst of Min. & Met. Annual General Meeting, Calgary, May.

Peck, J. & J. Vynne. 1994. Applications of drill monitoring systems in open pit mines. AIME-SME Annual Meeting, Minnesota Section, Duluth, Minnesota, Jan 18–20.
Peck, J. & J.F. Vynne. 1992. Current status and future trends of monitoring technology for rotary blasthole drills. Paper presented at American Mining Congress MINExpo International '92, Las Vegas, Nev. Oct 18–22. 47 pp.
Peck, J. & J.F. Vynne. 1993. Current status and future trends of monitoring technology for rotary blasthole drills in coal mines. Paper presented at the 95th Annual General Meeting of the CIM. May 9–12. Calgary, Alberta, Canada. 29 pp.
Peck, J. 1989. Performance Monitoring of Rotary Blasthole Drills. PhD thesis. Dept of Mining & Met. Eng., McGill University, Montreal, Canada.
Peck, J., D. Pollitt & M. Scoble. 1991. Characterization of bench geology using drill monitoring. Paper presented at CIM District 5 Meeting, Fort McMurray, Alberta, Canada. 26 pp.
Persson, A. 1993. Sandvik Rock Tools. Personal Communication, Sept.
Peterson, C.R. 1969. Rolling cutter forces. Preprints of the Fourth Conference on Drilling and Rock Mechanics, Univ of Texas, Austin, Jan 14–15. pp. 115–124.
Pollitt, M.D., J. Peck & M. Scoble. 1991. Lithological interpretation based on monitored drilling performance parameters. CIM Bulletin. July 1991. 84(951):25–29.
Ramström, M. 2005. Atlas Copco drill rigs for mine automation and communication – A totally new technology platform for advanced mining systems. CIM Magazine. June/July. pp. 313–316.
Reed Drilling Equipment. Drill stems, stabilizers, subs.
Reed Tool Company. Drilling and Reaming Handbook and Reference Guide.
Reed, R.L. 1972. A Monte Carlo approach to optimal drilling. Trans., Soc. of Pet. Eng. J., 12(5): 423–438.
Reichmuth, D.R., 1963. Correlation of Force-displacement data with physical properties of rock for percussive drilling systems. Rock Mechanics (Proceedings of the 5th U.S. Symposium on Rock Mechanics (C. Fairhurst, editor). Pergamon Press, New York. pp. 33–6l.
Rio Tinto. 2012. Mine of the future. www.mineofthefuture.com.au.
Rollow, A.G. 1962. Estimating drillability in the laboratory. Proceedings 5th U.S. Symposium on Rock Mechanics (C.Fairhurst, editor). University of Minnesota, May. Pergamon Press, New York. pp. 93–102.
Rose, C.K. & S. Utter. 1955. Cutting action of rotary bits in oil shale. USBM RI 5174. November.
Ross, N.A. & W. Hustrulid. 1972. Development of a tunnel boreability index. Bureau of Reclamation Report REC-ERC 72-7. Feb.
Rostami, J. & D.F. Hambley. 2011. Chapter 7.2 Blasthole Drilling. In SME Mining Engineering Handbook 3rd Edition (P. Darling, editor). SME, pp. 435–441.
Rowley, D.S., R.J. Howe & F.H. Deily. 1961. Laboratory drilling performance of the full scale rock bit. J Pet Tech. 13(1):71–81.
Sandvik Coromant. Roller Bits for Blast Hole Drilling.
Sandvik Mining and Construction. 2012a. RR320 Series Premium Air Bearing Bits.
Sandvik Mining and Construction. 2012b. RR440 Ultra Premium Rotary Bits.
Sandvik Mining and Construction. 2012c. RP400 Drill String Tools.
Sandvik Mining and Construction. 2012d. Personal communication with Joel Boman. June–July.
Scheck, D.E., L.H. Mitchell & G.W. Mack. 1982. Automated blast hole logging and design. U.S. Bureau of Mines, Grant No G 111 5006, Final Report.
Schiveley, G.P. 1995. Optimizing rotary drill performance. Proceedings, 21st Annual Conference on Explosives and Blasting Technique, ISEE, Nashville, TN. pp. 5–23.
Schiveley, G.P. 1997. Introduction to drilling vibration. Proceedings, 23rd Annual Conference on Explosives and Blasting Technique. ISEE. pp. 647–655.
Schiveley, G.P., Jr. 1987. Focus on rotary drilling rigs. Compressed Air Magazine. 92(6):30–38.
Schiveley, G.P., Jr. 1994. Personal communication.
Schiveley, G.P., Jr. 1994. Predicting rotary drill performance. Proceedings of the 20th Annual Conference on Explosives and Blasting Technique, (Jan 30–Feb 3, 1994) Austin, Texas. Int. Soc of Explosives Engineers (ISEE), pp. 181–193.
Schultz, D. & P.F. Ostwald. 1973. Percussive carbide applications. Soc. of Manufacturing Engineers. Dearborne, Michigan.
Schwarzkopf, P. & R. Kieffer. 1960. Cemented Carbides. McMillan Company, New York.

Scoble, M.J. J. Peck & D. Kennedy. 1989. Drill monitoring investigations in a western Canadian surface coal mine. Proc. of the 2nd Open Pit Conf. of the AusIMM, Morewell, pp. 51–61. April.
Scott, F.L. & L.E. Garfield. 1934. Three-cone bit. United States Patent Office. Patent 1,983,316.
Scott, F.L. 1924. Self cleaning roller drill. U.S. Patent 1,480,014. Jan 8.
Security. 1990. Raise Pilot Hole Drilling.
Security. 1992. Mining Bit Catalog.
Security. Super-Aire Blasthole Bit Catalog.
Sessoms, W. 1990. An introduction to the model 245 and 265 blasthole drill monitors. Proceedings of the Advances in Mining Equipment Performance Monitoring Symposium. McGill University, Montreal. March 21–23.
Sievers, H. 1950. Die bestimmung des bohr widerstandes von gesteinen. Gluckauf Hefte 37/38:776–784.
Simon, R. 1963. Energy balance in rock drilling. Soc. Pet Eng. J. 3(12):298–306.
Simon, R., D.E. Cooper & M.L. Stoneman. 1956. The fundamentals of rock drilling. Presented at the Spring meeting of Eastern District of API Div. of Prod., Columbus, Ohio, April 25–27.
Smith Mining. 1991. General Catalog. 28 pp.
Smith Tool. Mining and Industrial Drilling Products Catalog.
Smith-Gruner. 1986–1987. General Catalog.
Smith-Gruner. Blast Hole Bit Evaluation.
Somerton, W. H. 1959. A laboratory study of rock breakage by rotary drilling. Petroleum Trans (AIME). 216:92–97.
Somerton, W. H., F. Esfandiari & A. Singhal. 1969. Further studies of the relation of physical properties of rock to rock drillability. SPE Preprint 2390, Fourth Conference on Drilling and Rock Mechanics, Univ of Texas, Austin. Jan 14–l5. pp. 87–96.
Speer, J.W. 1958. A method for determining optimum drilling rates. Drilling and Production Practice, API. pp. 130–147.
Stacy, C. 2010. Automated drilling – Ups and downs of auto drilling. Mining Magazine. November. pp. 35–38.
Taggart, A.F. & R.S. Lewis 1927. Boring. Section 9: Mining Engineers' Handbook (2nd Edition). Wiley, New York. pp. 322–368.
Tamrock. 1988. Surface Drilling and Blasting (Jukka Naapuri, editor). Tamrock, Tampere, Finland. 479 pp.
Tandanand, S. & H.L. Hartman. 1961. Stress distribution beneath wedge-shaped drill bit loaded statically. Int .Symp. Min. Res. (Vol 2). Pergamon Press. pp. 799–831.
Teale, R. 1965. The concept of specific energy in rock drilling. Int J Rock Mech and Min. Sci. 2: 57–73.
Thomas, R.W. 1996. Radical improvements to blast hole drilling cost and productivity thru integrated teams. Proceedings, Twenty-second Annual Conference on Explosives and Blasting Technique. ISEE. Orlando, FL. pp. 314–319.
Thompson International, Inc. 1993. The Total Package.
Thompson, D. 1998. A new generation of drilling at the Minntac mine. Skillings Mining review. pp. 4–10.
Thompson, R.E. 1956. Rotary drilling of blast holes on the Mesabi Range. Proceedings of the 6th Annual Drilling and Blasting Symposium. Univ. of Minn. Oct 11–13. pp. 73–76.
Toll, G.L. 1977. Open pit rotary drilling of blastholes. Chapter 1 in the Workshop Proceedings "Drilling and Blasting Technology". Australian Mineral Foundation Inc. Adelaide 2–6 May. 43 pp.
Torrington Company. 1988. Engineering-load rating and life calculations. Service Catalog. pp. E50–E55.
Varel International, Inc. 2012a. Ridgeback Marathon Blast Hole Bits.
Varel International. Inc. 2012b. Personal communication with Chris Nicolson. July.
Vynne, J. 1990. How to improve the drilling and blasting efficiency in open pit mines. Proceedings, Advances in Mining Equipment Performance Monitoring. McGill University, Montreal, March 21–23.
Ward, T.A. & A.J. Bradley. 1978. Drilling practice and investigations into drilling techniques at the mine sites of Hamersley Iron Pty. Limited. Rock Breaking: Equipment and Techniques. Australasian Inst. of Min. and Met, Melbourne Branch, Nov.
Wardlaw, H.W.R. 1961. Simplified analysis aids in optimizing drilling factors for minimum cost. J. Pet. Tech. 13(5):475–482.
Wardlaw, H.W.R., 1969. Drilling Performance optimization and identification of overpressure formations. SPE Preprint 2388. Preprints of the Fourth Conference on Drilling and Rock Mechanics, Austin, Texas. Jan. pp. 67–78.

Warren, T.M. 1981. Drilling model for soft-formation bits. J. Pet. Tech. 33(6):963–970.
Warren, T.M. 1984. Factors affecting torque for a tricone bit. Jour. Pet. Tech. 36(9):1500–1508.
Warren, T.M., L.A. Sinor & D.C.K. Chen. 1998. Shock sub performance tests. 1998 IADC/SPE Drilling Conference. Dallas, TX Mar 3–6. pp. 267–278.
Weston, W.B. & T.A. Stoner. 1970. Automation of a blast hole drill. Preprint Number 70-AO-60. Paper presented at the AIME Annual Meeting at Denver, Cob. Feb 15–19. 15 pp.
Wetherelt, A. & K.P. Van der Wielen. 2011. Chapter 10.1 Introduction to Open Pit Mining. In SME Mining Engineering Handbook 3rd Edition (P. Darling, editor). SME, pp. 857–876.
Wijk, G. 1989. The stamp test for rock drillability classification. Int. J Rock Mech Min. Sci & Geomech Abst. 26(1):37–44.
Wijk, G. 1991. Rotary drilling prediction. Int J Rock Mech Min. Sci & Geomech Abstr, 28(1):35–42.
Williamson, T.N. 1968. Rotary drilling. Chapter 6.3 in Surface Mining(E.P. Pfleider, ed) AIME, New York, pp. 300–324.
Wilson, J.W. & Y. Ding. 1994. Financial impact of the cost and productivity improvement of an efficient rotary drilling operation. Seminar on Rotary Blasthole Drilling in the '90's. South African Inst. Min. & Met., Johannesburg.
Woof, M. 2005. Big blasthole rig makers change hands. E/MJ. 206(3):28–30. April.
Work, J.T. 1980. How blasthole shock absorbers reduce costs. Mining Engineering. 32(4):395–396.
Workman, L. & M. Szumanski. 1996. Which blasthole rig? Proceedings, Surface Mining 1996 (H.W. Glen, editor). SAIMM. Johannesburg.
Workman, L. & M. Szumanski. 1997. Which blasthole rig? World Mining Equipment. March. pp. 23–27. www.wme.com/wme/Wme0397/0397rigs.htm.
Young F.S. 1969. Computerized drilling control. Jour. Pet. Tech. 21(4):483–496.
Zaburunov, S.A. 1991. Production drilling technologies. E&MJ. 192(2):29–37.
Zink, C.R., Jr. 1997. Updating air practices for better open pit blasthole drilling. Proceedings, 23rd Annual Conference on Explosives and Blasting Technique. ISEE. pp. 617–635.
Zink, C.R., Jr. 1998. Bottom hole annulus pressure – A theoretical problem wireath real effects. Proceedings, 24th Annual Conference on Explosives and Blasting Technique. ISEE. pp. 377–391.

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REVIEW QUESTIONS AND EXERCISES

1. Summarize the tricone bit development including important dates.
2. What was the significance of the Spindletop oil field?
3. Compare the change going from the two-cone bit to the two-cone intermeshed bit to the change going from the two-cone intermeshed bit to the three-cone intermeshed bit.

4. What was the background of H.R. Hughes, Sr.? Who does it seem should be given the most credit for the development of the modern tricone bit? Collect information from the internet.
5. Summarize the breakage process occurring beneath an insert. How does this understanding help you think about pulldown, rotation rate and cuttings removal?
6. In selecting the appropriate bit to be used in a given formation it is of importance to understand a little about the insert metallurgy. Why? What are traditional mixtures of tungsten carbide and cobalt?
7. In the chapter, sealed bearing bits have not been discussed. How do these work? What are their advantages and disadvantages?
8. Examine the bit cutaways and locate the air-flow passages. Where is the flow-back valve located? What is its purpose?
9. Study the material on the IADC code. Examine the material from the different bit suppliers and see how their selections compare.
10. Not many years ago, steel-tooth bits had many mining customers. Today, the steel-tooth market is relatively small. What are the reasons for this change?
11. Summarize the main components of a rotary drilling rig.
12. Summarize the main steps in the drill selection process.
13. Percussion drills have not been included in the chapter. When should they be considered as opposed to rotary drills? What are their advantages/disadvantages?
14. Who are the major suppliers of rotary blasthole drills in the world today?
15. Typical specification sheets for a rotary blasthole drill have been included. Carefully review them and summarize some of the information you are most interested in.
16. One common "rule-of-thumb" is that the maximum pulldown a drill can apply is about ½ its weight. How well does this apply to the 250XPC drill?
17. What is the mast height on the 250XPC drill? What would be the maximum expected single pass hole length?
18. Based on the specifications, which might be a comparable drill to the P&H 250XPC in the Caterpillar/Bucyrus-Erie line of drills?
19. Based on the specifications, which might be a comparable drill to the P&H 250XPC in the Atlas Copco Pit Viper series of drills?
20. Summarize the components of a typical drill stem and drill stem stabilization system.
21. Why would one use a shock absorber sub?
22. Why would one use a stabilizer?
23. What is the practical effect of adding elements such as the shock sub and the stabilizer in the drill string?
24. What are practical bit roller cone bit diameters used in blasthole drilling? What are the associated drill stem diameters?
25. The "weight on the bit" consists of two components. One is the applied pulldown and the other is the weight of the drill string. When drilling a hole 50 ft in length, what might be the weight of the string?
26. Summarize the procedure used by Morris for predicting the penetration rate? Is it still relevant today?
27. Summarize the basis for the Bauer-Calder penetration rate formula. A nomograph developed by them is included. Check to determine if it is based on using the 250 or 300 constant.
28. Is the Bauer-Calder approach relevant today? Be careful!

29. Summarize the basis for the Smith-Gruner penetration rate – pulldown – rotation rate – strength curve. How might you produce this curve today? Is it worth doing? Why or why not?
30. Summarize the basis for the Workman-Szumanski modification to the Bauer-Calder equation.
31. Compare the Workman-Szumanski modification to the Smith-Gruner equations for the different strength ranges. What is the way forward?
32. Summarize the differences in the applied bit-weight recommendations from the early 1980's and now. What has changed?
33. Discuss ways for arriving at an appropriate rotary speed.
34. Bit life is obviously an important factor. Do you think there is an equivalent to the 80 hour bit life rule used in the 1980's?
35. From the list of technical tips for best bit performance presented by Atlas Copco, choose the 5 which are most important in your mind and try to remember them.
36. Why is effective cuttings removal an essential element in modern blasthole drilling? What is the standard air compressor capacity installed on the P&H 250XPC drill? What is the operating pressure? Try to find the specifications for a similar drill from the 1980's. What do you observe concerning air volumes and pressures?
37. Describe the flow of the compressed air. What happens in the bit region concerning the cooling of the bearings and the cuttings removal? Try to be as quantitative as possible.
38. Summarize the nozzle selection process.
39. Assume that the P&H 250XPC drill is used to drill a 9-7/8″ diameter hole. Assume that the minimum bailing velocity desired is 5000 ft/min. Select the nozzles and the drill stem. What is the final configuration?
40. For problem 39, assume that the minimum desired bailing velocity is 7000 ft/min. Select the drill stem and the nozzles.
41. Use the nomograph in Figure 10.39 to determine the air velocity for a 10-5/8″ diameter bit, a 8-5/8″ pipe size and 1400 cfm. Is this a good combination? Why or why not?
42. Use the cost calculation formula to evaluate the situation where by increasing the rotary speed one can increase the penetration rate by 30% but the bit life will decrease by 50%. Is this a good tradeoff?
43. At a large iron mine, 18 drill shifts are scheduled daily on a two shift basis to produce 140,000 tons of ore and waste per day.
    a. If the bench height is 35 ft, the rock has a specific gravity of 3.0, the pattern is 22 ft × 22 ft, the hole diameter is 9″, and the subdrilling is 6 ft, how many feet of hole are drilled per drill shift.
    b. It is proposed to switch from ANFO to an emulsion explosive. How might this impact the drilling?
44. The production at a mine is 28,000 tons of ore per day and 22,000 tons of waste. The mine works 7 days per week. Atlas Copco Pit Viper drills are used to drill 9″ diameter holes to a depth of 39 ft. The bench height is 33 ft. The burden is 20 ft and the spacing is 22 ft. The specific gravity of the rock is 2.56 and the stemming length is 14 ft. If the drill can drill 450 ft of hole per shift, how many drill shifts per day (assuming that drilling is also done 7 days/week) are required to meet production requirements? Which Pit Viper model might be chosen?
45. At an open pit mine it is planned to use blast holes 12-1/4″ in diameter. To do this, the mine is considering to buy one of the drills in the Sandvik rotary drill product line. Go

to their website and select an appropriate model. The rock has an unconfined compressive strength of 18,000 psi. Choose an appropriate Sandvik tricone insert bit for this application. Assume that the rock is of medium density

a. Recommend a pulldown weight and a rotation rate for the bit. What penetration rate would be expected?
b. Select the nozzles for the bit
c. Select the drill pipe
d. What is the expected bailing velocity?
e. Assume that you would like to increase the penetration rate by 20%. Would it be better to increase the weight on the bit or the rotation rate? Explain.

46. Repeat problem 45 using a Caterpillar/Bucyrus-Erie drill and Atlas Copco Secoroc tricone bits
47. Repeat problem 45 using a P&H drill and Varel International tricone bits.
48. At a limestone mine, the rotary drills used to drill the 6″ diameter holes rotate at 110 rpm. The pulldown capacity of the drilling machine is 35,000 lbs. The recommended loading of the bit is 4000 lbs/in. The hole length is 45 ft. There is no subdrilling due to the presence of a parting plane. The upper 20 ft of the formation is quite soft due to weathering (unconfined compressive strength = 10,000 psi). The lower part of the formation has a strength of 15,000 psi. The drill can drill 25 ft in a single pass but carries 4 pieces of drill steel and has automated rod handling. The time to add or subtract a piece of steel from the string is 1 minute. The rod hoisting speed is 100 ft/minute. The time between holes is 10 minutes (moving, setup, etc).
    a. How long does it take to drill one hole?
    b. Assume that the drill is scheduled 1-8 hr. shift/day, 5 days/week and 50 weeks/year. The availability during the scheduled time is 90%, the utilization is 80% and the operating efficiency is 85%. How many hours is the drill string actually rotating (making hole)/year?
    c. Assuming a rock density of 2.49 g/cm^3, how many tons will the drill block out/year? In other words, "What is the drill production/year expressed in tons?"
49. A rotary blasthole drill has been selected for a proposed mining operation. The following information applies:
    - rock compressive strength = 8000 psi
    - drill rotation rate = 90 rpm
    - bit type = insert type (medium rock)
    - bit cost: 12-1/4″ bit = $8000
15″ bit = $16000
    - bit life = 4000 ft for both bits
    - drill operating cost = $600/operating hour
    - rock specific gravity = 2.65
    - bench height = 40'
      a. Calculate the following for both hole diameters:
         – recommended maximum bit weight (lbs/in)
         – recommended maximum bit weight (lbs)
         – expected penetration rate (ft/hr) at maximum bit weight
         – hole length
         – blast pattern (assume S = 1.15 B)

- bit cost/hole ($)
- drill operating cost/hole ($)
- total drilling cost/hole ($)
- drilling cost/ton ($/ton)

b. Based on your answer to part (a), which hole size would you choose?

50. The rotary drills on a mining property use 9″ diameter medium formation tungsten carbide bits. The bench height is 35 ft. In ore, the blasthole pattern is 21 ft × 21 ft. In waste, the pattern is 24 ft × 24 ft. The explosive is ANFO. The ore in place density is 4300 lbs/yd^3. In waste, the density is 3800 lbs/yd^3. Assume that the Ash-based formulas have been used for designing the blasting patterns. Ore production at the mine is 18000 tons per day. The stripping ratio is 2.8:1. The mine works 6 days per week and 2 shifts per day. Drill productivity is 520 ft/shift in both ore and waste.
    a. How many drill shifts are required per week to meet the ore tonnage?
    b. How many drill shifts per week are required to meet the waste tonnage?
    c. How many drills should the property have? Explain.
51. A mine is using 9-7/8″ diameter medium formation tungsten carbide rotary bits. Production is 75,000 tons/day of waste and 40,000 tons/day of ore. The blast pattern is such that they achieve 30 tons/ft of drilling. The hole production per drill per shift is 900 ft. The rock density is 2.65 g/cm^3. Hole spacing is 1.15 B. The bench height is 30 ft. ANFO is used.
    a. If the rock has a compressive strength of 25,000 psi, the drill rotation rate is 85 rpm and the pulldown applied is that recommended for the bit, what would be the estimated penetration rate?
    b. What percentage of the 8-hr shift is actually spent with the bit penetrating the rock?
    c. How many feet of drill hole are required per day?
    d. How many drill shifts are required per day to maintain production?
    e. If the drill pipe has a diameter of 8-5/8″ and the air velocity needed to remove the cuttings is 6000 ft/min, what size compressor would be required?
52. Using the Smith-Gruner bit selection chart, what would be the:
    a. Weight per inch of bit diameter
    b. Rotation rate
    c. Pulldown recommended for a 12-1/4″ diameter bit in rock having a compressive strength of 35,000 psi?
53. Using the formula for penetration rate suggested by Bauer (1971), together with the values obtained in Problem 52, compare the estimated penetration rates.
54. Data for rotary drills from the 2011 edition of the Cost Reference Guide has been included in Chapter 2. Summarize the data presented for drills with 125,000 lbs of pulldown. Briefly describe the meaning of the different cost categories (how calculated, what's included).
55. Using the data in Chapter 2, summarize the hourly
    - ownership
    - overhaul
    - field repair and fuel expenses for a rotary drill capable of drilling a 12-1/4 inch diameter hole. What are the total operating costs/hour and the total hourly costs?
56. The drill in problem 55 is used to drill blast holes 12-1/4″ in diameter and 50 ft in length in one pass. The move time from hole to hole is 5 minutes, leveling requires

45 seconds, and collaring requires 30 seconds. The average penetration rate is 70 ft/hour. Hoist speed is 80 ft/min. If the drill is operated 6 hours per shift, how many holes could be drilled in a shift?

57. If the driller in problem 56 makes $24.00/hour (including fringes), the life of the bit is 6000 feet, and the life of the drill pipe is 50,000 feet, what would be the operating cost per foot of hole? If the hole spacing is 30 ft and the burden is 25 ft, the bench height is 40 ft and the specific gravity of the rock is 2.8, what would be the drilling cost (operating) per ton?
58. Tests indicate that the required bailing velocity for the material drilled in problem 56 is 5500 ft/min. The drill is equipped with an air compressors having a capacity of 2200 CFM. What sizes of drill pipe could be used with this bit? What size would you recommend? If the compressors provide air at 65 psi and the line losses are 10 psi, what are the appropriate nozzle sizes to be used with the bit?

CHAPTER 11

Shovel loading

11.1 INTRODUCTION

In medium to large-scale surface mines, the principal piece of production loading equipment is the shovel. Although several different varieties exist and are used, this short section will only deal with electric rope shovels. One such shovel is shown diagrammatically in Figure 11.1 with some of the major points labeled.

Although the figure is somewhat dated, in general it still applies. For P&H shovels, the shipper shaft and pinion crowd are still used in rock but a dipper handle replaces the dipper stick. Caterpillar shovels have a dipper stick but no shipper shaft.

The digging element of a power shovel is a dipper attached to a handle. The dipper lip, the principal cutting edge, is attached to the dipper front and is equipped with replaceable teeth. The handle is connected to the back side of the dipper. The dipper bottom is a hinged door and has a latch for dumping the contents (Sargent, 1990).

Shovel dippers are sized to match the unit material weights to be handled with ruggedness increasing with the severity of the expected digging conditions. The dipper represents a dead weight load that subtracts from the load that can be raised by the machine (payload) and hence its weight must be kept to a minimum. The angle between the handle axis and the bottom surface of the dipper with teeth is generally about 65°. The dipper is tapered to

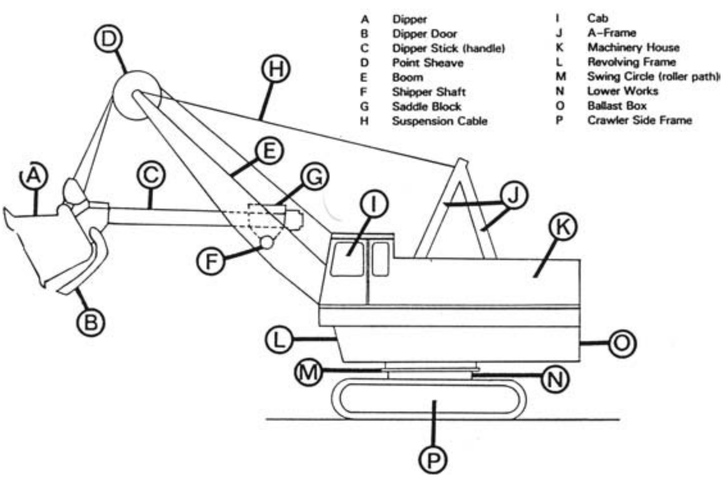

Figure 11.1. Shovel nomenclature (Martin, 1982).

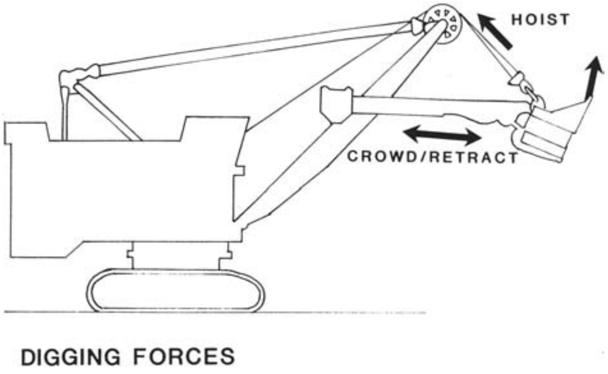

Figure 11.2. Shovel digging force (Martin, 1982).

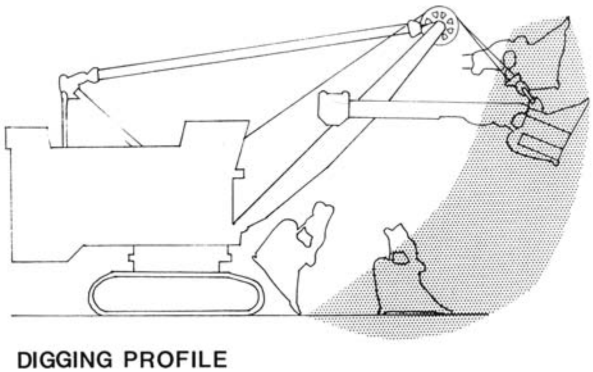

Figure 11.3. Digging profile. Martin (1982).

reduce jamming. The dipper latch is a simple sliding bar arrangement activated through a light cable driven by a small electric motor. Dippers for regular duty are roughly square in cross-section. A door hinged at the rear is unlatched to dump the load. The door opens under the force of gravity in an uncontrolled discharge. If there are large boulders in the dipper, it must be lowered close to the truck body or other discharge surface to avoid high impact loads. The door automatically re-latches as the dipper is lowered into the start of the dig position. Martin (1982).

In operation, the dipper is pulled through the bank by hoist cables. It is held against the bank by the crowd motion which extends or retracts the handle length wise. The handle is pivoted about a fixed point as it is positioned lengthwise by crowd motion. The dipper is pulled through the cut (Fig. 11.2).

The cutting path geometry is relatively restricted and is generally typified by a vertical circular arc. Martin (1982). A typical digging profile is shown in Figure 11.3.

When the dipper is full, the machine swings sideways and the dipper load is dumped through the door.

The entire machine is usually mounted on a system of crawlers. The crawlers are endless belts of links hinged or pinned together. They permit mobility and the positioning of the machine with respect to the bank. Steering may be accomplished by powering one belt and locking the opposite belt. The actions of

- hoist
- crowd

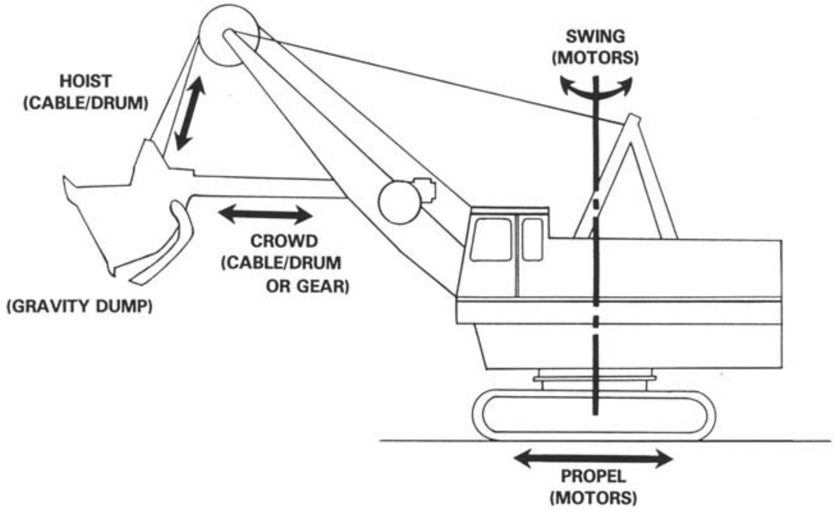

Figure 11.4. Shovel powered functions (Martin, 1982).

- swing
- dump
- propel

are shown diagrammatically in Figure 11.4.

Although various systems are available for powering shovels, this chapter will focus entirely on those powered electrically through a trailing cable. Although the dependence on the trailing cable somewhat limits mobility, trailing cable handling can be facilitated through the use of special cable handling trucks. For isolated sections of the pit, skid mounted diesel-electric power units can be employed. Power consumption is between 0.2 and 0.45 kwh per bank cubic yard of material moved. The advantages of electric cable power are considered to be

- efficient utilization of energy
- reliability of the system
- simple controlling equipment

Repairs and maintenance on the shovel can be facilitated by incorporating modular and solid state electrical components.

The pit power supplied to the shovel is three phase, 60 cycle. It may be 4160, 6900 (7200) or 14400 volts (Martin, 1982).

On the shovel there are currently three possible types of drive systems (Martin, 1982).

1. A main A.C. motor with power supplied through the trailing cable drives a separate D.C. generator for each function. The D.C. generators are connected to D.C. motors for hoist, crowd, propel and swing.
2. A transformer is employed to change high voltage A.C. from the trailing cable to low voltage A.C. The A.C. power is then converted to D.C. by solid state electronics. D.C. motor drives are used to power the various functions.
3. A transformer and rectifier are used to convert to D.C. Inverters then convert to controlled frequency A.C. with separate units for each function. Individual functions are powered by A.C. induction squirrel cage motors.

The *hoist* machinery consists of a motor driving a cable drum through a gear train.

There are two basic types of *crowd* drive systems (Martin, 1982).

1. Rope crowd
2. Rack and pinion crowd.

In the rope crowd system, wire ropes anchored at the drum, transmit power up the boom, over the sheaves mounted on the shipper shaft, then along the handle. The *crowd* cable passes over a deflection sheave on top. The *retract* cable is attached to an adjustable take-up screw at the bottom. Wire rope elasticity, combined with cushioned sheaves reduces shock loads. The dipper handle is a circular tube and is essentially free to rotate. Power is supplied by variable speed D.C. or A.C. motors. In the rack and pinion crowd, a motor drives a pinion which engages a rack on the lower side of the handles. The handle is made of either

- a single box-sectioned member
- twin box members designed to straddle the boom.

Power is supplied by variable D.C. or hydraulic motors. Various methods are used to cushion shock loads.

Swing machinery consists of a motor, gear reduction, and a large circular rack centered at the principal vertical swinging axis.

11.2 OPERATIONAL PRACTICES (Martin (1982))

There is a great deal of accumulated experience for using electric shovels in mining operations. Common practices can be briefly summarized as follows:

- Strongly consolidated materials should be drilled and blasted prior to excavating.
- The shovel should operate on a level, flat digging floor whenever possible.
- Digging downhill permits higher digging thrust forces because gravity adds to the machine's resistance to movement away from the digging face.
- Normally, the digging face should not be higher than the boom point sheave.
- The toe of the bank should be below the rear of the point sheave or shipper shaft.
- Crawlers should be perpendicular to the face to minimize possible damage from materials sliding down the face and to facilitate positioning maneuvers.
- Short frequent moves are desirable to keep the shovel close to the face maximizing the effectiveness of the crowd and hoist forces.
- Hoist and crowd motions should be coordinated for an efficient dipper path up through the bank, starting at the toe and filling in a single pass.
- Penetration of the face should be uniform with a depth sufficient to fill the dipper in 2 or 3 dipper lengths. The dipper should then be retracted clear of the face to minimize travel in the bank.
- To optimize motor life, prolonged stalling of the motors while digging is to be avoided.
- Excessive crowd will jack the boom and lead to reduced suspension cable life.
- Under difficult conditions, the top of the bank should be dug away to ease hoisting through the lower portion.
- Boulders or frost caps are removed by digging underneath and then lifting out with the dipper.
- If a hard toe is present, the machine should be moved as close to the bank as practical, to gain maximum benefit of crowd forces and improve tooth orientation for penetration.
- The face can be prepared (while waiting for trucks) by raking the bank with the dipper with the door open.

- To dislodge a boulder jammed in the dipper, the dipper should be passed through the bank with the door open, or the dipper teeth bumped on the floor.
- Oversized boulders must be cast clear of the operations or carefully set behind the machine. They should not be rolled over the teeth into the truck. (Swinging up to the truck is dangerous.)
- The dipper should not be swung until clear of the bank. Neither should it sweep the floor or face to push rocks aside. These actions produce very high torsional loads in the boom and can also result in bending the handle.
- The swing angle must be kept as low as possible to minimize cycle time.
- The machine is accelerated rapidly at the start of the swing and then plugged (reverse power) to decelerate. The action should be smooth to avoid spillage.
- The truck or hopper to be loaded should be positioned under the boom point sheave. The suspended dipper can be used as a target to spot trucks.
- Good practice requires loading over the rear of the truck to avoid any spillage near the truck driver. The empty/loaded dipper should not be swung over personnel or other equipment.
- The dipper should be lowered close to the truck body (or hopper) prior to dumping to minimize the impact from the discharge.
- It is desirable to avoid unnecessary crowding or retracting when positioning the dipper over the truck or hopper.
- If there is the potential for face slides, it may be desirable to load the trucks on the blind side (opposite from the operator station) so that the operator has a good view of the face during the entire operating cycle.
- The trailing cable should be clear of the truck maneuvering area.
- When practical, fines should be loaded into the truck or hopper first to provide a cushion for subsequent coarse materials.
- The typical digging cycle should take 25 to 35 seconds: dig 24%, swing loaded 32%, dumping and swinging empty 34%, and bucket positioning 10%.
- The operator should be in front, in the direction of travel, for all machine moves.
- During propel, drive sprockets should be in the rear so that any track belt slack is accumulated along the top.
- Gradual intermittent turns should be made to minimize material build-up on the side of the track (15 to 20 degree increments).
- Care must be exercised in negotiating steep downhill grades. The propel brakes and steering clutches should be checked, the dipper held close to the ground to serve as a possible emergency retarder, and the trailing cable kept on the uphill side of the machine.
- Track lubrication should be checked before and during long machine moves.
- No personnel should climb on board without prior notification of the shovel operator.
- To obtain the high production capabilities of the shovel during truck loading, the loading site must be carefully synchronized.

11.3 DIPPER CAPACITY

In the past, shovel capacity was normally expressed in terms of a nominal dipper capacity expressed in volume units. For example, 17 cubic yards (13 m^3). In connection with this number, it was understood, although sometimes not clearly written, that the density of the nominal material being handled was 3000 lbs/yd^3 (1.78 t/m^3). Using this rule, the actual "load" capacity of the shovel was 51000 lbs (23 mt). However, in this presentation there was also some vagueness concerning the meaning of "volume" although it was generally assumed that this referred to the "struck" capacity. By this it is meant the volume of water that the dipper would hold if brim full. When loading rock and soil as opposed to water, it is obviously possible to retain material lying above the brim and the *heaped capacity* will be greater than the struck capacity. The Society of Automotive Engineers (SAE) has developed standard methods for rating capacities thereby creating a consistency amongst the different manufacturers. Different materials stand at different slope angles. The convention used to express the slopes by the SAE is the horizontal distance (ΔX) divided by the vertical height ΔY). Hence, a 2:1 slope designation means that the slope rises one unit vertically for every 2 units moved horizontally. A 3:1 slope has a rise of one unit vertically for every 3 units moved horizontally. This way of expressing slopes, common in civil engineering and construction, is opposite to that

$$\text{slope} = \frac{\text{rise}}{\text{run}} = \frac{\Delta Y}{\Delta X} = \text{tangent (slope angle)}$$

generally used in mining.

Today, it is normal to express the capacity of the shovel in terms of a nominal payload rather than in nominal volume. For the P&H 4100XPC^{TM} shovel (P&H, 2012a), for example, the capacity is given as 120 short tons (st). Depending on the expected density of the material to be handled and the expected filling of the dipper, one arrives at the dipper size to be selected for the particular application. In their technical data for the 4100XPC shovel, the following typical ranges are provided:

- nominal dipper capacity:
  - SAE struck: 69–82 yd^3 (52.8–61.2 m^3)
  - SAE (heaped) 2:1: 74.4–88.4 yd^3 (58.3–67.6 m^3)
- Optimum truck size: 240–400 st (218–363 mt)

Beginning with the given payload of 120 st and assuming (1) a typical material density of 3400 lb/yd^3, and (2) the dipper typically filled to its struck capacity, one might choose a dipper of volume 71 yd^3 (or the closest "standard" size thereto). If, on the other hand, the expected density of the material was 3000 lbs/yd^3, then one might choose a dipper with a capacity of 80 yd^3 (assuming average filling to its struck capacity). As can be seen, one needs to have good information concerning the fragmentation characteristics of the material, the density of the shot material, the loading characteristics, etc in order to choose the proper loading shovel – dipper combination.

11.4 SOME TYPICAL SHOVEL DIMENSIONS, LAYOUTS AND SPECIFICATIONS

Figure 11.5 shows a diagrammatic representation in which a P&H 4100XPC shovel is loading a 400 st capacity truck. (P&H, 2012a). In this way, one can check in an easy way the match between the truck and shovel geometries.

Tables 11.1 and 11.2 summarize some of the important shovel dimensions for several models of shovel as extracted from data sheets provided by P&H (2012a,b,c,d).

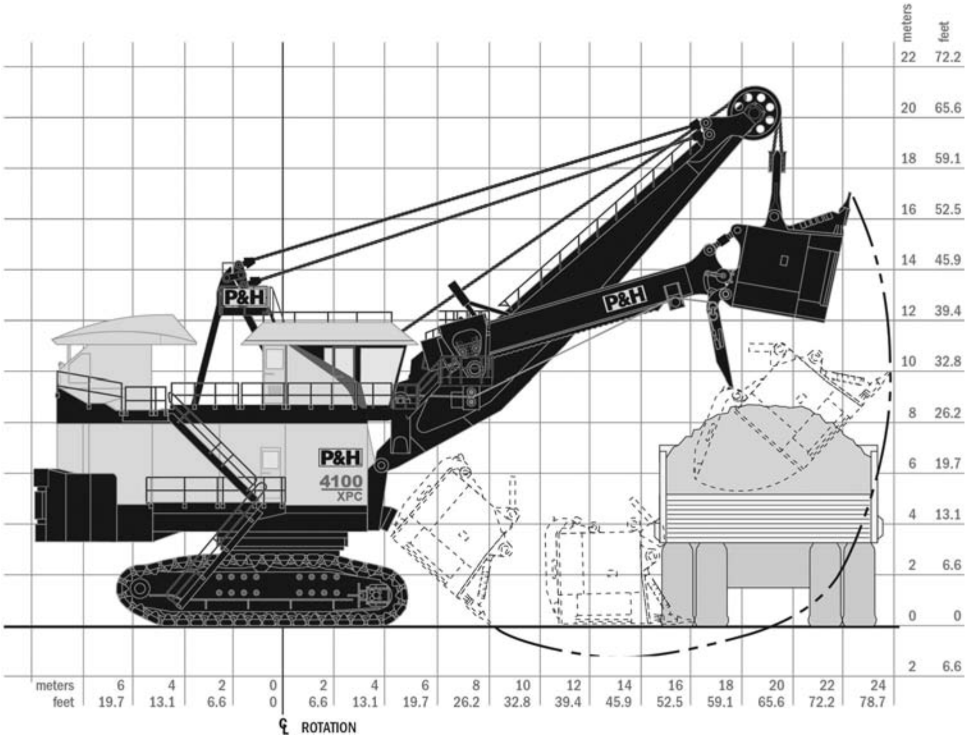

Figure 11.5. Diagrammatic representation of the P&H 4100XPC shovel. P&H (2012a).

Table 11.1. Some shovel dimensions (meters).

Make	Model	Height of cut	Radius of cut	Dumping height (door open)	Floor level radius	Tail swing radius	Operator eye level
P&H (Joy Global)	1900AL	13	17.8	8.2	11.6	7	7.4
	2300XPC	13.5	21.3	8.5	14.2	10.1	7.9
	2800XPC	16.6	24.2	9.1	16.4	9.9	9.6
	4100XPC	16.8	23.9	9.5	16	9.8	10.1

Table 11.2. Some shovel dimensions (Imperial units).

Make	Model	Height of cut	Radius of cut	Dumping height (door open)	Floor level radius	Tail swing radius	Operator eye level
P&H (Joy Global)	1900AL	42 ft 6 in	58 ft 6 in	27 ft 0 in	38 ft 0 in	23 ft 0 in	24 ft 3 in
	2300XPC	44 ft 3 in	70 ft 0 in	28 ft 0 in	46 ft 7 in	33 ft 0 in	26 ft 0 in
	2800XPC	54 ft 6 in	79 ft 3 in	30 ft 0 in	53 ft 9 in	32 ft 6 in	31 ft 4 in
	4100XPC	55 ft 2 in	78 ft 8 in	31 ft 0 in	52 ft 6 in	32 ft 3 in	33 ft 1 in

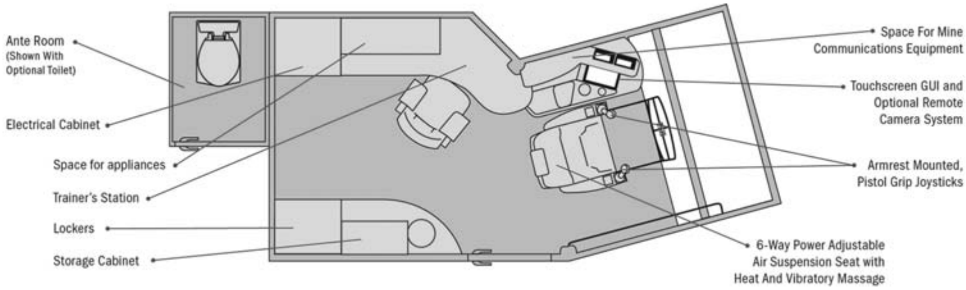

Figure 11.6. Loading control center for the P&H 4100XPC shovel. P&H (2012a).

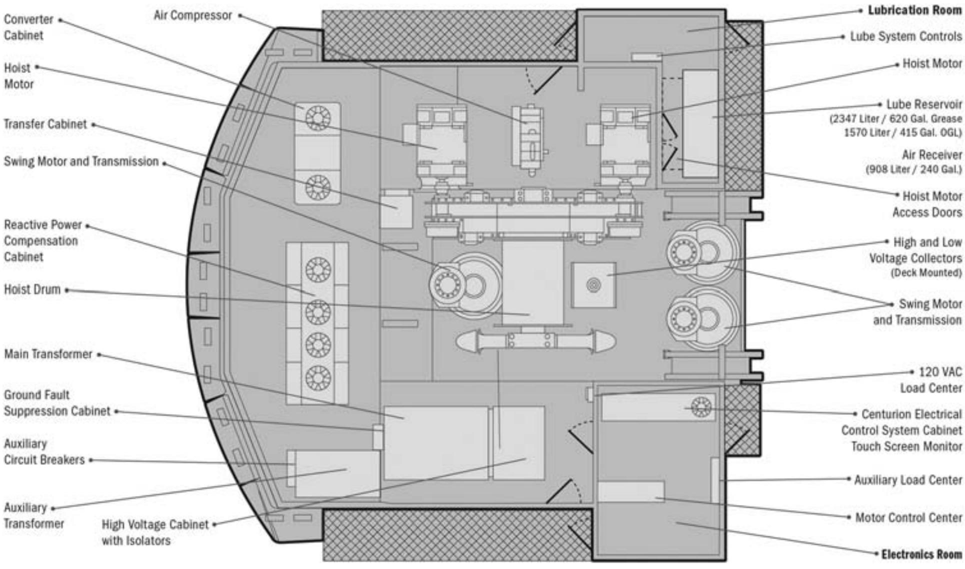

Figure 11.7. Machinery deck plan for the P&H 4100XPC shovel. P&H (2012a).

Figures 11.6 through 11.7 are plan views through the 4100XPC shovel at two elevations showing the nature and locations of the different components and facilities.

11.5 BALLAST/COUNTERBALANCE REQUIREMENTS

In order to balance the moment about the centerline of shovel rotation produced by the dipper and the material in the dipper, a large ballast/counterweight box extends across the rear of the revolving frame. This is filled with scrap iron or another high density material during shovel erection. Table 11.3 presents ballast values for certain models of shovel. The ballast is typically provided by the customer.

Table 11.3. Ballast requirements for different standard shovel models (P&H (2012a,b,c,d)).

Make	Model	Nominal payload given by mfg		Nominal dipper capacity		Ballast mt	Ballast st
		Metric tons (mt)	Short tons (st)	m^3	yd^3		
P&H	1900AL	18	20	10.7	14	54.4	60
	2300XPC	45.4	50	25.2	33	136.1	150
	2800XPC	59	65	33.6	44	230	253.5
	4100XPC	109	120	61	80	204.1	225

The "nominal dipper capacity" values provided in the Table correspond to applying a density of 1.78 mt/m^3 or 3000 lbs/yd^3 to the given nominal payload.

11.6 SHOVEL PRODUCTION PER CYCLE

Prior to describing the shovel production calculation procedure, it is important to provide some definitions:

"Bank" versus "loose" properties.

In the mine, the material which is to be loaded into the dipper must generally first be loosened from its "in-place (in situ)" or "bank" position. In mining, this is often accomplished by blasting. Assuming that the material occupies a volume of 1 yd^3 in place (denoted as 1 bank cubic yard, or 1 bcy), that same amount of material would be expected to occupy a larger volume in the loose condition. The weight of one cubic yard of loose material (one lcy), due to the presence of void (air) spaces between the pieces, would be less than the weight of one bcy. Values of the bank and loose weight densities for various materials are given in Table 11.4.

Swell:

The "swell" is defined as the ratio of the bank to loose weight densities.

$$\text{Swell} = \frac{\text{bank weight/unit volume}}{\text{loose weight/unit volume}} \tag{11.1}$$

Percent swell:

The "percent swell" is defined as

$$\text{Percent swell} = 100(\text{swell} - 1) \tag{11.2}$$

Swell factor:

The "swell factor" is the inverse of the swell

$$\text{Swell factor} = \frac{1}{\text{swell}} = \frac{\text{loose weight/unit volume}}{\text{bank weight/unit volume}} \tag{11.3}$$

The "swell factor" may also be expressed as

$$\text{Swell factor} = \frac{100}{(100 + \text{percent swell})} \tag{11.4}$$

Table 11.4. Some properties for various materials which are important for loading machine evaluation. Atkinson (1971), Atkinson (1992).

Rock	Specific gravity (bank)	Weight density		Swell	Percent swell (5%)	Swell factor	Fill ability	Disability
		lbs/bcy	lbs/lcy					
Asbestos ore	1.9	3200	2885	1.4	40	0.71	0.85	*M*
Basalt	2.95	50000	3125	1.6	60	0.62	0.80	*H*
Bauxite	1.9	3200	2370	1.35	35	0.74	0.90	*M*
Chalk	1.85	3100	2384	1.3	30	0.76	0.90	*M*
Clay (dry)	1.4	2400	1920	1.25	25	0.80	0.85	*M*
Clay (light)	1.65	2800	2153	1.3	30	0.76	0.85	*M*
Clay (heavy)	2.1	3600	2666	1.35	35	0.74	0.80	*M-H*
Clay and gravel (dry)	1.5	2500	1923	1.3	30	0.76	0.85	*M*
Clay and gravel (wet)	1.8	3000	2222	1.35	35	0.74	0.80	*M-H*
Coal (anthracite)	1.6	2700	2000	1.35	35	0.74	0.9	*M*
Coal (bituminous)	1.25	2100	1555	1.35	35	0.74	0.9	*M*
Coal (lignite)	1.0	1700	1307	1.3	30	0.76	0.9	*M*
Copper ore (low-grade)	2.55	4300	2866	1.5	50	0.66	0.85	*M-H*
Copper ores (high-grade)	3.2	5400	3375	1.6	60	0.62	0.80	*H*
Earth (dry)	1.65	2800	2153	1.3	30	0.76	0.95	*E*
Earth (wet)	2.0	3400	2615	1.3	30	0.76	0.9	*M*
Granite	2.41	4000	2580	1.55	55	0.64	0.8	*H*
Gravel (dry)	1.8	3000	2400	1.25	25	0.80	1.0	*E*
Gravel (wet)	2.1	3600	2880	1.25	25	0.80	1.0	*E*
Gypsum	2.8	4700	3133	1.5	50	0.66	0.85	*M-H*
Limenite	3.2	5400	3857	1.4	40	0.71	0.85	*M*
Iron ore 40% Fe	2.65	4500	3214	1.4	40	0.71	0.8	*M-H*
Iron ore – 40% Fe	2.95	5000	3448	1.45	45	0.68	0.8	*M-H*
Iron ore – 60% Fe	3.85	6500	4193	1.55	55	0.64	0.75	*H*
Iron ore (taconite)	4.75	8000	4848	1.65	65	0.60	0.75	*H*
Limestone (hard)	2.6	4400	2750	1.6	60	0.62	0.80	*M-H*
Limestone (soft)	2.2	3700	2466	1.5	50	0.66	0.85	*M-H*
Manganese ore	3.1	5200	3586	1.45	45	0.68	0.85	*M-H*
Phosphate rock	2.0	3400	2266	1.5	50	0.66	0.55	*M-H*
Sand (dry)	2.0	2900	2521	1.15	15	0.86	1.00	*E*
Sand (wet)	2.0	3400	2956	1.15	15	0.86	1.00	*E*
Sand and Gravel (dry)	1.95	3800	3304	1.15	15	0.86	1.00	*E*
Sandstone (pourous)	2.5	4200	2625	1.6	60	0.62	0.8	*M*
Sandstone (cemented)	2.65	4500	2812	1.6	60	0.62	0.8	*M-H*
Shales	2.35	4000	2758	1.45	45	0.68	0.8	*M-H*

An example of this type of calculation as applied to a copper ore is:

$$\text{weight density (bank)} = 3800\,\text{lbs/bcy}$$

$$\text{weight density (loose)} = 2800\,\text{lbs/lcy}$$

$$\text{Swell} = \frac{3800}{2800} = 1.36$$

$$\text{Percent swell} = 100(1.36 - 1) = 36\%$$

$$\text{Swell factor} = \frac{1}{1.36} = 0.74$$

Fillability or fill-factor

The "fillability" or the "fill-factor" is another factor which must be taken into account when estimating production per cycle. This refers to the ratio of the loose volume of rock contained in the dipper to the rated dipper capacity.

$$\text{fillability} = \frac{\text{loose volume of rock (yd}^3)}{\text{rated dipper capacity (yd}^3)} \tag{11.5}$$

Some care must be taken to assure that the "fillability" or the "fill-factors" and the rated dipper capacities go together. For shovels, the rated dipper capacity (nominal dipper capacity) is the struck capacity. The "fillability" factors given in Table 11.4 apply to this volume. Rated capacities for front end loaders, for example, generally refer to a 2:1 heaped capacity and the "fillability" factors based on this volume would be different from those given in Table 11.4. In estimating fill factors, the following rules of thumb apply:

- a typical value is about 0.85 but higher values are sometimes quoted.
- fill factors will be higher where the material is easy to dig and lower where digging is hard
- where material breaks finely, the fill factors will be higher
- fill factors are higher for large equipment and lower for small equipment

There are various ways of expressing the production per cycle. In mining, it is most common to discuss *tons*. The basic equation may be written

$$T_c = B_c \times F_f \times W_B \times S_w \tag{11.6}$$

where
T_c = tons/cycle
B_c = nominal dipper capacity (yd^3)
F_f = fillability factor
W_B = weight density (lbs/bcy)
S_w = swell factor

In civil construction, one is often more interested in the *bank cubic yards* moved per cycle. In this case the expression may be written as,

$$Q_c = B_c \times F_f \times S_w \tag{11.7}$$

where

Q_c = bank cubic yards/cycle

To obtain the production over a given time period, for example one shift, one needs to simply multiply the production per cycle by the cycles in the time period.

It is clear that the tons per cycle calculated using equation (11.6) must be compared to the rated capacity for the shovel. If it is higher or lower than the rated, adjustments can be made in

- dipper size
- blasting patterns (to improve fragmentation).

11.7 CYCLE TIME

The average time (T_A) required for a mining shovel to complete one cycle of
- loading the dipper
- hoisting and swinging
- dumping
- returning and lowering

is presented in Table 11.5 for different dipper volumes and digging conditions. These values are applicable for a swing angle of 90° and an optimum digging depth.

As can be seen, Table 11.5 was developed for the shovels available at the time the initial paper by Atkinson (1971) was written. For the modern loading shovels of today being used in medium digging conditions, the cycle time is in the range of 33–35 seconds, basically independent of dipper size (P&H, 2012e).

For swing angles less than 90°, the time required would be less. For angles greater than 90°, it would be more. The modified cycle times can be estimated by application of a swing correction factor (C_S) values for which are given in Table 11.6.

With the inclusion of the angle of swing correction factor, the estimated cycle time becomes

$$T_c = \frac{T_A}{C_s} \tag{11.8}$$

where

C_s = swing correction factor from Table 11.6.

Table 11.5. Loading shovel cycle times (sec) assuming a 90° swing angle. Atkinson (1971).

B_c		Digging conditions			
yd^3	m^3	*E*	*M*	*M-H*	*H*
4	3	18	23	28	32
5	4	20	25	29	33
6	5	21	26	30	34
7	5.5	21	26	30	34
8	6	22	27	31	35
10	8	23	28	32	36
12	9	24	29	32	37
15	11.5	26	30	33	38
20	15	27	32	35	40
25	19	29	34	37	42

E. Easy digging. loose. free-running material, e.g. sand, small gravel. *M. Medium digging*, partially consolidated materials. e.g. clayey gravel, packed earth, clay, anthracite, tect. *M-H Medium-hard digging*, e.g. well blasted limestones, heavy wet clay, weaker ores, gravel with large boulders, etc. *H. Hard digging*-materials that require heavy blasting and tough plastic clays. e.g. granite, strong limestone, taconite strong ores, etc.

Table 11.6. Swing correction factor. Atkinson (1992).

Angle of swing, degrees	45	60	75	90	120	150	180
Swing factor	1.2	1.1	1.05	1.00	0.91	0.84	0.77

A second modifying factor to the cycle time concerns the vertical distance (depth of cut or cutting height) through which the dipper must move in order to be filled to capacity.

Assuming an average bank penetration P, the vertical travel (L) required to fill a dipper of width W_d and volume B_c is

$$L = \frac{B_c}{W_d P} \tag{11.9}$$

where
L = vertical travel
B_c = dipper volume
W_d = dipper width
P = bank penetration

The depth of penetration depends upon the characteristics of the material being dug. In Table 11.5, the four categories; easy (E), medium (M), medium-hard (M-H), and hard (H) digging are used to classify the material. Since the filling portion (t_f) of the cycle time is related to L, the penetration would be high for easy digging materials and hence the needed cutting height would be low. On the other hand, for hard digging material the penetration would be low and hence the needed cutting height would be high.

For a given dipper (B_c and W_d fixed) operating in a given material (P fixed), there corresponds an optimum depth of cut (L_o) which produces the greatest shovel output (tons/hour). For each pass the dipper comes up with a full load without undue crowding. The practical application of this is that the bench height (H) should be greater than or equal to L_o to achieve the highest output.

$$H \geq L_o \tag{11.10}$$

SAE specification J732C specifies digging depth as

> *"the vertical distance in mm (inches) from the ground line to the bottom of the bucket cutting edge at the lowest position with the bucket cutting edge with the bucket cutting edge horizontal."*

A general rule of thumb that can be used is

> "The optimum depth of cut is equal to the vertical distance from the dipper stick pivot shaft to the ground level."

This position is shown in Figure 11.8.

The relationship between optimum cutting height and dipper capacity shown in Figure 11.9 can be expressed by either

$$L_o = 22 + \frac{B_c}{3} \tag{11.11a}$$

or

$$L_o = 10.5\sqrt[3]{B_c} \tag{11.11b}$$

A second rule of thumb has been given by Martin (1982). They indicate that

> *"for good cutting, the penetration of the face should be uniform with a depth sufficient to fill the dipper in 2 or 3 dipper lengths".*

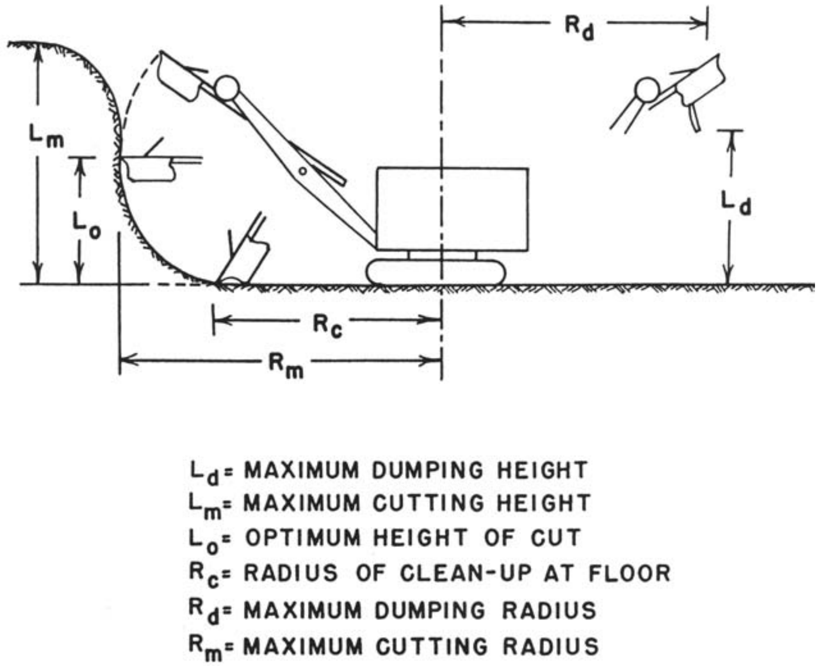

Figure 11.8. General working specifications of power shovels. (Ash, 1968).

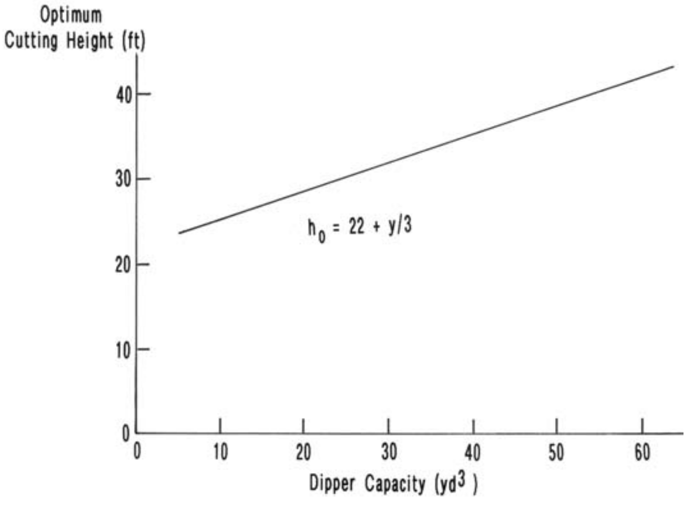

Figure 11.9. Optimum cutting height as a function of dipper capacity.

Since the vertical distance from the dipper stick pivot to the ground is of the order of 2.5 times the dipper length, the rules are fully consistent.

When digging in bench heights less than L_o the cycle times must be multiplied by a bench correction factor (C_{BH}). Appropriate values are given in Table 11.7.

Table 11.7. Correction factor for shovel cycle time as a function of relative digging depth. Atkinson (1971).

Optimum digging depth, percent	40	60	80	100
Cycle time correction factor	1.25	1.10	1.02	1.0

It should be pointed out that the optimum cutting depth has nothing to do with the maximum cutting height of the shovel. This latter quantity is important in matching the shovel and the maximum bench height. In this regard the rule of thumb is

> *"the maximum bench height should be less than or equal to the maximum cutting height of the shovel."*

An alternate expression of this rule of thumb is that the digging face should not be higher than the boom point sheave. This alternate form yields slightly higher values than that using the maximum cutting height (Fig. 11.8).

The general formula for the expected cycle time can be written as

$$T_c = \frac{T_A \times C_{BH}}{C_s} \tag{11.12}$$

where
T_c = cycle time
T_A = average cycle time (90°) for an optimum or greater digging depth
C_s = swing correction factor
C_{BH} = bench height correction factor

11.8 CYCLES PER SHIFT

Now that the time to perform one shovel cycle has been determined, it is possible to calculate that which is really desired, the cycles per shift. With this information and the load per cycle one can calculate the production per shovel shift. It will be assumed that the length of the shift (T) is

$$T = 3600\ H \tag{11.13}$$

where
T = shift length (sec)
H = shift length (hrs)

The theoretical number of cycles which could be performed is

$$C_{Theoretical} = \frac{3600H}{T_c} \tag{11.14}$$

In fact, the length of time that the shovel is actually cycling, is much less than 3600H.

When examining the shift production from a shovel it is important to distinguish between cycling time and clock (shift) time. Cycling time is that portion of the shift when the shovel is rotating between bank and truck. Clock time is the total time that an operator or crew is

Table 11.8. Propel factor. Atkinson (1971).

Strip mines	0.75
Multi-bench mines	0.85
Sand and gravel pits	0.90
High-face quarries	0.95

assigned to the machine. For an eight hour shift the value would be 8 hours or 480 minutes. The cycling time would be less than the clock time due to

- Mechanical availability (less than 100% machine availability during the scheduled shift).
- Job operational factors (less than 100% utilization of the available time).
- Propel time (time required to maneuver the shovel to the next loading position)

Included under the general heading of job operational factors are such things as:

- travel to and from the working place
- lunch time
- lubrication and fueling time
- waiting time
- time lost due to weather conditions, spent in safety meetings, etc.

The time required to propel the shovel during maneuvering is sometimes included as one of the job operational factors. Here it will be kept separate. For this discussion the mechanical availability factor (M_A) will be defined as

$$M_A = \frac{\text{clock time} - \text{down time}}{\text{clock time}} \tag{11.15}$$

The available time (T_A) then becomes

$$T_A = \text{clock time} - \text{down time} \tag{11.16}$$

The utilization factor (U_A) is defined as

$$U_A = \frac{\text{cycling time}}{\text{available time}} \tag{11.17}$$

The job efficiency factor (E) is the product of the mechanical availability and utilization factors

$$E = M_A \times U_A \tag{11.18}$$

As indicated, a portion of the total time is spent moving the shovel into position (propelling). Propel factors (P) are given in Table 11.8 for various types of operations.

Table 11.9 is a portion of a shovel shift summary report from a large open pit copper mine for which the clock time is 480 minutes.

The mechanical availability, utilization and job efficiency factors for each of these shovels for this shift is given in Table 11.10.

Table 11.9. Shovel shift report.

A Shift 15-Mar-82 at 15:50:15						
Shovel	Operator name	Idle (min)	Down (min)	Delay (min)	T&L (min)	Spare (min)
9	MARQUEZ, R.M.	88.	0.	32.	45.	0.
10	SMITH, L.W.	165.	0.	0.	50.	11.
11	MARTINEZ, F.M.	33.	0.	6.	50.	11.
12	FLORES, O.T.	70.	71.	5.	50.	15.
13	WOOD, R.G.	31.	208.	0.	30.	16.

Table 11.10. Mechanical availability, utilization and job efficiency from Table 11.9.

Shovel	Mechanical availability	Utilization	Job efficiency
9	1.00	0.66	0.66
10	1.00	0.53	0.53
11	1.00	0.79	0.79
12	0.85	0.66	0.56
13	0.57	0.72	0.41
Average	0.88	0.67	0.69

A total of 1415 minutes was spent cycling out of the possible 2400 minutes clocktime by the 5 shovels. The average factors were:

$$\text{Mechanical availability} = 0.88$$

$$\text{Utilization} = 0.67$$

$$\text{Job Efficiency} = 0.59$$

For an actual operation it is important to examine such figures to see where improvements might be made. In this case the utilization is quite low due in large part to time spent waiting for trucks. Shovel productivity improvements could be achieved by the assignment of more trucks.

The average number of cycling minutes (M) to be expected during a shovel shift can be calculated using

$$M = \text{clock time (minutes)} \times M_A \times U_A \times P \tag{11.19}$$

or

$$M = \text{clock time (minutes)} \times E \times P \tag{11.20}$$

For the case examined in Tables 11.8 and 11.9 (assuming P = 1 since it is not reported) one would find for a single shovel that

$$M = 480 \times 0.88 \times 0.67 = 283 \text{ minutes}$$

or

$$M = 480 \times 0.59 = 283 \text{ minutes}$$

Table 11.11. Time distribution for average delays per hour expressed in minutes and % for a shovel operation. Church (1981).

Time distribution Delays to shovel operation	min	%
Minor delays of less than 15 min duration:		
Insufficient rock haulers	4.2	7.0
Spotting of rock haulers at shovel	1.0	1.7
Shovel moves at face of cut	1.4	2.3
Shovel repairs and maintenance	1.4	2.3
Shovel handling of outsize rocks and stumps	0.7	1.2
Checking grade	0.3	0.5
Shovel operator's and other delays	0.8	1.3
Subtotals	9.8	16.3
Major delays of more than 15 min duration		
Moving to next cut and opening up cut	2.4	4.0
Shovel repairs and maintenance	3.0	5.0
Subtotals	5.4	9.0
All minor and major delays, totals	15.2	25.3

For the five shovels operating during the shift, the total is 1415 minutes or very close to the actual value as would, of course, be expected.

From an estimating viewpoint there are some guidelines from the literature regarding values of M_A, U_A and E.

Crawford (1979) states that electric shovels can normally sustain 75 to 80% availability and 80 to 90% utilization of availability for a substantial portion of their nominal 20-year life. The corresponding values would be

$$M_A = 0.75 \text{ to } 0.80$$

$$U_A = 0.80 \text{ to } 0.90$$

$$E = 0.60 \text{ to } 0.72$$

As a rule of thumb, Crawford (1979) indicates

> *"Allowing for minimum delays, an attainable target shovel productivity approaches 1185 t/dipper m^3 (1000 st/dipper yd^3) per 8 hour shift. The rule does not apply to coal or other light weight materials."*

Steidle (1977) in his rules of thumb for mining shovels suggests

$$M_A = 0.70 \text{ to } 0.80 \text{ (diesel)}$$

$$M_A = 0.90 \text{ (electric)}$$

$$U_A = 0.83$$

Church (1981) has presented the delay data (Table 11.11) in the operating cycle for shovel earth-rock production.

As can be seen the average hourly delays (clock time) have been split into (a) minor delays of less than 15 min. duration and (b) major delays of more than 15 min. duration. Repair as well as operating delays are lumped together.

11.9 SHOVEL PRODUCTIVITY EXAMPLE

To practice the concepts, calculate the production (bank volume per hour) from a shovel for which the following conditions apply:

- Dipper capacity (volume) = 15 yd^3
- Swing angle = 120° (time for a 90° swing is 30 sec.)
- Mechanical availability = 95%
- Working time = 50 min/hr.
- Optimum digging depth = 100% of optimum
- Swell = 30%
- Fill factor (fillability) = 90%
- Multi-bench mine
- Ore S.G. = 2.7

The calculation process will now be demonstrated:

Cycles/hour:

$$C = \text{theoretical cycles/hr} = \frac{60\,\text{min/hr}}{0.5\,\text{min/cy}} = 120\,\text{cy/hr}$$

$$C_{BH} = 1.0$$

$$C_S = \text{swing correction } (120° \text{ rather than } 90°),\ \text{therefore } C_S = 1.10$$

Thus, the corrected cycles = 120/1.1 = 109.09 cy/hr.

$$A = 0.95 \text{ Availability}$$

$$O = 50/60 = 0.83 \text{ working time}$$

Thus, corrected cycles = 109.09 × 0.95 × 0.83 = 86

$$P = \text{propel} = 0.85$$

Thus, actual cycles = 0.85 × 86 = 73.12 cy/hr.

Material/Cycle

$$Q_c = B_c \times F_f \times S_w$$

$$B_c = 15\,\text{yd}^3 = \text{dipper capacity (yd}^3) - \text{“loose”}$$

$$F_f = \text{fill factor (expressed as a ratio)} = 0.90$$

Thus, material/cycle (loose) = 15 × 0.90 = 13.50 yd^3/cy

But the material has broken and swelled.

$$\%\ \text{Swell} = 30\%$$

$$S_w = \frac{100}{100 + \%\text{Swell}} = \frac{100}{100 + 30} = 0.769$$

Thus,

$$\text{Material/cycle (bank} - \text{solid)} = 13.50/1.30 = 12.38\,\text{yd}^3$$

Bank Volume/Hr

$$\begin{aligned}\text{Cycles/hr} \times \text{bank material/cycle} &= 73.12 \times 12.38\\ &= \underline{759 \text{ bank yd}^3/\text{hr}}\end{aligned}$$

Tons/hr

$$\text{Tons/yd}^3 = \frac{2.7 \times 62.4 \times 27}{2000} = \frac{4949}{2000} = 2.27\,\text{tons/yd}^3$$

$$\text{Tons/hr} = 759 \times 2.27 = \underline{1723\,\text{tons/hr}}.$$

Tons/Shift = 8 hrs/shift × 1723 tons/hr = <u>13784 tons/shift</u> for 15 yd^3 shovel.
Rule of Thumb = 1000 tons/yd^3 dipper capacity/8-hour shift.

In this case,

$$13784/15 = 919\,\text{tons/yd}^3 \text{ dipper capacity/shift}$$

which is similar to the rule-of-thumb value.

11.10 DESIGN GUIDANCE FROM REGULATIONS

For a safe and efficient shovel-truck loading operation it is important that the geometries involved are well matched. Some design guidance is provided by regulation some of which are paraphrased below:

- Standards for the safe control of pit walls, including the overall slope of the pit wall, shall be established and followed by the operator. Such standards shall be consistent with prudent engineering design, the nature of the ground and the kind of material and mineral mined, and the ensuring of safe working conditions according to the degree of slope. Mining methods shall be selected which will ensure wall and bank stability, including benching as necessary to obtain a safe overall slope.
- To ensure safe operation, the width and height of benches shall be governed by type of equipment to be used and the operation to be performed.
- Safe means for scaling pit banks shall be provided. Hazardous banks shall be scaled before other work is performed in the hazardous bank area.
- Men shall not work near or under dangerous banks. Overhanging banks shall be taken down immediately and other unsafe ground conditions shall be corrected promptly, or the areas shall be barricaded and posted.
- Berms or guards shall be provided on the outer bank of elevated roadways.
- Dippers, buckets, loading booms, or heavy suspended loads shall not be swung over the cabs of haulage vehicles until the drivers are out of the cabs and in safe locations, unless the trucks are designed specifically to protect the drivers from falling material.
- Haulage equipment shall be loaded in a manner to minimize spillage during haulage.

The requirement for providing a safe means for scaling pit banks impacts the choice of equipment and the bench heights. Although some special pieces of equipment and techniques have and are being used to scale the pit walls, it is most common to do this with the primary excavator. For a loading shovel, this means that the bench height should be less than or equal to the maximum cutting height (h_m) of the shovel.

REFERENCES AND BIBLIOGRAPHY

Aksoy, C.O. 2005. Minimizing the economic unit cost of dragline-truck-shovel operations at Yatagan-Eskihisar open pit mine, Turkey. CIM Magazine. June/July. pp. 331–334.

Aksoy, C.O., Kose, H., Malli, T., Ozfirat, K., Gonen, A., & S. Gurgen. 2005. Truck-shovel optimization accounting for cost, safety, and operational practice. CIM Magazine. November. pp. 402–406.

Anonymous. 2007. Equipment – Buckets: Bucket basics. Mining Magazine. February. pp. 43–48.

Anonymous. 2007. Management in action – Excavator selection: Electric versus hydraulic. Mining Magazine. April. pp. 42–51.

Anonymous. 2009. Equipment – Rope shovels: Shovel shake-ups. Mining Magazine. April. pp. 27–29.

Anonymous. 2010. Equipment – Rope shovels: Super shovels. Mining Magazine. April. pp. 40–44.

Anonymous. 2010. News – Caterpillar to buy Bucyrus. Mining Magazine. December. p. 2.

Anonymous. 2011. Surface loaders – fully loaded. Mining Magazine. April. pp. 38–45.

Ash, R.L. 1968. Chapter 7.3 The design of blasting rounds. In Surface Mining (E.P. Pfleider, editor). AIME, NY. pp. 373–397.

Atkinson, T. 1971. Selection of open pit excavating and loading equipment. Institution of Min. & Met. Trans. Section A. London. July. pp. A101–A129.

Atkinson, T. 1992. Chapter 13.3 Selection and sizing of excavating equipment. In SME Mining Engineering Handbook, 2nd Edition, Vol. 2 (H.L. Hartman, editor). SME. Baltimore, Maryland. pp. 1311–1333.

Awuah-Offei, K., Osei, B., & H. Askari-Nasab. 2012. Improving truck-shovel energy efficiency through discrete event modeling. 2 Paper 12-069. SME Annual Meeting. Feb 19–22. Seattle, WA.

Bastock, M., & P. Knights. 2009. Applying modular transporters to relocate large hydraulic excavators. Iron Ore 2009 Conference. Perth, WA. July 27–29. pp. 217–222.

Berkhimer, E.N. 2011. Chapter 10.4 Selection and sizing of excavating, loading, and hauling equipment. In SME Mining Engineering Handbook, 3rd Edition (P. Darling, editor). pp. 931–939.

Bogunovic, D., & V. Kecojevic. 2011. Impact of bucket fill factor on dragline production rate and energy consumption. Mining Engineering. 63(8): 48–53. August.

Carter, R.A. 2005. Mining with precision: The latest generation of global positioning system (GPS) and geographic information system (GIS) technologies offers unprecedented accuracy for earthmoving and mapping. E/MJ. 206(4): 42–46. May.

Carter, R.A. 2006. Giving new life to old shovels. E/MJ. 207(7): 40–42. September.

Carter, R.A. 2007. Digital diggers: When it comes to digging and loading efficiency, equipment size may not always be the supreme factor. E/MJ. 208(1): 34–38. January/February.

Carter, R.A. 2008. GET smart: Ground engaging tools (GET) are designed to make the ultimate sacrifice – offering their service lives to promote better penetration and protect vulnerable bucket areas. E/MJ. 209(2): 58–60. March.

Carter, R.A. 2008. Hydraulic excavator overhauls – when and what to plan for. E/MJ. 209(3): 50–52. April.

Carter, R.A. 2010. Tracking new developments in dozer design. E/MJ. 211(2): 30–32. March.

Carter, R.A. 2010. New wheel loaders roll ahead to higher reliability, improved productivity. E/MJ. 211(6): 64–67. July/August.

Chanda, E.K., & R.J. Hardy. 2011. Selection criteria for open pit production equipment – Payload distributions and the '10/10/20' policy. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 351–364.

Church, H.K. 1981. Excavation Handbook. McGraw-Hill Book Company, New York. pp. 12-21 to 12-32. l Institution of Min. & Met. Trans. Section A. London. July. pp. A101–A129.

Crawford, J.T., III. 1979. Section 19. Shovel and haulage truck evaluation. Open Pit Mine Planning and Design (J.T. Crawford, III and W.A. Hustrulid, editors). AIME. pp. 258–270.

Dessureault, S., Yildirim, M., & M. Baker. 2007. Data mining mine data: truck-shovel fleet management systems. CIM Magazine. June/July. pp. 134–142.

Ercelebi, S.G., & A. Bascetin. 2009. Optimization of shovel-truck system for surface mining. JSAIMM. 109(7): 433–439.

Fiscor, S. 2005. Optimized dippers fine-tune shovel performance. E/MJ. 206(7): 76–79. September.

Fiscor, S. 2007. Productivity considerations for shovels and excavators. E/MJ. 208(7): 38–42. September.

Fiscor, S. 2009. Smarter shovels dig and load more efficiently. E/MJ. 210(3): 44–47. April.

Fiscor, S. 2010. Shovel OEMs make major design improvements. E/MJ. 211(7): 76–81. September.

Frimpong, S., & O. Brown. 2011. Optimization of shovel dipper – formation interactions in excavation. Paper 11-100. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Frimpong, S., & Y. Hu. 2005. Dynamic hydraulic shovel simulator for improved machine performance. CIM Magazine. January/February. pp. 155–159.

Frimpong, S., & Y. Li. 2006. Real-time monitoring of cable shovel stress for improved machine health. Paper 06-039. SME Annual Meeting. Mar 26–29. St. Louis, MO.

Frimpong, S., & Y. Li. 2007. Cable shovel health and longevity in formation excavation. Mining Engineering. 59(12): 50–56. December.

Frimpong, S., Hu, Y., & J. Szymanski. 2003. Intelligent cable shovel excavation in surface mining. Paper 03-37. SME Annual Meeting. Feb 24–26. Cincinnati, OH.

Hansen, J. 2001. Electric rope shovel monitoring. Paper 01-104. SME Annual Meeting. Feb 26–28. Denver, CO.

Harcus, M. 2011. LeTourneau update – born in the USA. Mining Magazine. January/February. pp. 9–10.

Haut, H., & M. Dietrich. 2011. A multibody simulation solution to analyse attachment kinematics of hydraulic excavators and other linkages. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 679–687.

Humphrey, J.D., & J. D. Wagner. 2011. Chapter 10.3 Mechanical extraction, loading, and hauling. In SME Mining Engineering Handbook, 3rd Edition (P. Darling, editor). pp. 903–929.

Joseph, T.G., & N. Shi. 2011. Scale field tests and visual modeling to evaluate a curved front dipper concept. CIM Journal. 2(3): 125–134.

Joseph, T.G., & N. Shi. 2011. Scaling analysis via geometric and performance data for shovel dipper design. CIM Journal. 2(4): 171–177.

Joseph, T.G., & N. Shi. 2011. Simulation of cable shovels and dippers in mining applications. CIM Journal. 2(3): 135–142.

Joseph, T.G., & N. Shi. 2012. A revised dipper-ground equilibrium derivation for shovels operating in oil, sand, and soft ground. CIM Journal. 3(1): 47–53.

Kirmanli, C. & S.G. Ercelebi. 2009. An expert system for hydraulic excavator and truck selection in surface mining. JSAIMM. 109(3): 727–738.

Komatsu. 2009. Komatsu Specifications and Application Handbook. Edition 30.

Li, Y., & S. Chang. 2009. The modeling and simulation of Bucyrus shovel, Part 1: Spacial kinematics and virtual prototype modeling. Paper 09-122. SME Annual Meeting. Feb 22–25. Denver, CO.

Li, Y., Liu, W., Shi, X., & S. Frimpong. 2009. Cable shovel dynamic modeling for determining component cycle stress. Paper 09-121. SME Annual Meeting. Feb 22–25. Denver, CO.

Lucio, J.C., Senra, C.T., & A. Souza. 2009. Paving the future – A case study replacing truck-and-shovels by shovel-and-conveyor continuous mining at Carajas open pit mines. Iron Ore 2009 Conference. Perth, WA. July 27–29. pp. 269–276.

Martin, J.W., T.J. Martin, T.P. Bennett, & K.M. Martin. 1982. Surface Mining Equipment. Martin Consultants, Golden, CO. 455 pp.

Mkhatshwa, S.V. 2009. Optimization of the loading and hauling fleet at Mamatwan open pit mine. 109(4): 223–232.

Moore, P. 2006. Equipment – Surface loaders: The essence of excavators. Mining Magazine. August. pp. 15–21.

Moore, P. 2008. Equipment – Large wheel loaders: Big wheels keep on turning. Mining Magazine. June. pp. 26–31.

Moore, P. 2010. Equipment access – Access all areas. Mining Magazine. December. pp. 30–33.

Norrena, K., & C.V. Deutsch. 2001. Automatic determination of dig limits subject to geostatistical, economic and equipment constraints. Paper 01-41. SME Annual Meeting. Feb 26–28. Denver, CO.

Oraee, B., Lashgari, A., & A. Sayadi. 2011. Estimation of capital and operation costs of backhoe loaders. Paper 11-040. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

P&H. 2012a. P&H 4100XPC Digital AC Drive Electric Mining Shovel.

P&H. 2012b. P&H 2800XPC Digital AC Drive Electric Mining Shovel.

P&H. 2012c. P&H 2300XPC Digital DC Drive Electric Mining Shovel.

P&H. 2012d. P&H 1900AL Electric Mining Shovel.

P&H. 2012e. Personal communication. June–July.

Porter, D. 2006. Equipment – Surface load-haul: Riding on a super cycle. Mining Magazine. April. pp. 11–17.

Porter, D. 2006. Opinion – Equipment trends: Payload weighing: In search of the Grail. Mining Magazine. August. pp. 22–23.
Porter, D. 2008. Disciplined digging – P&H C-series. Mining Magazine. February. pp. 26–30.
Porter, D. 2010. P&H Prevail – Advances in remote health monitoring. Mining Magazine. November. pp. 16–19.
Rathmann, B., & U. Heuer. 2007. Refit of an electric shovel or dragline – a cost saving alternative between frequent repairs and the purchase of a new machine. Paper 07-026. SME Annual Meeting. Feb 25–28. Denver, CO.
Raza, M., & S. Frimpong. 2011. Cable-shovel dipper mechanics. Paper 11-096. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.
Rogers, B. 2007. Equipment – Large wheel loaders: Wheeled versatility adds value. Mining Magazine. April. pp. 52–55.
Roos, C., Conrad, P. & S. Rosenthal. 2009. Estimating excavator teeth consumption rates. Paper 09-021. SME Annual Meeting. Feb 22–25. Denver, CO.
Roos, C.J., Conrad, P.W., & S.D. Rosenthal. 2010. Estimating excavator teeth consumption rates. Mining Engineering. 62(8): 45–51. August.
Saavedra, P.N., & C.J. Molina Vicuña. 2007. Online monitoring for mining shovels focused on vibration analysis. CIM Magazine. May. pp. 101–110.
Sargent, F.R. 1990. Section 6.4 Loading. pp. 620–657. Surface Mining, 2nd Edition (B.A. Kennedy, editor). SME.
Segarra, P., Sanchidrian, J.A., Montoro, J.J., & L.M. López. 2007. Mucking efficiency in open pit blasting. CIM Magazine. September/October. pp. 246–256.
Shi, N., & T.G. Joseph. 2006. A new Canadian shovel dipper design for improved performance. CIM Magazine. March/April. pp. 99–104.
Singh, S.P., & R. Narendrula. 2006. Productivity indicators for loading equipment. CIM Magazine. May. pp. 125–132.
Steidle, E.J. 1977. Evaluation of draglines and shovels in mining. Min. Cong. J. April. pp. 30–33.
Turnbull, D., & A Cooper. 2009. In-pit crushing and conveying (IPCC) – A tried and tested alternative to trucks. The AusIMM 2009 New Leaders Conference. Brisbane QLD. April 29–30. pp. 59–66.
Turnbull, D., & A. Cooper. 2010. In-pit crushing and conveying (IPCC) – A tried and tested alternative to trucks. Part 1. AusIMM Journal. Issue 5. Oct.
Turnbull, D., & A. Cooper. 2010. In-pit crushing and conveying (IPCC) – A tried and tested alternative to trucks. Part 2. AusIMM Journal. Issue 6. Dec.
Woof, M. 2005. A boom market for excavators. E/MJ. 206(5): 50–52. June.
Yavuz, M. 2008. An application of the TOPSIS method to solve hydraulic excavator selection problem in mining. Paper 08-042. SME Annual Meeting. Feb 24–27. SLC, UT.

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REVIEW QUESTIONS AND EXERCISES

1. The chapter is focused entirely on "rope" shovels. Hydraulic shovels and wheel loaders play an important part in modern mining operations. The reader is encouraged to construct a table showing the primary operating regime for rope shovels, hydraulic shovels and wheel loaders.

2. Given figure 11.1 with the various parts numbered, be able to supply the labels.
3. Describe the digging action of a modern shovel. Try to include as much of the "physics" in your discussion as possible.
4. Examine the digging profile in Figure 11.3. How does this relate to the final bench face angles observed in open pit mines? Is it easy to achieve vertical faces? Why or why not?
5. In Figure 11.3, it is seen that the maximum digging height is about equal to the height of the sheave wheel. If the shovel is used to scale the bench faces and make them safe, what design rule would emerge?
6. Assume that the geomechanics department has specified certain bench face angles in their pit slope designs. How easy is it to construct them in practice using typical shovel practices?
7. With regard to question 6, review Figure 11.4. Assume that via a computer I can control the hoist, crowd and propel all at the same time. Would I now be able to construct the bench angles as specified by geomechanics? Why or why not? Make a drawing in support of your thinking.
8. Martin (1982) has presented a rather long list of "operational practices." Review the list and identify five which you deem to be the most important.
9. Turn your attention to Figure 11.1. Note the position of the dipper and the ballast box. Construct a simple free-body diagram representing the loaded shovel. If you were the shovel designer, would you specify the dipper capacity by volume or by weight? Why?
10. In continuing the thinking process of problem 9, what modifications can you make when loading materials of different densities? Think about loading coal versus loading iron ore.
11. What is meant by struck capacity? What is meant by heaped capacity? What controls how full the dipper is actually filled?
12. In the past, it was quite common to quote dipper capacity in terms of volume with the assumptions that the material had a nominal weight of 3000 lb/yd^3 and the struck capacity applied. Review the specification sheets for the shovels of various manufacturers and see how well this applies.
13. Some of the dimensions for the P&H 4100XPC electric rope shovel are provided in Figure 11.5 and Table 11.1. If the bench height should be matched to the maximum cutting height, what would that be? What would it be if the sheave height is used instead?
14. For the P&H 4100XPC shovel, what is the dumping height? How is this dimension considered in practice?
15. What is the significance of the "Radius of the level floor" dimension?
16. The angle between the dipper stick and the dipper is typically 65 degrees. Why is this a logical choice?
17. A cross-section through the P&H 4100XPC shovel is shown in Figure 11.5. From a dimension point of view, how does this shovel compare to the competitors? What are their respective capacities?
18. Some of the dimensions for the P&H shovel line are summarized in Tables 11.1 and 11.2. Are there some dimensions which you think should be added to the tables?
19. Figures 11.6, 11.7 and 11.8 show the arrangement of the different pieces of equipment, the facilities and the operating accommodations on the 4100XPC shovel. The operator seems to have all the comforts of home and then some. Summarize these.

20. Using the ballast and payload numbers provided in Table 11.3, perform a simple moment balance for the 4100XPC shovel. What is your conclusion?
21. Define the following terms:
    – Bank weight
    – Loose weight
    – Swell
    – Percent swell
    – Swell factor
    – Fillability
    – Fill factor
22. Can you think of a practical way to get a handle on the fillability factor?
23. Often unit operations are considered as separate entities or even "kingdoms." But there is a very clear connection between the fragmentation achieved and the efficiency of the loading operation. How would one optimize the process, that is, when is it of interest to spend extra resources on the fragmentation?
24. Material has been classified as easy digging, medium digging, medium-hard digging and hard digging. How is this distinction being made? How would you do it?
25. A curve relating the optimum cutting height to the dipper capacity has been prepared based on earlier models of loading machines. Does this curve also represent the new models presented in this chapter?
26. Check the old rule-of-thumb that for good cutting the dipper should fill in a cutting height of 2 to 3 dipper lengths, i.e. compare the dipper stick pivot position to the approximate dipper length.
27. Carefully review and understand the calculations and input data leading to the estimation of production per shift.
28. An old rule-of-thumb is that during an 8-hr shift a shovel should load 1000 short tons/cubic yard of dipper capacity. Hence for the P&H 4100XPC shovel equipped with an 82 yd^3 dipper, one would expect it to load 82,000 tons over an 8-hr shift. Peruse the mining literature and/or check with mining operations to determine the applicability today.
29. Wheel loaders are often used both as primary and secondary excavators in mining operations today. One problem is that they do not have the same digging height capability as rope shovels and thus have problems in scaling bench faces. How is this to be overcome?
30. If one applies the rule that a shovel should fill a truck to capacity in 3 to 5 swings, what size trucks should be assigned to the P&H line of shovels?
31. How is shovel capacity influenced by the number of trucks assigned to the shovel, the loading time and the haul time?

CHAPTER 12

Haulage trucks

12.1 INTRODUCTION

Over the years off-highway trucks have established themselves as the primary means by which both ore and waste are moved in large open pit mines. Because haulage costs can be as much as 50 percent of the total operating costs at a given mine, it is imperative that a thorough understanding of haulage in general, but particularly truck haulage, be achieved. This chapter will focus on some selected aspects of trucks needed by mining engineers in estimating truck requirements, production capacities and costs. In particular, the following topics will be discussed in some detail.

a. Sizing the container.
b. Powering the container
c. Propelling the container: Principles of mechanical and electric drive systems.
d. Estimation of cycle times
e. Calculation of production rates.

There are many other factors which enter into the selection of a haulage system. Unfortunately they cannot be covered here. Although somewhat dated, the reader is encouraged to read the excellent series of articles on the subject written by Burton (1975a, 1975b, 1975c, 1975d, 1976).

12.2 SIZING THE CONTAINER

A haulage truck is in simplest terms, a container (the body) on drive wheels. Generally the overall haulage objective is to accomplish the movement of material from one point to another with the lowest possible cost. The amount of material moved per cycle depends upon the size of the container used. The time required over a given route depends upon the characteristics of the driving mechanism (prime mover, drive train, wheels, tires, etc.). In this section, the characteristics of the container will be examined. The following section will deal with the driving mechanism. The haulage truck is but one element in the transportation chain stretching from the broken material in the pit to the final destination, be it the primary crusher or the dump. A basic consideration is that the loading and hauling equipment should be matched.

Clearly for a given shovel (loader) production capacity (tons/hour) one could assign to the shovel (a) a few large capacity trucks requiring a high number of fill cycles or (b) many small capacity trucks requiring a few fill cycles. The correct "match" obviously depends

Table 12.1. Example truck specification.

Payload = 170 tons (340,000 lbs) Load Capacity (Volume)	(yd^3)	(m^3)
Struck (SAE)	89.5	68.4
Heap 3:1	115.1	87.9
Heap 2:1 (SAE)	126.9	97.0

Table 12.2. Slope angle and angle of repose for various materials (International Payline Division, 1975).

Material	Slope in repose	Angle of repose (degrees)
Bauxite	1.0:1	45
Clay, dry	2.0:1	27
Clay, light	2.0:1	27
Clay, wet	1.0:1	45
Coal, anthracite	1.2:1	40
Coal, bituminous	1.2:1	40
Copper ore	1.0:1	45
Earth, dry	2.0:1	27
Earth, moist	1.0:1	45
Earth, wet	2.0:1	27
Gravel, dry	2.0:1	27
Gravel, wet	2.0:1	27
Granite	1.0:1	45
Iron ore, hematite	1.0:1	45
Limestone, blasted	2.0:1	27
Rock & stone, crushed	2.0:1	27
Sand, dry	3.0:1	18
Sand, wet	2.0:1	27
Shale, soft rock	1.0:1	45
Slate	1.0:1	45
Trap rock	1.0:1	45

upon a number of criteria some of which are site specific. A general rule-of-thumb used in mining is that 3 to 5 shovel swings should be used to fill the truck to capacity.

The meaning of truck capacity is important. If one examines truck specification sheets, one will see a variety of terms being presented. Table 12.1 shows an example of this. The truck in this case is equipped with the standard body.

Standards regarding the calculation of truck and heaped capacity are set by the Society of Automotive Engineers (SAE). The struck capacity is how much water the truck body would hold if brim full. The heaped capacity is the amount of additional material that can be placed above the sides of the container. A slope of 1:1 rises one vertical unit in one horizontal. Different broken materials stand at different slope angles. These angles are referred to as the angle of repose. Values for different materials are given in Table 12.2.

The reason that both volume and weight must be considered is that the loading machine places a certain volume of rock into the truck each swing.

A general rule-of-thumb which is often applied to the sizing of haulage units is the assumption that 1 yd^3 of broken material weighs 3000 lbs. In applying this factor to the body sizes listed in Table 12.1, one obtains the values in Table 12.3.

Table 12.3. Tonnage values corresponding to the volumes in Table 12.1.

Volume Notation	Volume (yd^3)	Tons
Struck (SAE)	89.5	134
Heap 3:1	115.1	173
Heap 2:1 (SAE)	126.9	190

For this standard body and this material, the tonnage capacity is matched by a 3:1 heaped load (115.1 yd^3). If the dipper capacity is 23 yd^3/swing, the truck could be filled to capacity in 5 swings. The concept of swell and swell factors has been discussed in some detail in the loading section. The same principles apply here.

For materials having densities less than 3000 lbs/yd^3, to achieve the rated payload capacity

- either a greater heap would be required,
- or a larger body size should be ordered.

For materials having densities greater than 3000 lbs/yd^3, the rated payload would be achieved by utilizing only a portion of its heaped/struck capacity or a smaller truck body could be ordered. Overfilled trucks result in spillage costs, higher tire cost, higher truck maintenance cost. Under-filled trucks also result in higher costs/ton hauled. Therefore a thorough evaluation of truck capacity within the overall material transport chain must be made.

12.3 POWERING THE CONTAINER

Vehicle motion is achieved by applying a force parallel to the road surface by the rim (outer edge) of the tire. This force which is called the rimpull (RP) or tractive effort (TE) is shown in Figure 12.1.

The radius through which the force is applied is the rolling radius (RR) or static loaded radius of the tire. Hence the wheel torque (W_T) required is

$$W_T = RP \times RR \tag{12.1}$$

where

W_T = wheel torque
RP = rimpull
RR = rolling radius

For a large haulage truck equipped with 27.00–49 tires, the rolling radius is 51.6 ins (4.3 ft). If, in starting from rest, the required rimpull is 15,000 lbs, the wheel torque would be 64500 ft-lbs. The rotary power required to rotate the wheel at n_{wheel} rotations per minute is

$$W = \frac{2\pi n_{wheel} W_T}{33000} \tag{12.2}$$

where

W = required rotary power (HP)
n_{wheel} = wheel rotation rate (RPM)
W_T = wheel torque (ft-lbs)
33000 = factor required to convert ft-lbs/minute to horsepower.

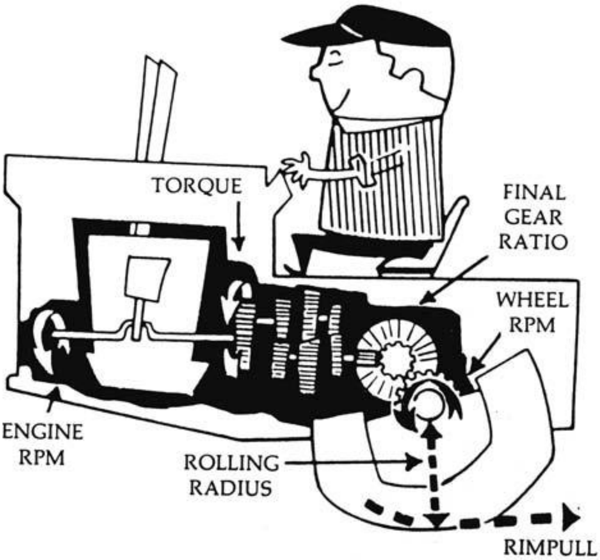


Figure 12.1. Diagrammatic representation of the driving elements. Gessel (1977).

For different operating and road conditions (grade, surface, etc.) the required amount of wheel torque and rotation varies. To start a vehicle from rest, for example, the required torque is high and the rotation rate low. To maintain speed on a level haul, the rotation rate is high and the torque low.

Gross horsepower is the rating of an engine as equipped with muffler, air cleaner, water, and fuel pumps, etc. Net or flywheel horsepower is the rating after deducting fan, generator, alternator and air compressor requirements. These items as a group are sometimes called the "parasitic load".

12.4 PROPELING THE CONTAINER – MECHANICAL DRIVE SYSTEMS

12.4.1 *Introduction*

For mechanical drive systems, the power train (as shown in Fig. 12.2) consists of the

a. engine,
b. torque converter,
c. gear ratios (including transmission), and
d. tire radius.

The torque converter is a device which hydraulically multiplies engine torque. At high engine speed and low vehicle speed, the torque converter automatically multiplies engine torque providing the force necessary to get the unit moving faster. At higher vehicle speeds, no torque multiplying is required and the torque converter transmits engine torque (no multiplying) at engine speed (less losses due to hydraulic inefficiencies). This transmission of engine torque can be made nearly 100% efficient (like a direct drive transmission) by locking the torque converter pump and turbine together. A torque converter with this feature is said to have "lock up" (Terex, 1981).

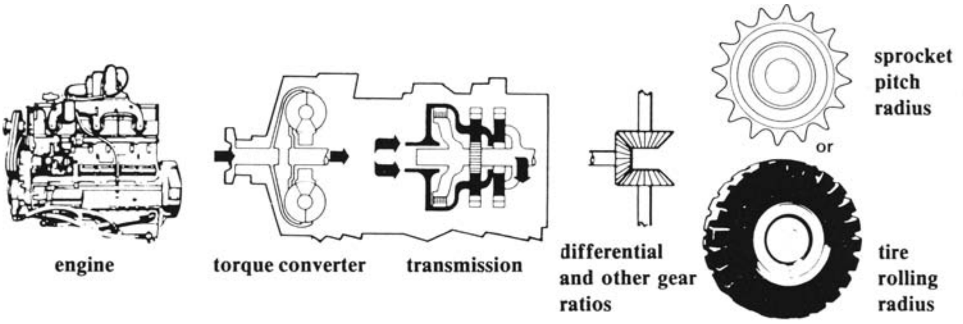


Figure 12.2. Power train of a mechanical drive system. Terex (1981).

Figure 12.3. The Terex 33-09 truck. Terex (1974).

Gear ratios provide an increase in the torque and a resultant reduction in speed. The total gear reduction (TGR) is a multiple of all the gear ratios which make up the gear train; i.e. transmission gear ratio × differential ratio × planetary ratio (Terex, 1981).

The total efficiency of transmission and final drive gears in off-highway equipment can be estimated at 85% for step gear and clutch power trains, and 75% for torque converter-power shift transmission. This would increase to 85% in lockup ranges if the converter has a lockup clutch (Gessel, 1977). The net result is horsepower developed through a certain set of torque – rotation rate characteristics available at the wheel hub. This can be used to propel the vehicle.

In order to facilitate the discussion, the Terex 33-09 Hauler (Terex, 1974) shown in Figure 12.3 has been selected as an example for use in this section.

The power train elements are described in Table 12.4.

The rolling radius values for the different tire choices are provided in Table 12.5.

Figure 12.4 presents drawings of the Terex 33-09 truck.

The weights and capacities for the truck are provided in Table 12.6.

Table 12.4. Characteristics of the Power Train Elements for a Terex 33-09 Hauler. Terex (1978).

ENGINE
Detroit Diesel 16V-71T Turbocharged, 2 Cycle Diesel

Gross Horsepower @ 2100 RPM	665
Flywheel Horsepower @ 2100 RPM	624

NOTE: Above are SAE ratings at 500 ft. altitude and 85°F (29°C). Gross horsepower rating includes standard engine equipment, such as water pump, fuel pump, ana lubricating oil pump. Flywheel horsepower is the net horsepower after deductions from gross horsepower for fan, alternator, and air compressor requirements. Turbocharged engine requires no deration to 10,000 ft. altitude.

Number of Cylinders	16
Bore and Stroke	4¼″ × 5″ (108 mm × 127 mm)
Piston Displacement	1136 cu. in. (18.6 liters)
Maximum Torque @ 1400 RPM	1840 ft-lbs (254 kg m)
Fan Diameter	48.74 in. (124 cm)
Radiator Frontal Area	20 sq. ft. (18,580 cm^2)
Air Starting System	125 psi with neutral start feature

TRANSMISSION – Allison CLBT-6061
Allison transmission with integral torque converter and manual electric shifting mechanism mounted amidship in frame. Six speeds forward, one reverse. Automatic converter lock up in all forward speed ranges. Standard downshift inhibitor prevents downshifting into lower speed ranges at high engine speeds. Hydraulic retarder is standard.

Stall speed	2000 RPM

Maximum speeds @ 2100 RPM:

	MPH (km/hr)
Standard Axle	42.4 (68.3)
Optional Axle	36.0 (58.0)

DRIVE AXLE
Heavy duty, full floating axle shafts with single reduction spiral bevel gear differential and planetary final reduction in each wheel.

	Standard Axle Ratio	Optional Axle Ratio
Ratios: Differential	3.15:1	3.73:1
Planetary	5.80:1	5.80:1
Total Reduction	18.27:1	21.63:1

TIRES AND RIMS (Tubeless)

Bias Type:		
Standard Front & Rear	24.00-35-36 PR, E3	17″ (432 mm)
Optional Front & Rear	24.00-35-36 PR, E4	17″ (432 mm)
	24.00-35-42 PR, E3	17″ (422 mm)
	24.00-35-42 PR, E4	17″ (432 mm)
Optional Radial Type:		
	24 R35 UE-7L (X)	17″ (432 mm)
	24.00-35 XRB Two Star	17″ (432 mm)

12.4.2 *Performance curves*

There are a number a different engine-transmission-gear-tire combinations possible for a given truck. For each available combination, the manufacturers provide curves of rimpull as a function of vehicle speed. Different characteristic curves apply for the vehicle traveling

Table 12.5. Rolling radius values.

Tire Type	Rolling Radius (ins)
Standard Front and Rear	
24.00–35(36 PR) E-3	38.7
Optional Front and Rear	
24.00–35(36 PR) E-4	39.7
24.00–35(42 PR) E-3	38.7
24.00–35(42 PR) E-4	39.7

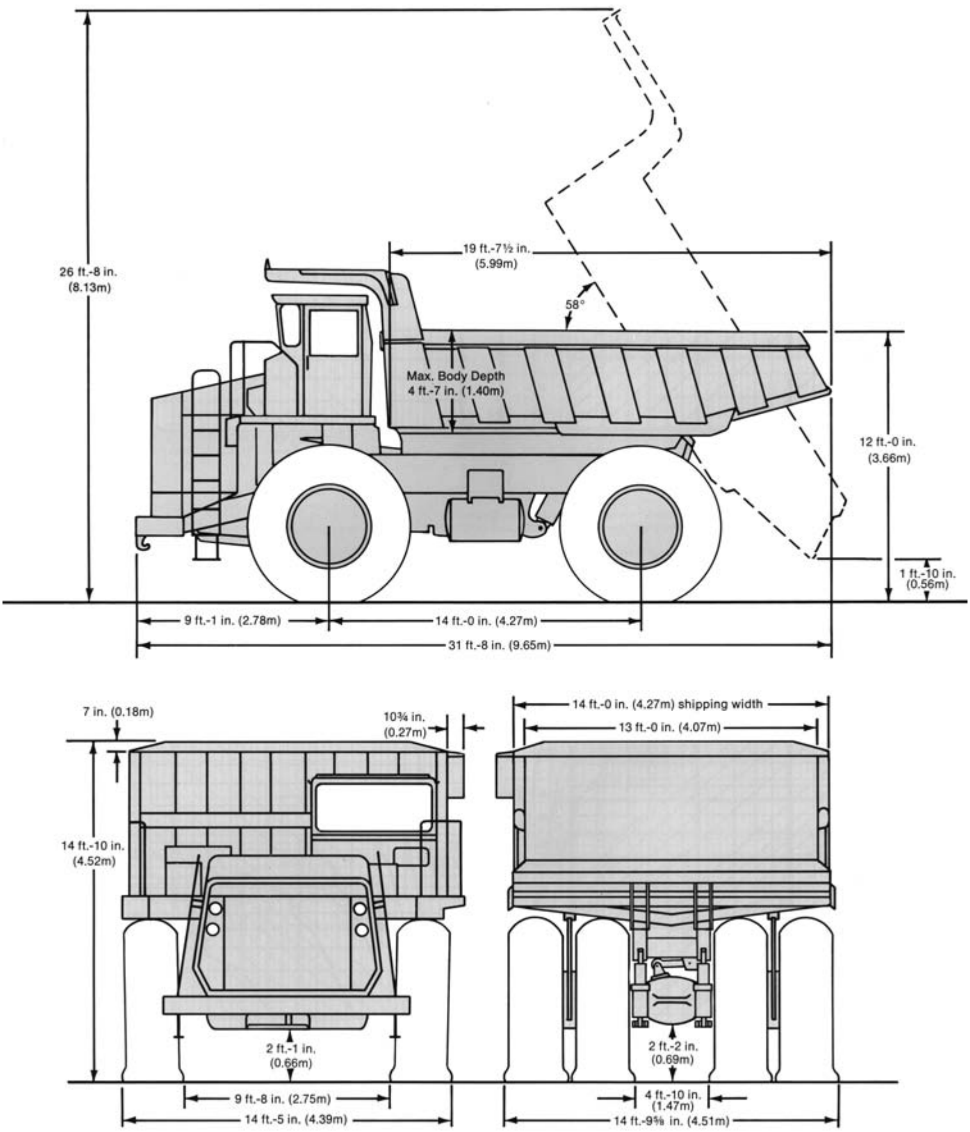


Figure 12.4. Drawings of the Terex 33-09 truck. Terex (1978).

Table 12.6. Weights and capacities for the Terex 33-09 truck (1978).

CAPACITY		
Struck (SAE)	33.7 cu. yds.	(25.7 m^3)
Heaped 1:1	54.5 cu. yds.	(41.6 m^3)
Heaped 2:1 (SAE)	44.1 cu. yds.	(33.7 m^3)
Heaped 3:1	40.6 cu. yds.	(31.1 m^3)
WEIGHTS		
NET WEIGHT DISTRIBUTION		kg
Front Axle	45,200 lbs.	(20,503)
Rear Axle	48,000 lbs.	(21,773)
Total	93,200 lbs.	(42,276)
PAYLOAD	110,000 lbs.	(49,896)
GROSS WEIGHT DISTRIBUTION		
Front Axle	67,800 lbs.	(30,754)
Rear Axle	135,400 lbs.	(61,418)
Total	203,200 lbs.	(92,172)
Chassis with Hoists	70,650 lbs.	(32,048)
Standard Body	22,550 lbs.	(10,228)
Full Liner Weight (Optional)	10,750 lbs.	(4,870)

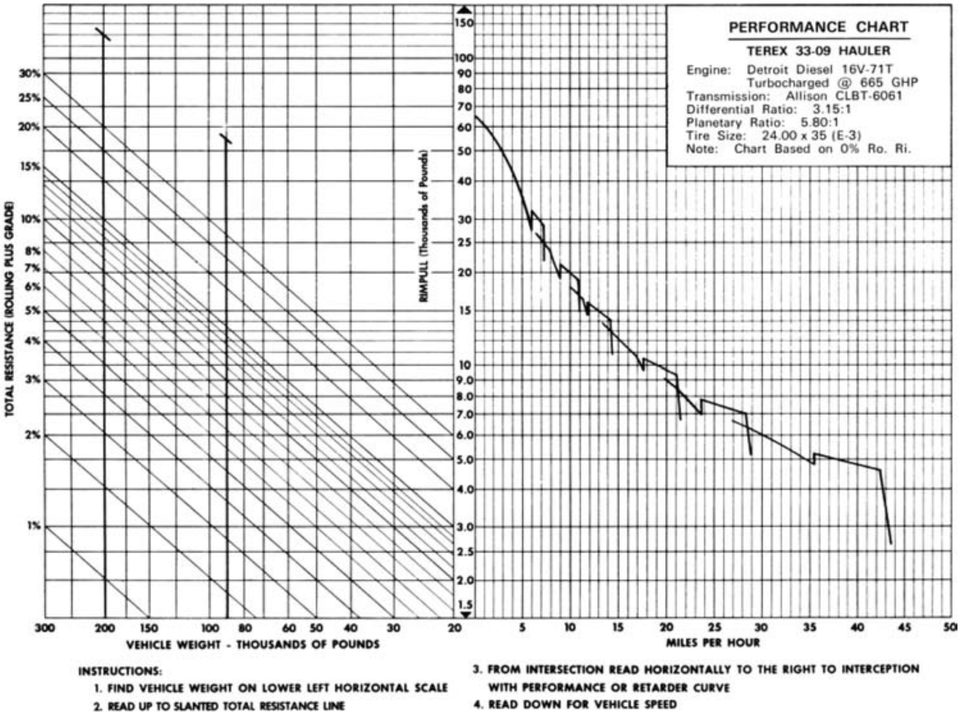


Figure 12.5a. The performance curve for the Terex 33-09 Hauler including the speed calculator. Terex (1978).

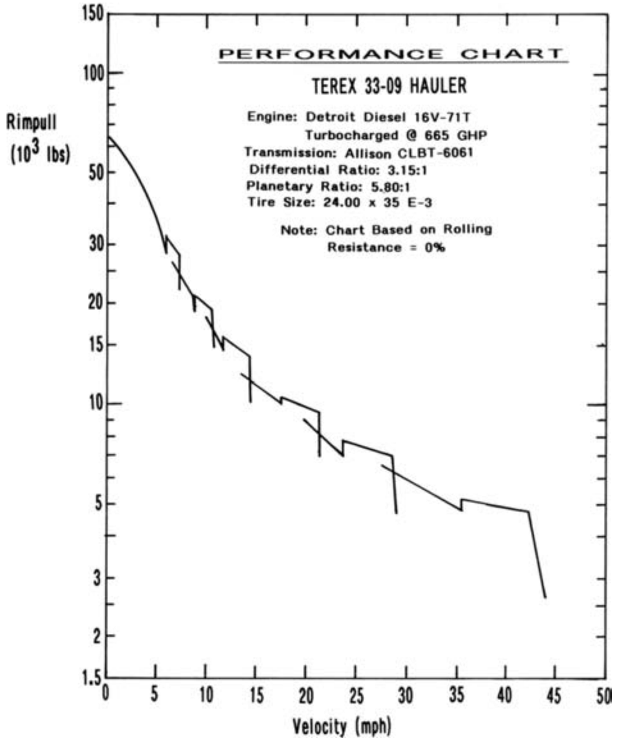


Figure 12.5b. Performance curve for a Terex 33-09 Hauler (Terex, 1978).

over adverse (uphill) and favorable (downhill) grades. This section will examine the curves applicable for uphill road sections. These are called "performance" curves.

Figure 12.5a shows the relationship between rimpull and vehicle speed for the Terex 33-09 Hauler for the following arrangement of components.

Engine: Detroit Diesel 16V-71T Turbocharged @ 665 GHP
Transmission: Allison CLBT-6061
Differential Ratio: 3.15:1
Planetary Ratio: 5.80:1
Tire Size: 24.00 × 35 E-3

The gross vehicle horsepower is 665 and the flywheel power is 624. Figure 12.5b shows a simplified version of Figure 12.5a with only the relationship between rimpull and vehicle speed provided.

It is considered instructive to try and demonstrate to the reader in a simple way the basis for such a curve. Simply considering units one can see that horsepower can be written as

$$\text{hp} = \frac{\text{force (lbs)} \times \text{velocity (ft/min)}}{33000} \tag{12.3}$$

For velocity expressed in miles per hour rather than feet/minute, equation (12.3) becomes

$$\text{hp} = \frac{\text{force (lbs)} \times \text{velocity (mph)}}{375} \tag{12.4}$$

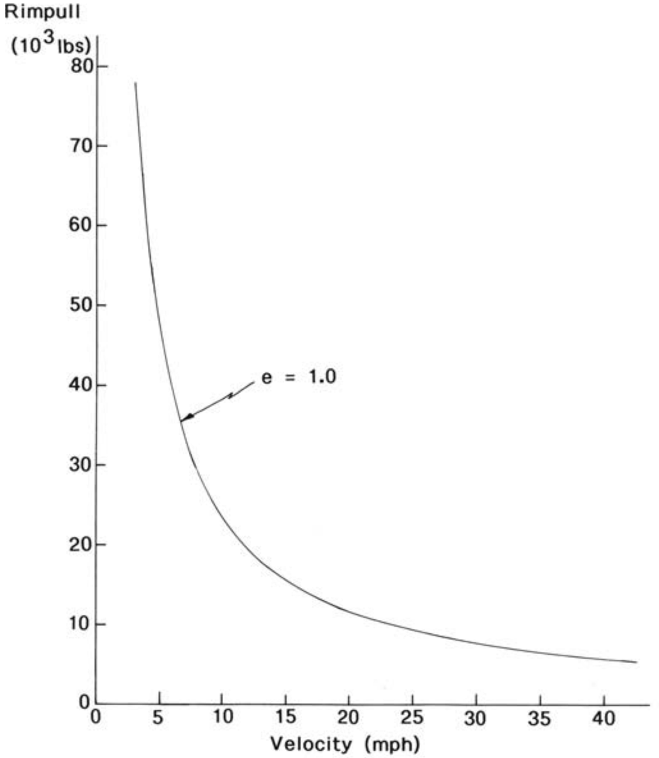


Figure 12.6. Theoretical performance curve for e = 1.

Comparing the terms in equation (12.4) to those quantities in Figure 12.3, it can be seen that the force is just the rimpull and the horsepower (hp) is that available at the wheel. If the horsepower (hp) is known, then the curve in Figure 12.5 could be constructed. The flywheel horsepower is known to be 624 hp but there are losses between the engine and the wheel. These losses can be expressed in terms of an efficiency factor (e)

$$\text{hp} = \text{e} \times \text{flywheel horsepower} \tag{12.5}$$

where

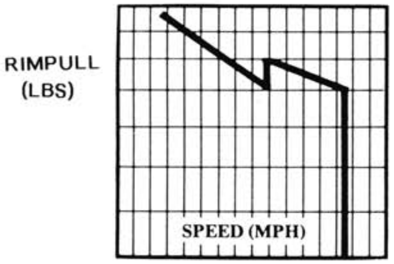
e = efficiency factor

The maximum value of e is 1.0 which means that no loss occurs. The curve corresponding to this value is shown on Figure 12.6.

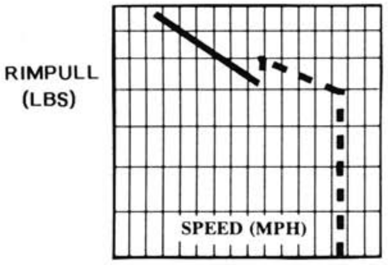
Next, one must consider how the shifting of gears occurs. For the Terex truck there are 6 forward gears. This shifting from one gear to another takes place in the torque converter. This acts similarly to the clutch in a standard shift automobile. In the case of an automobile, if you are in 1st gear, the motor is in direct drive with the wheels. In shifting into second gear, the clutch is depressed and second gear is engaged. Again the motor directly drives the wheels. In a large haulage truck during the shifting process the torque converter is in action. Eventually the transmission is "locked-in" to a particular gear and the engine is in direct drive once again. Direct drive is called "lock-up".

This process is shown diagrammatically in Figure 12.7.

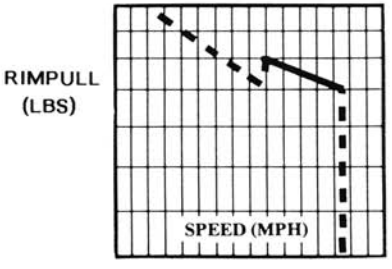
(a) For each transmission change, the total curve is as shown below



(b) That portion of the curve representative of the torque converter power is shown by the solid line



(c) The portion of the curve representative of lock-up/direct drive is shown by the solid line



(d) Within a given gear range, the vertical line to the left of the lock-up portion represents the transition between converter and lock-up

(e) The vertical line to the right of the lock-up curve represents the maximum RPM of the engine and thus the maximum speed for that range.

(f) The combination of these elements for the different gear ranges forms the rimpull versus speed curve.

Figure 12.7. Diagrammatic representation of the torque converter and direct drive process. Terex (1974).

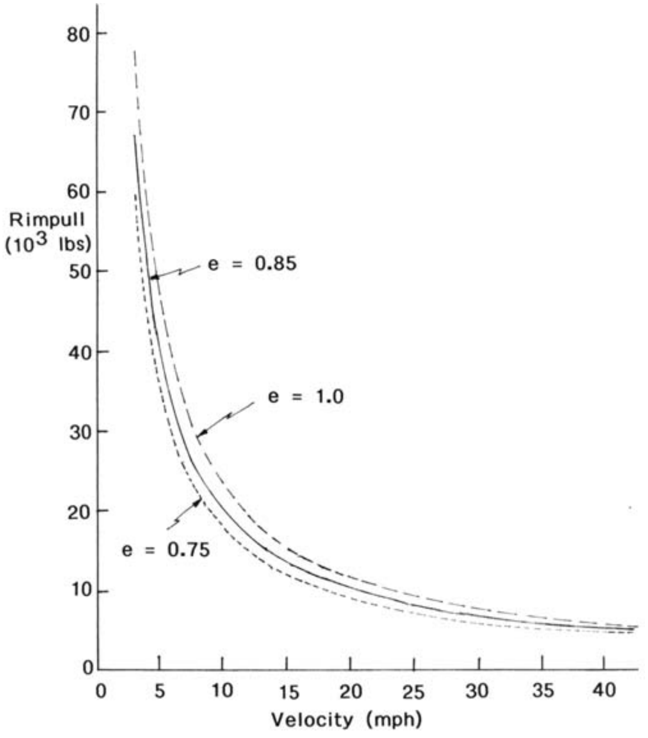


Figure 12.8. Addition of the e = 0.75 and e = 0.85 efficiency lines to the theoretical performance curve.

Table 12.8. Location of gear change events. Terex (1974).

Gear	Torque Converter	Lock-up	Gear Change
1	0.0	6.0	7.3
2	7.3	8.9	10.6
3	10.6	11.8	14.2
4	14.2	17.5	21.3
5	21.3	23.7	28.5
6	28.5	35.4	42.4* (Max speed)

The efficiency values are different in the torque converter and direct drive modes.

$$e_{torque\ \ converter} = 0.75$$

$$e_{lock\text{-}up} = 0.85$$

The curves corresponding to these two efficiency values have been plotted on Figure 12.8.

From the performance curve (Fig. 12.5) the event times have been read. These are given in Table 12.8.

Superimposing the values at which the efficiencies change on Figure 12.8, one gets the expected overall performance curve for propulsion shown in Figure 12.9.

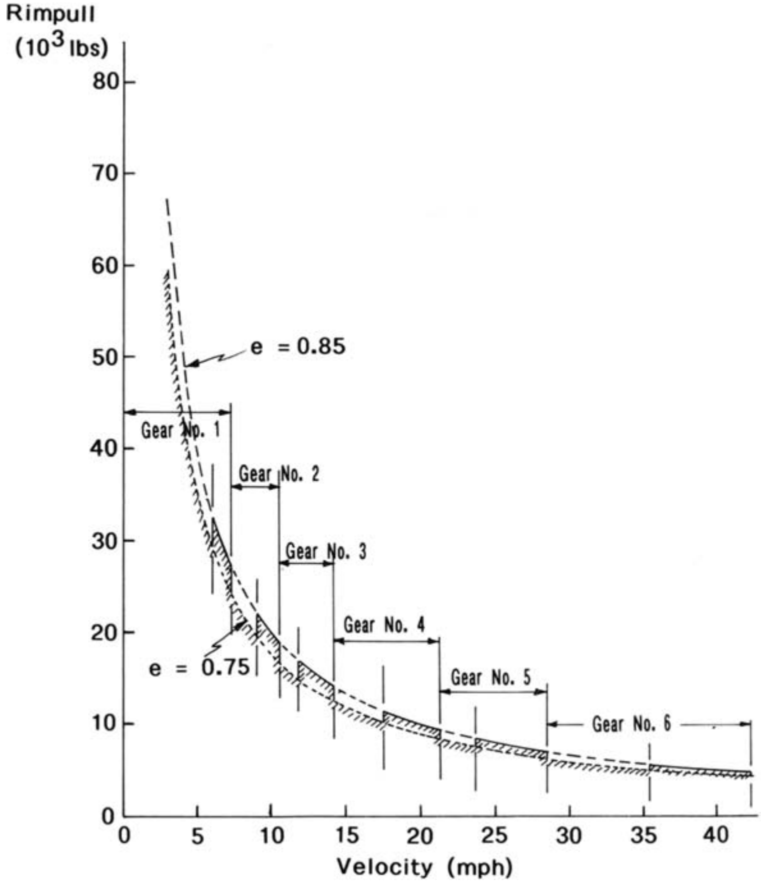


Figure 12.9. Addition of the gear change events on the curves in Figure 12.8.

The curve given by the manufacturer has been superimposed on the estimated curve in Figure 12.10. As can be seen, the agreement is very good for speeds greater than about 5 mph.

12.4.3 *Rimpull utilization*

The rimpull discussed in the previous section is the force available at the rim of the tire. Here, three important practical concepts will be presented. First, the amount of rimpull which can actually be developed will be examined. Next, the tractive effort required to move a vehicle at constant velocity along a horizontal surface will be considered. Finally, the amount of tractive effort required to propel a vehicle at a constant velocity up a constant grade will be evaluated.

Maximum developed rimpull
Tire treads are designed and selected so that they develop a certain degree of traction (friction) between the tire and the road surface. Figure 12.11 shows the rimpull force (RP) and the normal force (N) acting between the tire and the road surface.

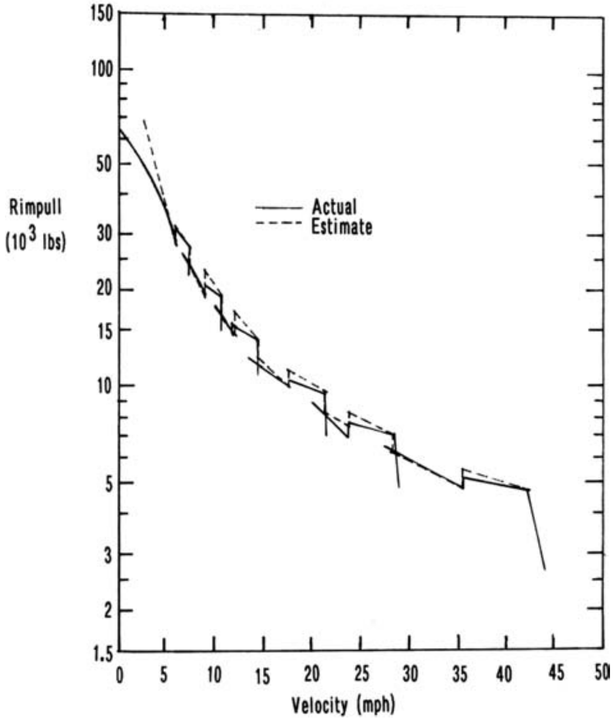


Figure 12.10. Derived performance curve with the actual superimposed.

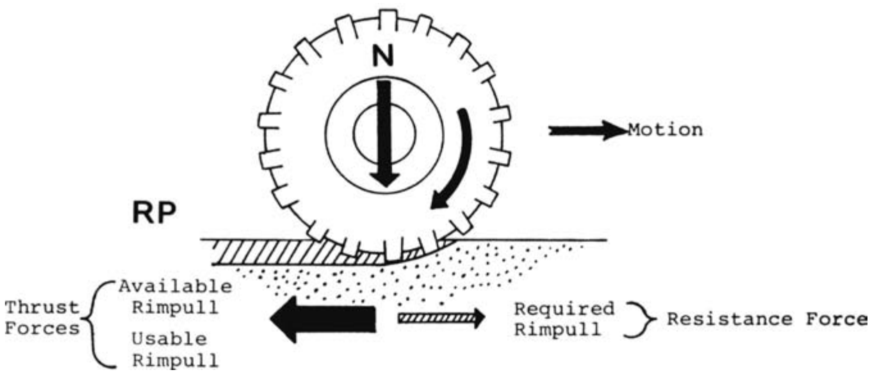


Figure 12.11. Diagrammatic representation of the normal and rimpull forces.

As the normal force is increased, the amount of rimpull which can be exerted without the tire spinning also increases. The relationship between the maximum rimpull and the normal force can be written as

$$RP_{\max} = \mu N \tag{12.6}$$

where

RP = rimpull

μ = coefficient of traction

N = weight component on drive wheels normal to the surface.

Table 12.9. Coefficient of traction values for rubber tires (International Payline Division, 1975).

Type of surface	Dry surface	Wet surface
Smooth blacktop	0.8–1.0	0.6–0.9
Rough concrete	0.9–1.0	0.9–1.0
Hard smooth clay	0.6–1.0	0.1–0.3
Hard clay loam	0.5–0.8	0.15–0.4
Firm sandy loam	0.4–0.8	0.25–0.8
Spongy clay loam	0.4–0.6	0.15–0.3
Rutted clay loam	0.3–0.5	0.15–0.3
Rutted sandy loam	0.3–0.4	0.2–0.5
Gravel road, firm	0.5–0.8	0.3–0.9
Gravel, not compacted	0.3–0.5	0.4–0.6
Gravel, loose	0.2–0.4	0.3–0.5
Sand, loose	0.1–0.2	0.1–0.4
Snow, packed	0.1–0.4	0.0–0.3
Ice, roughened	0.1–0.3	0.0–0.2
Ice, smooth	0.0–0.1	0.0–0.1

For an empty truck, the weight of the truck is normally about equally distributed on the front and rear axles (2 axle truck). For a truck loaded to capacity, about 2/3 of the gross weight is on the rear (drive) axle. For the Terex 33-09 truck this means

W = weight on drive wheels (empty) = ½(93200) = 46600 lbs

W = weight on drive wheels (full-loaded) = ½(203200) = 136144 lbs

Some values for the coefficient of traction are given in Table 12.9.

For the truck at rest on a horizontal surface with a coefficient of 0.6, the maximum rimpull which could be delivered before the wheels start to spin is

RP_{max}(empty) = 0.6(46600) = 28000 lbs

RP_{max}(loaded) = 0.6(136144) = 81700 lbs

From the performance curve shown in Figure 12.5, the maximum available rimpull for the truck starting from stop is about 64000 lbs. This value is more than that which could be generated by the empty truck but less than that for the loaded truck. Thus

RP_{limit}(empty) = 28000 lbs

RP_{limit}(loaded) = 64000 lbs

If the truck was at rest on an uphill grade of 10%, for example, the component of the weight acting normal to the road surface would first be calculated before applying the traction coefficient.

$$N = W \text{ (drive wheels)} \times \cos\theta$$

where

θ = slope angle

Tractive effort calculation and application

The force required to propel a truck under different situations is called the tractive effort. If this value is compared to the available rimpull, the speed which can be maintained as well as well as the acceleration/deceleration rates can be determined. The tractive effort requirements for two simple, but basic, situations will be illustrated below.

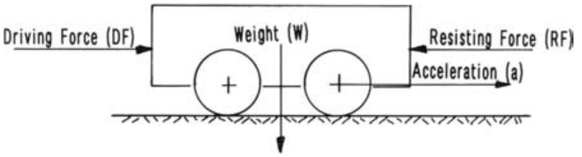


Figure 12.12. Diagrammatic representation of a truck moving along a level road.

Case 1: Tractive effort required to maintain a constant velocity on a horizontal surface.
In Figure 12.12 a diagrammatic representation of a truck moving along a haulage road is shown.

The difference between the driving force and the resisting force on the truck is the net force. The net force acting on the mass results in truck acceleration or deceleration. The basic formula which will be used in all the examples is

$$F_{net} = DF - RF = \frac{w}{g}\bar{a} = m\bar{a} \tag{12.7}$$

Assume a truck traveling at a constant velocity over a horizontal surface. In this case the acceleration is equal to zero, and the driving force is the tractive effort supplied by the truck. The tractive effort is equal in magnitude and opposite in direction to the resisting force. The resisting force depends upon a number of factors including:

- road condition (degree of penetration)
- internal friction in bearings, etc.

It is calculated by multiplying the vehicle weight times a factor called the rolling resistance.

$$\text{RF (lb)} = \text{R}_\text{o}\text{R}_\text{i} \times \text{Vehicle Weight (lb)} \tag{12.8}$$

where
$\text{R}_\text{o}\text{R}_\text{i}$ = rolling resistance expressed as a ratio.

In the literature the rolling resistance is usually expressed as a percent (i.e. 2%) or as a ratio (i.e., 0.02 or equivalently 40 lbs/ton). The force due to rolling resistance always opposes the driving force.

A vehicle weighing 203200 lbs traveling on a level road of 2% rolling resistance must develop 4060 lbs of rimpull to maintain a constant velocity. From Figure 12.5, it can be seen that a velocity of 43 mph corresponds to this rimpull value. Some common values for rolling resistance are given in Table 12.10.

Case 2: Tractive effort required to propel a truck (Fig. 12.13) at constant speed up a grade inclined upward at θ degrees with the horizontal.

The rolling resistance is maintained at $\text{R}_\text{o}\text{R}_\text{i}\%$.

The resisting force is now made up of two components. The first, as discussed in the previous example, is that due to the rolling resistance

$$\text{RF}_1 = \text{R}_\text{o}\text{R}_\text{i} \times \text{Vehicle Weight (lb)} \tag{12.9}$$

The second is due to the component of the truck weight acting parallel to the slope

$$\text{RF}_2 = \text{Vehicle Weight} \times \sin\theta \tag{12.10}$$

where
θ = slope angle (degrees)

Table 12.10. Commonly applied rolling resistance values. Terex (1981).

Ground Surface	Rolling Resistance
Asphalt	1.5%
Coal, crushed	5–7%
Concrete	1.5%
Dirt-Smooth, hard, dry; well-maintained; free of loose material	2.0%
Dirt-Dry, but not firmly packed; some loose material	3.0%
Dirt-Soft, unplowed; poorly maintained	4.0%
Dirt-Soft, plowed	8.0%
Dirt-Unpacked fills	8.0%
Dirt-Deeply rutted	16.0%
Grave-Well compacted; dry; free of loose material	2.0%
Grave-Not firmly compacted; buty dry	3.0%
Gravel-Loose	10.0%
Mud-With firm base	4.0%
Mud-With soft, spongy base	16.0%
Sand-Loose	10.0%
Snow-Packed	2.5%
Snow-To 4″ depth; loose	4.5%

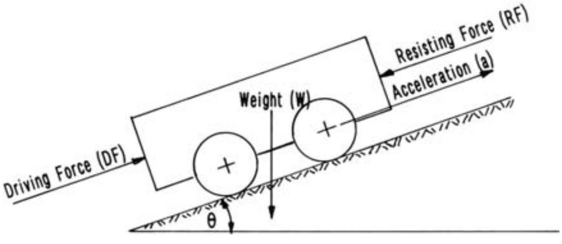


Figure 12.13. Diagrammatic representation of a truck moving up a slope.

Table 12.11. Relationship between grade (percent) and grade (degrees).

Grade (%)	θ (degrees)	$\sin\theta$	$\tan\theta$ = Grade Ratio (GR)
0	0	0	0
1	1.146	0.020	0.020
4	2.291	0.040	0.040
6	3.434	0.599	0.060
8	4.574	0.0797	0.080
10	5.711	0.0995	0.100
12	6.843	0.119	0.120
14	7.970	0.139	0.140
16	9.090	0.158	0.160

For small values of θ, the following holds

$$\sin\theta \cong \tan\theta = \frac{rise}{run} = \frac{\%grade}{100} = GR \tag{12.11}$$

This is shown in Table 12.11.

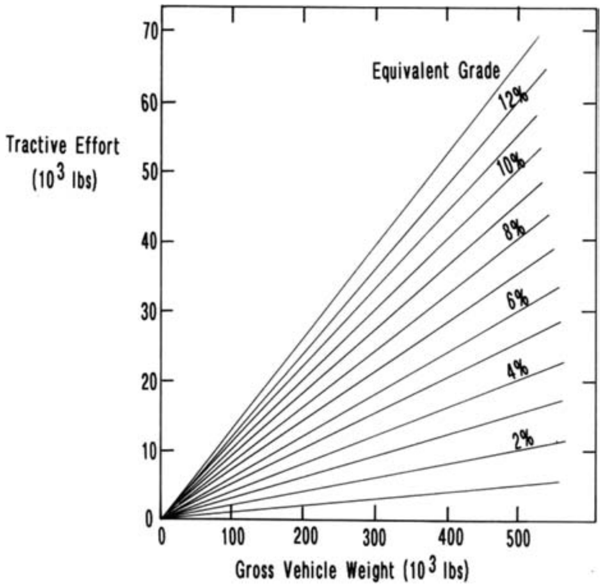


Figure 12.14. Tractive effort versus gross vehicle weight as a function of equivalent grade.

As can be seen, for the range of grades normally encountered in open pit mines, the replacement of sin θ by tan θ (which is equal to the grade expressed as a ratio) introduces little error. Thus

$$RF_2 = GR \times \text{Vehicle Weight} \tag{12.12}$$

where
GR = grade resistance expressed as a ratio (rise/run).
The total resisting force becomes

$$RF = RF_1 + RF_2 = \text{Vehicle Weight}\ (R_oR_i + GR) \tag{12.13}$$

If the vehicle weighs 203200 lbs, the rolling resistance is 2% and grade resistance is +10% (up), the tractive effort required to maintain a constant velocity is

$$TE = 203200(0.02 + 0.10) = 24384\,\text{lbs}.$$

The velocity which can be maintained is 7 mph.

To avoid constant calculation of this force, plots of the tractive effort versus gross vehicle weight (GVW) graph with equivalent grade lines plotted on it are used in conjunction with the performance (and retardation) curves. Figure 12.14 is one such example.

To use the curve simply locate the GVW line required and follow it up to the total equivalent grade line (grade resistance plus rolling resistance). Find the intersection of the GVW line and the equivalent grade curve. Then move horizontally to the vertical axis. The required force (tractive effort) can be read at the left side of the graph.

12.4.4 *Retardation systems*

In the previous section, *performance* curves were discussed. They are called this because the overall cycle time is greatly affected by how well the truck performs under adverse grades (working against the effects of gravity). The generated horsepower is used to perform useful

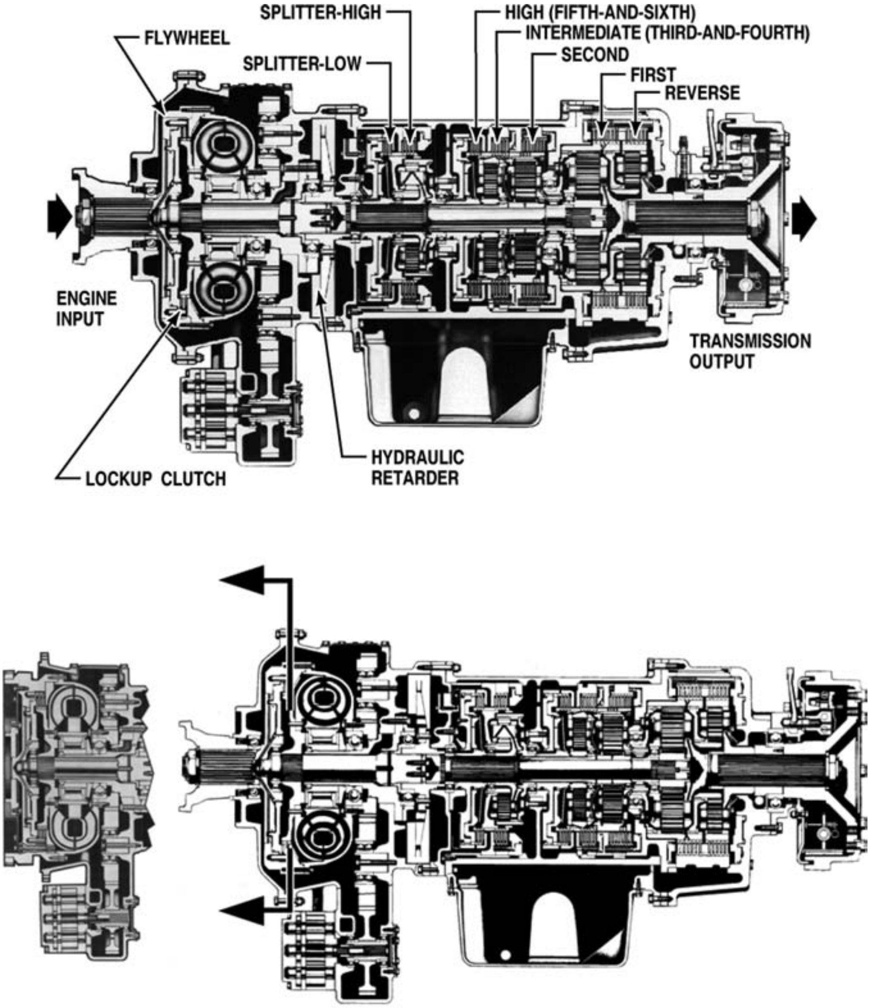


Figure 12.15. Cross-sectional view of engine, retarder and transmission. Allison (2012). Included with permission.

work. A portion of the cycle is generally conducted under favorable grades in which gravity assists the truck. This section will deal with the means by which the truck can be slowed and stopped. There have been two approaches used by truck manufacturers.

1. Retarding systems
2. Braking systems.

In Figure 12.15, a hydraulic retarder (a dynamic braking device) is shown to lie between the torque converter and the transmission.

An exploded view of a retarder is shown in Figure 12.16.

As can be seen, the retarder consists of chamber in which an impeller mounted to the drive shaft rotates. The impeller consists of a series of paddles. When retardation is desired, oil is pumped into the chamber. The paddles pick up the oil and toss it against baffles.

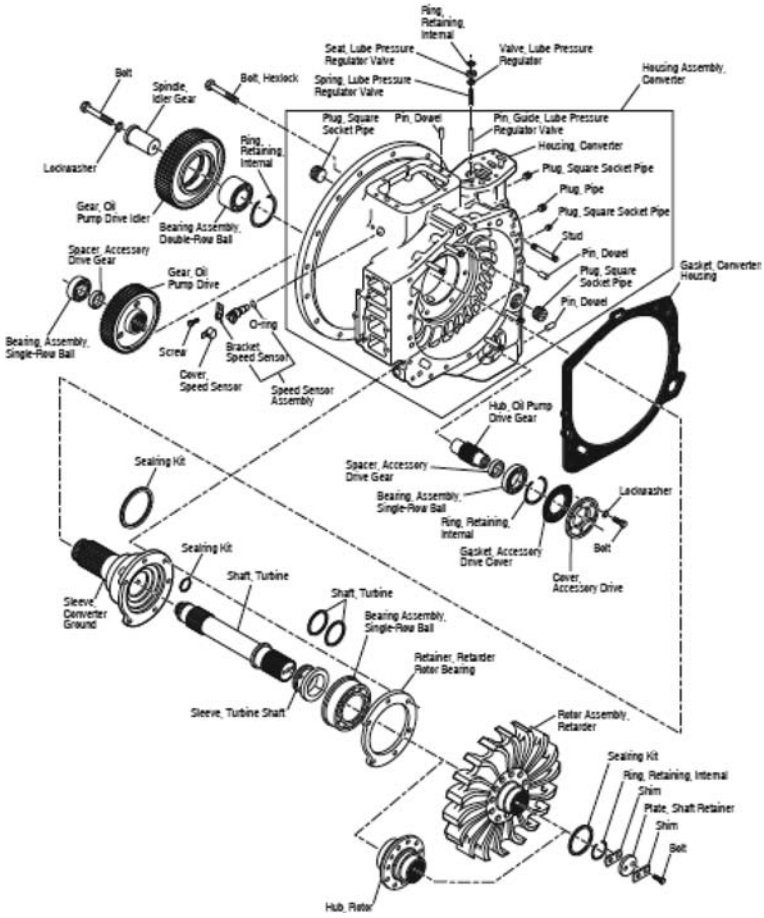


Figure 12.16. Exploded view of a retarder. Allison (2012). Included with permission.

The baffles are oriented so that they splash the oil back toward the paddles. In this way the drag on the drive train can be increased or decreased by adding or removing oil from the retarder chamber. When going downhill with very little fuel being injected in to the engine, the retarder is driven by the driving wheels through the transmission gear ranges. Filling the retarder chamber with oil slows the paddle wheel and consequently, slows the driving wheels and the unit. A retarder works by converting mechanical power in the paddle wheel into heat in the oil. This heat is dissipated through the unit's cooling system. The retarder will not provide speed control beyond its own rimpull capabilities. At low speeds, the viscosity of the oil and the splashing effect is relatively small. Hence the retarder is used for slowing but not stopping the truck. Stopping is accomplished using the service brakes. Figure 12.17 shows the retardation curves for the Terex 33-09 truck.

Such curves are developed based on data supplied by the transmission manufacturer modified to account for special characteristics of the truck. In the case shown, there are six gear ranges. Just as discussed with the performance curves, there is a torque converter and lockup portion (Fig. 12.18) in each gear range.

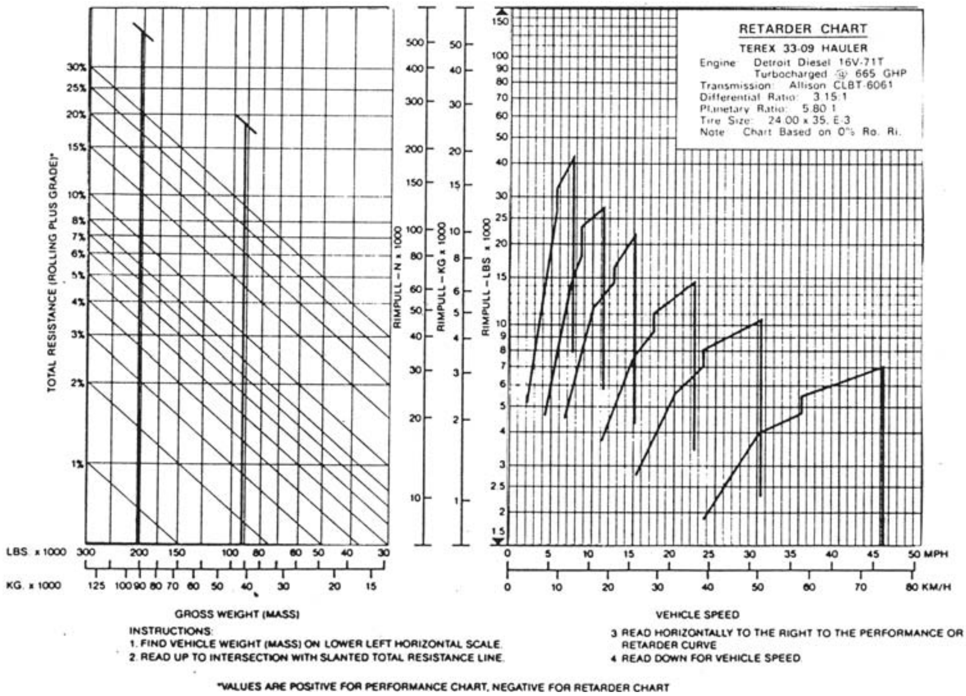


Figure 12.17. Retardation curve for the Terex 33-09 Hauler. Terex (1978).

The highest retardation rimpull occurs in lock up. As can be seen, the total retardation available depends both upon the gear which the truck is in and the retarder. In high gears, the gear effect is low whereas in low gear ranges it is high.

The retarding force needed to balance truck acceleration can easily be obtained by drawing a free-body diagram of the situation. As an example of the procedure, consider a truck traveling at constant speed down a grade of slope X% having a rolling resistance of R_0R_i% (Fig. 12.19).

The force balance is

$$F_{net} = \text{Resisting Force (RF)} + W \times GR - W \times R_oR_i = \frac{W}{g} \times \bar{a} \tag{12.14}$$

To prevent the truck from accelerating down the decline

$$\bar{a} = 0$$

F_{net} becomes

$$F_{net} = \text{Resisting Force (RF)} + W \times GR - W \times R_oR_i = \frac{W}{g} \times \bar{a} = 0 \tag{12.15}$$

Hence,

$$RF = W(R_oR_i - GR) \tag{12.16}$$

For a grade resistance greater than the rolling resistance, a resisting force in the form of braking or retarding must be applied to avoid the truck from accelerating down the decline.

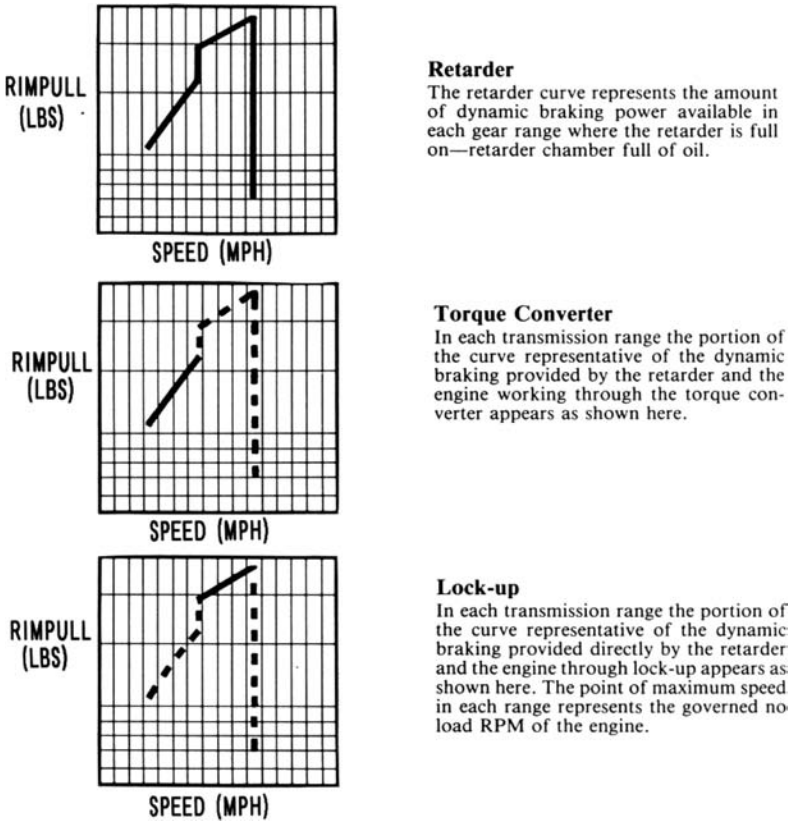


Figure 12.18. Enlarged view of a torque converter – lock-up section of a retarder curve. Terex (1981).

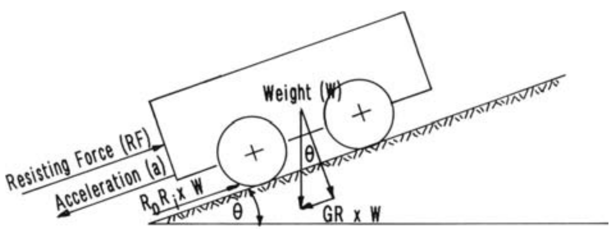


Figure 12.19. Diagrammatic representation of a truck traveling down a decline.

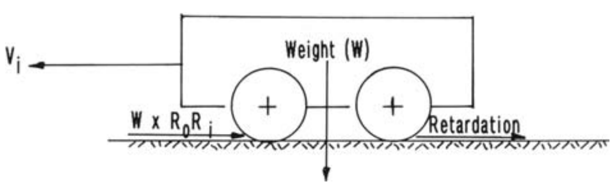


Figure 12.20. Truck traveling along a horizontal segment.

Assume that a fully loaded Terex 33-09 truck is truck traveling at 30 mph along a horizontal surface having a rolling resistance of 2% (see Fig. 12.20).

If the truck is simply allowed to coast to a stop (no retardation or braking force is applied), the truck would slow down due to the action of the rolling resistance alone. The stopping distance can be found in the following way. Using equation (12.13) one finds that the average deceleration is

$$-W \times R_oR_i = \frac{W}{g} \times \bar{a} \tag{12.17}$$

Or

$$\bar{a} = -0.644\,\text{ft/sec}^2$$

The stopping distance (x) is found using the familiar equation

$$V^2 = V_o^2 + 2\bar{a}x \tag{12.18}$$

Where
V = final velocity
V_o = initial velocity
x = distance traveled
$\bar{a}$ = average acceleration

Substituting the appropriate values in equation (12.17) one finds that

$$x = 1503\,\text{ft}$$

If, instead of being able to roll unhindered to a stop, a retardation force of 10,000 lbs. was continually applied, the deceleration would be

$$\bar{a} = -2.23\,\text{ft/sec}^2$$

and the truck would stop in

$$x = 434\,\text{ft}$$

For a truck rolling down a negative grade of 2%, at least in theory, the truck would maintain a constant velocity even with the engine shut off. As the slope of the downward grade is increased, the truck would accelerate unless the retarder or the brakes were applied.

Assume a vehicle weighing 203200 lbs enters a stretch of road for which the rolling resistance is 2% and the grade is −8%. The total resistance is thus −6%. The retarding force required to keep the vehicle from accelerating is

$$RF = -0.06 \times 203200 = -12192\,\text{lbs.}$$

If the vehicle enters the grade at a speed of 30 mph, the available retardation as read from Figure 12.17 is only about −10400 lbs. If nothing more were done, the vehicle would continue to accelerate down the grade. However the service brakes can be applied to reduce the speed to below 22.5 mph where the retarder would once again control the speed of the vehicle.

The values in Table 12.12 refer to the peak values for the different gear ranges.

For a given gross vehicle weight and total resistance, retarder charts provide a wide range of speeds over which the truck remains under retarder control (see Fig. 12.21).

If one uses the maximum speed (point a) then an optimistic estimate of travel time would be calculated. On the other hand use of the minimum value (point b) would yield a very conservative estimate. The true value is probably somewhere in between. Some of the factors which affect this are

- road conditions
- time of day

Table 12.12. Peak values as read from the retardation curve.

Velocity (mph)	Retarder Force (lbs)
7.6	−42400
11.4	−28500
15.1	−21300
22.5	−14300
30.4	−10600
45.4	−7100

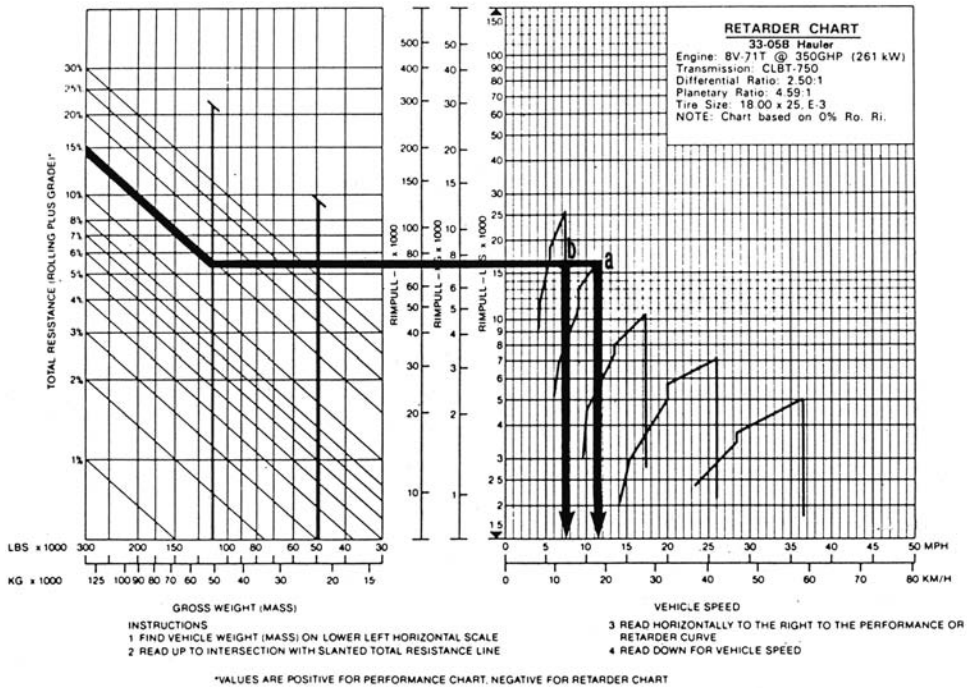


Figure 12.21. Example application of the retarder curve. Terex (1981).

- experience of driver
- level of traffic.

Table 12.13 presents some maximum recommended values which can be used for estimating.

12.4.5 *Specifications for a modern mechanical drive truck*

The Terex 33-09 truck used to demonstrate the principles in the previous section was modern in the 1980's. To illustrate the changes which have occurred over this period of time and to provide the reader with a chance to practice the concepts learned, the specifications for a modern Terex truck, the Terex TR-100 shown in Figure 12.22, have been included in this section (Terex, 2012).

Table 12.13. Maximum recommended downgrade speeds.

Recommend Downgrade Speeds Grade (%)	Speeds (mph)
0–6	30–35
7–8	21–25
9–10	17–20
11–12	13–16
Over 12	Less than 13 mph

Figure 12.22. Photo of the Terex TR100 truck. Terex (2012b).

Figure 12.23 presents several drawings showing the truck geometry. Table 12.14 provides the weights and payload dimensions. Figures 12.24 and 12.25 provide the respective performance and retardation curves.

At this point in time it is considered worthwhile to demonstrate the use of the performance curve shown in Figure 12.24. Assume that the truck is loaded to its maximum payload of 100 tons. In this case, the gross vehicle weight is 347715 lbs. The vertical line corresponding to this value is located toward the right side of Figure 12.24. Beginning at the top of the graph, one follows the line vertically downward until the appropriate total resistance line is intersected. Assuming that the rolling resistance is 2% and the grade resistance is 8%, the total resistance is 10%. From the intersection point, one moves horizontally until the performance curve for the truck is intersected. In this case, one finds that the truck is in gear 2. Moving vertically downward, it is found that a speed of about 8.8 mph can be maintained on this ramp segment. The interested reader is encouraged to repeat this procedure assuming that the fully loaded truck is travelling down a grade of 8%. Assume a rolling resistance of 2%. The retardation curve shown in Figure 12.25 is used.

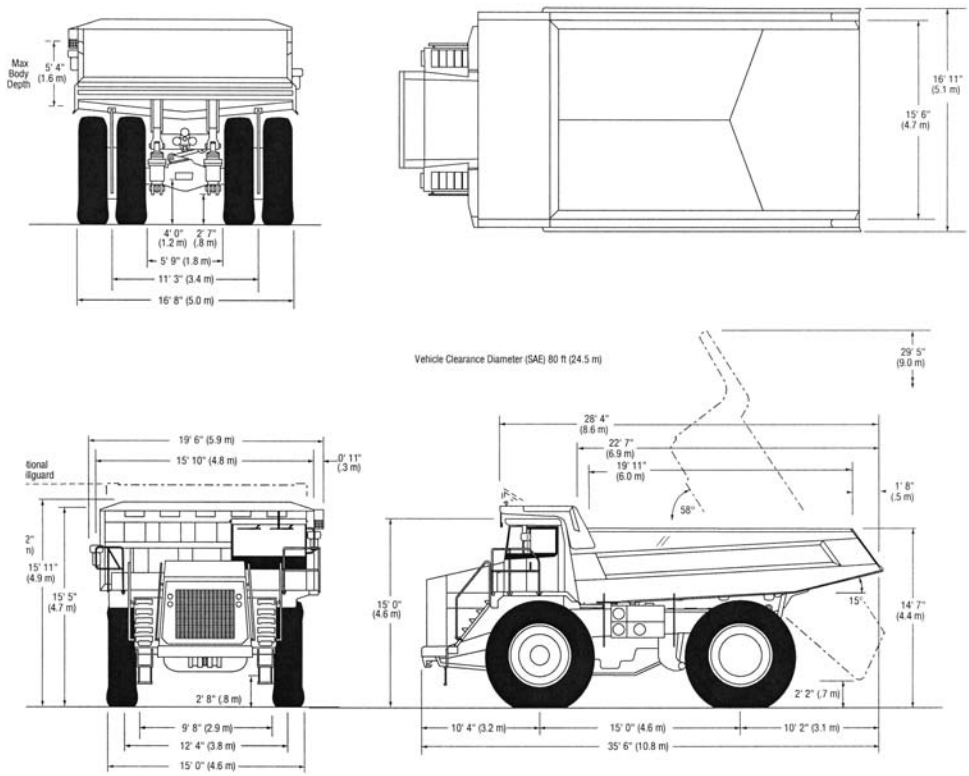


Figure 12.23a. Dimensions of the Terex TR100 truck. Terex (2012b).

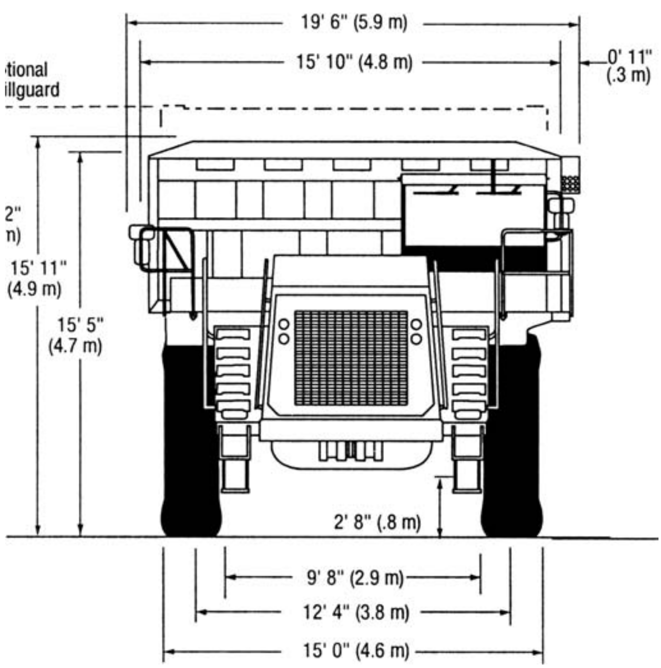


Figure 12.23b. Dimensions of the Terex TR100 truck. Terex (2012b).

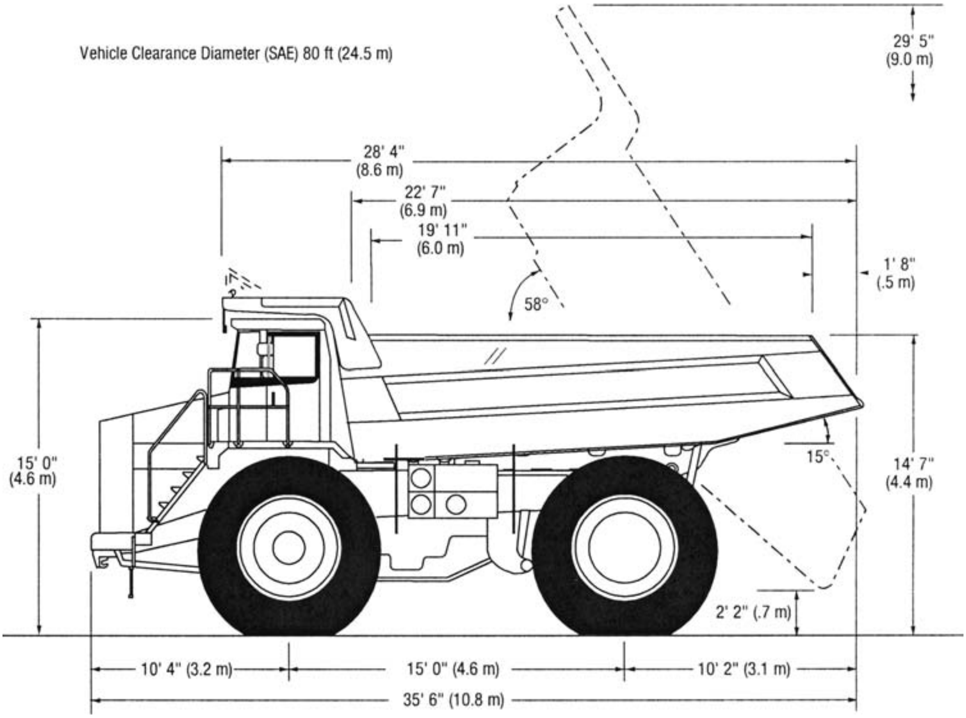


Figure 12.23c. Dimensions of the Terex TR100 truck. Terex (2012b).

Table 12.14. Weight/payload of the Terex TR100 truck. Terex (2012b).

Net weight	159,895 lbs	(72,530 kg)
Payload, maximum	187,810 lbs	(85,190 kg)
Maximum Gross Weight*	347,715 lbs	(157,720 kg)

*Maximum permissible gross vehicle weight with options, attachments, full fuel tank and payload.

Weight Distribution	Front Axle	Rear Axle
Empty	49%	51%
Loaded	34%	66%

Maximum Gross Weight	347715 lbs (112690 kg)
Gross Engine Power	1050 hp (783 kW)
Maximum Payload	100 tons (91 metric tons)
Heaped Capacity	74.5 yd^3 (57.0 m^3)

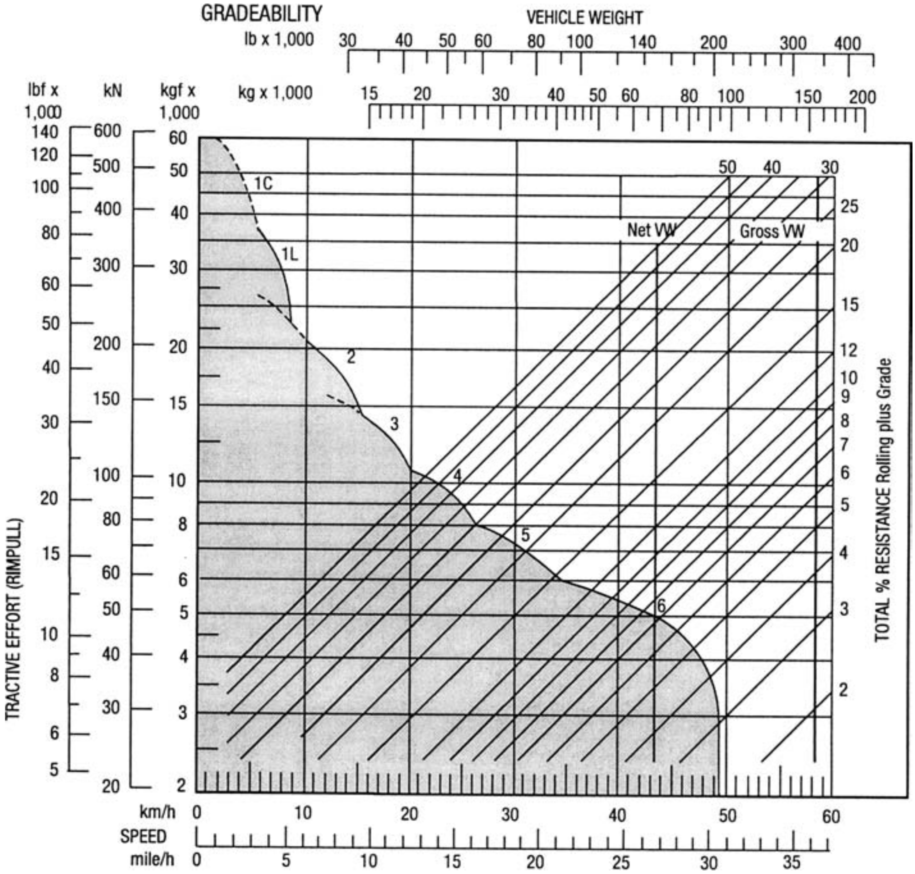


Figure 12.24. Performance curve for the Terex TR100 truck. Terex (2012b).

In this text, the term "performance curve" has been used to denote the curve being used when the total resistance is positive. The term "retardation curve" has been applied to the curve being used when the total resistance is negative. This simplifies the discussion and removes some potential confusion. Taken together the two curves describe the overall "performance" of the truck.

12.4.6 *Braking systems*

Brakes have been the standard way of slowing down and stopping vehicles over the years. They function by forcing a stationary surface against that rotating and a frictional force is developed. Several different types of mechanical braking systems are used. Figure 12.26 is a diagrammatic view of a disk brake system.

For all of the systems, the change in potential energy of the truck is translated into heat energy. This heat must be carried away from the brake area otherwise damage can result making the brakes ineffective. This is often the reason why trucks or even cars sometimes runaway on long steep downhill grades.

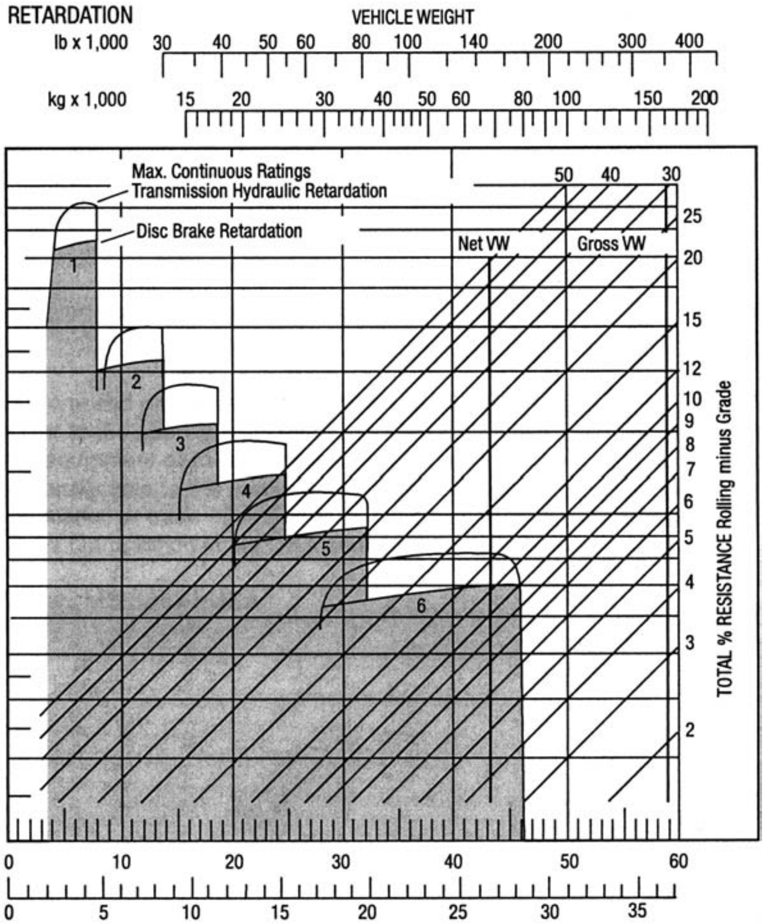


Figure 12.25. Retardation curve for the Terex TR100 truck. Terex (2012b).

Figure 12.26a. Diagrammatic view of a disk braking system. Komatsu (2012)

12.5 PROPELLING THE CONTAINER – ELECTRICAL DRIVE SYSTEMS

12.5.1 *Introduction*

In the way of a brief review, three types of drive train are currently in use (Humphrey and Wagner (2011)):

1. Mechanical drive
2. Electrical drive
   a. Direct-current (DC) electric drive
   b. Alternating-current (AC) electric drive

The mechanical drive train (which was the subject of the prior section) contains five major components (Humphrey and Wagner (2011):

- engine
- torque converter
- transmission
- differential
- planetary gear sets (wheels)

The power source is the diesel engine plus torque converter which transmits rotational power from the engine to the main driveshaft. Machine torque and speed are controlled by the transmission during operation. The differential transfers output torque to the wheels. Finally, the entire system is activated by means of a variety of electronic control modules and hydraulic control systems.

DC and AC electric drive trains contain six major components (Humphrey and Wagner (2011):

- engine
- generator
- power converter
- wheel motors
- planetary gear sets (wheels)
- retarding grid

The power source is the diesel engine plus generator. The latter converts mechanical power from the engine into electric power. AC current from the generator is then converted into useable form. In a DC-drive truck, a rectifier converts it into DC power. In an AC-drive truck, a rectifier converts it into DC power and inverters convert it back to a controllable version of AC power suitable for managing the amperes, volts and frequencies of the wheel motors in order to create machine speed and torque. The DC or AC wheel motors receive the electric power and feed it mechanically to the planetary gear sets (wheels). The retarding grid – a bank of resistor elements – provides braking force by turning the wheel motors into generators, creating power rather than receiving it. The power is sent through a control cabinet and on to the resistor elements. The resistors impede the flow of the electric power, which causes the wheel motors to slow rotation. Heat generated is cooled by an electric fan (Humphrey and Wagner, 2011).

In considering the advantages and disadvantages of the electrical and mechanical drive systems one finds (Birkhimer, 2011) that electric-drive trucks, typically:

- travel at higher speed on grades ranging from 4% to 10%
- have potentially lower maintenance costs
- offer slightly better fuel economy

Figure 12.26b. Cutaway showing the propulsion system. Komatsu (2012).

- have a smoother operator ride
- offer better retarding capacity to stop the truck

On the other hand, they

- have higher capital cost
- require more specialized technical training and capabilities

In comparison, mechanical drive trucks:

- more effectively travel on steeper grades (greater than 10%)
- have a larger market presence resulting in more knowledge in the field
- have a lower capital cost
- require a lower level of technical specialization
- are lighter weight vehicles

As indicated, within the electric-drive option, there is a choice to be made between direct current (DC) and alternating current (AC). The AC-drive offers advantages over the DC-drive and is becoming the more common choice in today's market (Berkhimer, 2011). The remainder of this section will focus on the AC-drive option.

12.5.2 *Application of the AC-drive option to a large mining truck*

This section is based entirely on material kindly provided by Siemens (2012) and Komatsu (2012). Figure 12.26b presents a cutaway of a large haulage truck showing the components of the AC-drive propulsion system.

Figures 12.27 and 12.28 present different schematic views of the wheel motor assembly.

A block diagram depicting the elements in the AC-drive system is presented in Figure 12.29.

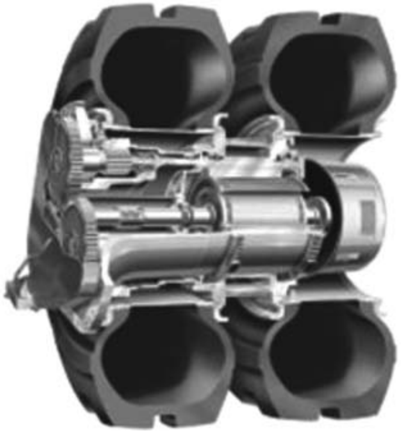


Figure 12.27. Diagrammatic representation of a wheel motor assembly. Komatsu (2012).

Figure 12.28. Diagrammatic representation of a wheel motor. Komatsu (2012).

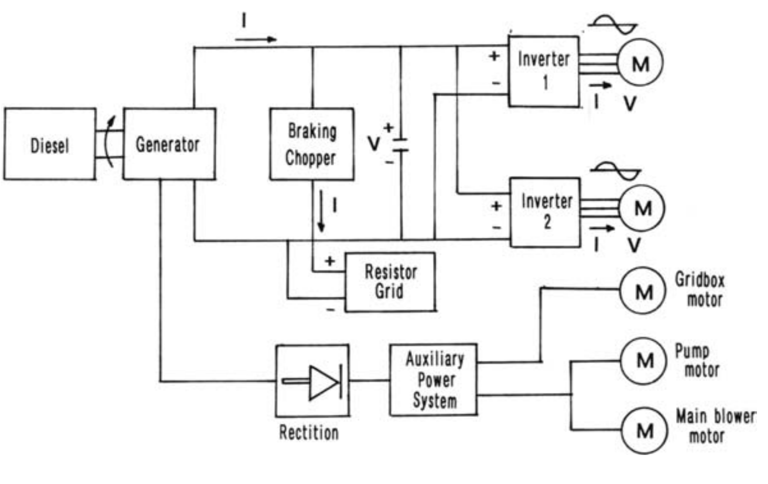


Figure 12.29. Block diagram showing the components of a typical electric drive system. Siemens (2012a).

In general terms, during propulsion, the traction alternator converts the mechanical energy of the diesel engine into DC current, charging the DC bus capacitors. The electric power passes through the capacitors to two inverters, which produce AC current for the traction motors (M). The output frequency and voltage of the inverters is controlled to provide precise motor torque and speed. In the retardation mode, the motors apply braking force by generating electricity which is converted to DC by the inverters. Braking choppers, connected to the inverters, channel that power straight into a resistor grid that continuously dissipates the energy until the truck reaches standstill (the stop position). So braking is smooth, like driving a car, but without mechanical brake wear (Siemens, 2012a).

Each of the components shown in Figure 12.29 will now be briefly described (Siemens, 2012a):

1. The power unit consists of the diesel engine, the alternator, and the rectifier. AC power generated in the alternator is converted to 1,500V DC in the rectifier and flows through the DC link capacitors to the traction inverters.
2. The inverters, motors, and alternator are cooled by a dedicated electric blower that operates independently of vehicle speed or engine rpm. This maximizes cooling performance and system reliability. The inverters, controlled by a specialized control unit, transform DC power at constant voltage into AC power at variable frequency and voltage to drive the truck wheel motors. During braking, the inverters send power from the motors back to the DC link. The inverters allow high switching frequencies, which improves the current quality to the motors. This means the drives require no snubbers (devices to suppress or "snub" voltage transients), and smaller, less complex gate drivers. They have a high overload capability, which enables electronic protection circuits without fuses. This results in greatly increased reliability.
3. Robust, high-torque, squirrel-cage induction motors with integrated speed sensors power the two-stage planetary wheel gears.
4. Maintenance-free electric braking choppers are used in place of mechanical contactors to connect the powerful grid resistors. The retard pedal initiates immediate and smooth braking action. Capable of dissipating up to 6,000 horsepower, the braking system improves control and greatly reduces mechanical brake wear.
5. The auxiliary power system functions both as a rectifier and as a generator.
6. The water cooling system is completely self-contained. The speed and cooling are controlled by numerous sensors. The liquid pump circulates coolant to/from the radiator while the air heat exchanger equalizes the coolant temperature with the cabinet temperature. The high power density, closed-cycle cooling, and standardized modules optimize the system's capability, reliability, and efficiency.
7. The drive system has a very high tractive effort (rimpull) for pulling away in soft ground and fully utilizes all available engine horsepower. Retard utilizes the full power capability of the AC drive system for the highest safe downhill speeds.

12.5.3 *Specifications of a large AC-drive mining truck*

Figure 12.30 shows the Komatsu 860E electric drive truck. The truck is able to carry a 280 ton payload and weighs 1,001,700 lbs when fully loaded. It is powered by a Komatsu SSDA16V160 engine which delivers 2700 hp. The AC drive is supplied by Siemens.

The specifications for the truck are given in Tables 12.15–12.17 and in Figures 12.31–12.33.

Figure 12.30. Photo of the Komatsu 860E truck. Komatsu (2009a).

Table 12.15. The weight/payload of the Komatsu 860E truck. Komatsu (2009a).

Vehicle Weights		
Standard Chassis	137325 kg	302,749 lbs
Komatsu Body	33643 kg	74,171 lbs
Standard Tile Weight	23033 kg	50,780 lbs
Option Allowance	6350 kg	14,000 lbs
Empty Vehicle Weight	200351 kg	441,700 lbs
Front Axle (49%)	98361 kg	216,850 lbs
Rear Axle (51%)	101990 kg	224,850 lbs
Max. Gross Vehicle Weight	454363 kg	1,001,700 lbs
Front Axle (33.5%)	152392 kg	335,871 lbs
Rear Axle (66.5%)	301971 kg	665,829 lbs
Nominal Payload	254363 kg	560,000 lbs
	254 metric tons	280 short tons

Nominal payload is defined by Komatsu America Corp's payload policy documentation. In general, the nominal payload must be adjusted for the specific vehicle configuration and site application. The figures above are provided for basic product description purposes. Please contact your Komatsu distributor for specific application requirements.

12.5.4 *Calculation of truck travel time*

Figure 12.34 shows the path to be taken by the Komatsu 860E truck in going from the shovel to the dump and return.

The characteristics of the segments are given in Table 12.18.

Table 12.16. Specification sheet (1) for the Komatsu 860E truck. Komatsu (2009a).

Make and model		Komatsu SSDA16V160 Tier 2
Fuel		Diesel
Number of cylinders		16
Operating cycle		4 cycle
Gross horsepower* @ 1900 rpm	2,700 HP	2014 kW
Net flywheel power** @ 1900 rpm	2,550 HP	1902 kW
Weight (wet)	21,182 lb	9608 kg

*Gross horsepower is the output of the engine as installed in this machine, at governed rpm and with engine manufacturer's approved fuel setting. Accessory losses included are water pump, fuel pump and oil pump.
**Net flywheel power is the rated power at the engine flywheel minus the average accessory losses. Accessories include fan and charging alternator. Rating(s) represent net engine performance in accordance with SAE J1349 conditions.

Table 12.17. Specification sheet (2) for the Komatsu 860E truck. Komatsu (2009a).

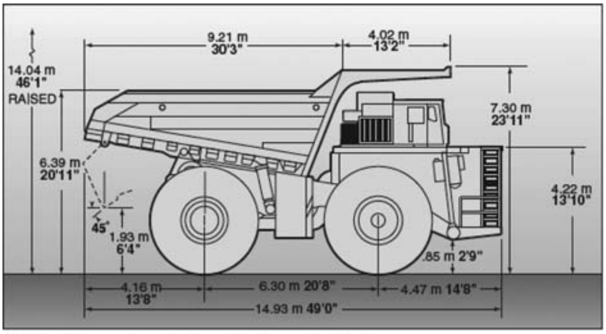
Dimensions	
Box Dimensions	30 ft. 3 in. × 27 ft. 4 in. (9.21 × 8.33 m)
Overall Length	49 ft. (14.93 m)
Overall Height	23 ft. 11 in. (7.3 m)
Overall Width	30 ft. 10 in. (9.39 m)
Wheelbase	20 ft. 8 in. (6.3 m)
Weight	Empty: 4441,700 lbs. (200,351 kg)
Engine & Drivetrain	
Engine Make	Komatsu
Engine Model	SSDA16V160
Engine Type	4-cycle
Horsepower	2,700 hp (2,014 kW) @ 1,900 rpm
Fuel Type	Diesel
Tank Capacity	1,200 gal. (4,542 l)
Operational	
Brakes	Service brakes, emergency brakes, wheel brake locks, parking brakes, electric dynamic retarder
Suspension	Variable rate hydro-pneumatic with integral rebound control.
Tires	50/80 R57
Steering	Accumlator-assisted twin cylinders provide constant rate steering.
Dimensions	
Payload	560,000 lbs. (254,363 kg)
Operational	
Number of Rear Axles	1

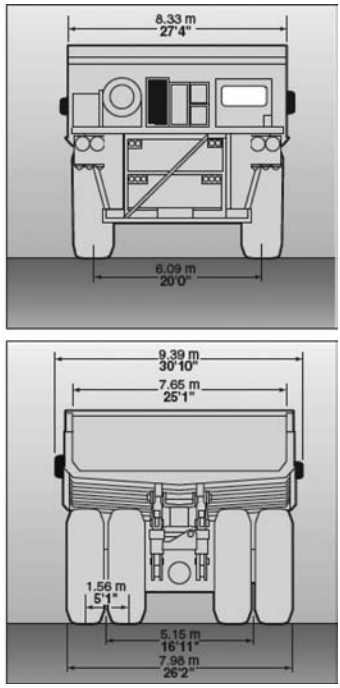
Assuming the truck is loaded to its rated capacity and using the Komatsu Travel Time Simulator, it is found that (Komatsu, 2012a):

$$\text{loaded haul} = 12.4 \text{ minutes}$$

$$\text{empty return} = 3 \text{ minutes}$$

In running this example, no speed limits have been imposed. The interested reader is encouraged to try and duplicate these values using the two manual approaches described in section 12.7.





All dimensions are for unladen truck with standard body.

Body	Capacity		Loading Height*
	Struck	2:1 Heap	
Standard	122 m³ **160 yd³**	169 m³ **221 yd³**	6.37 m **20'11"**

*Exact load height may vary due to tire make, type, and inflation pressure.

Figure 12.31. Dimensions of the Komatsu 860E truck. Komatsu (2009a).

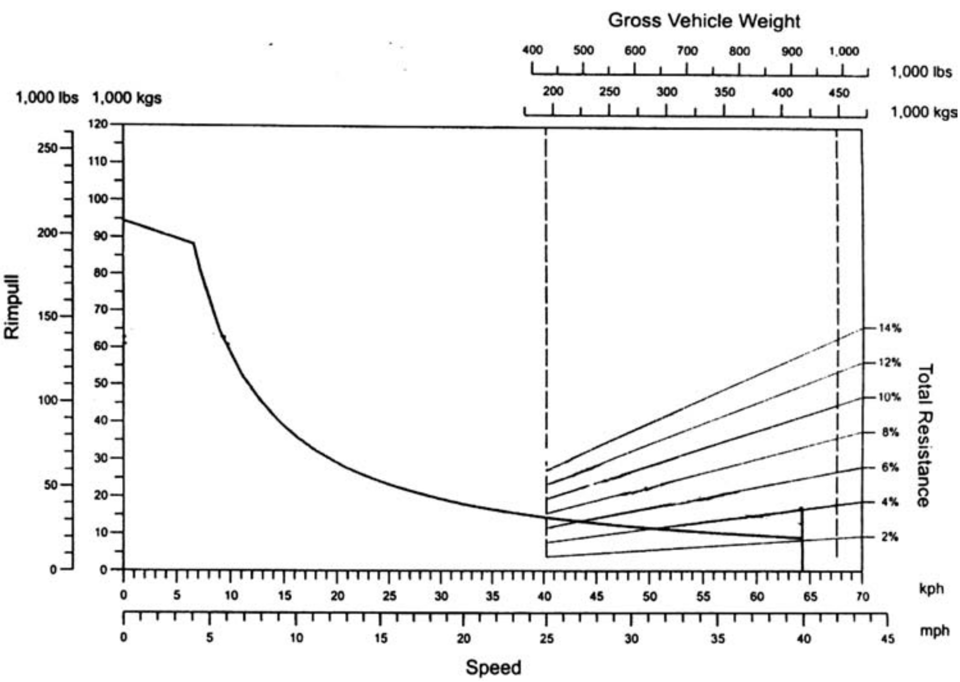


Figure 12.32. Performance curve for the Komatsu 860E truck. Komatsu (2012).

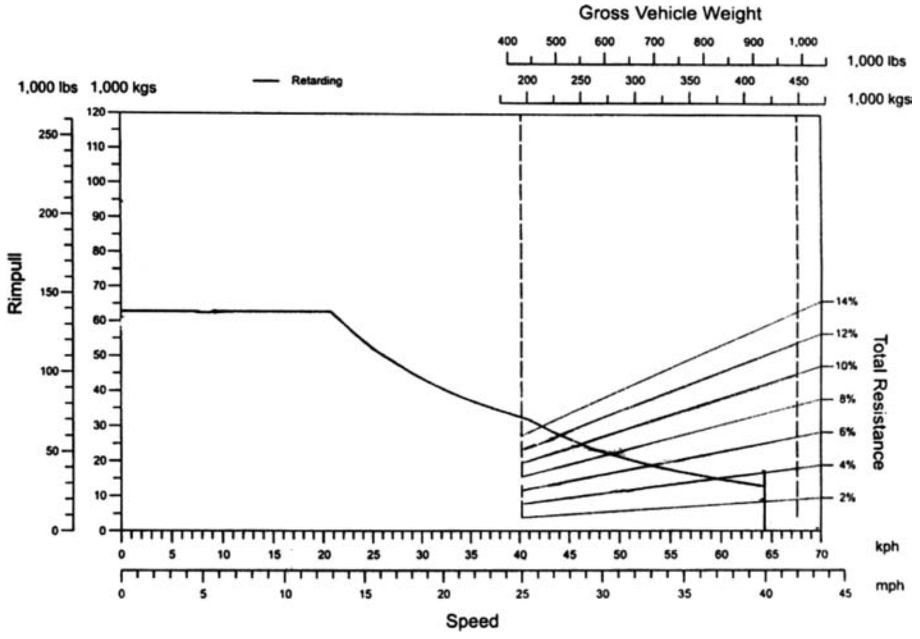


Figure 12.33. Retardation curve for the Komatsu 860E truck. Komatsu (2012).

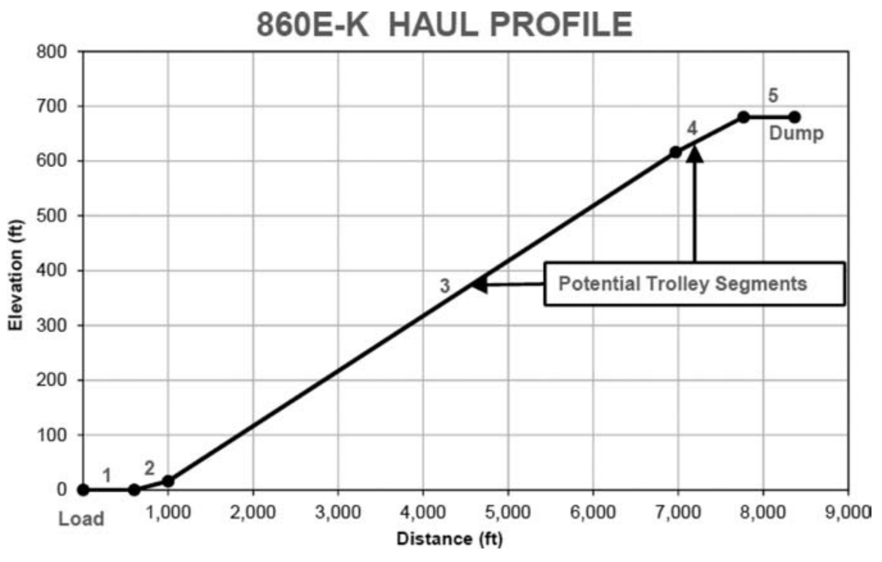


Figure 12.34. Haul profile used by the Komatsu 860E-K truck. Komatsu (2012a).

Table 12.18. Segment characteristics assumed. Komatsu (2012a).

Segment	Length (ft)	Grade	Rolling Resistance
1	600	0%	2%
2	400	4%	2%
3*	6000	10%	2%
4*	800	8%	2%
5	600	0%	2%

12.6 PROPELLING THE CONTAINER – TROLLEY ASSIST

12.6.1 *Introduction*

According to Humphrey and Wagner (2011),

"A trolley-assist mining truck is a unique application for mining trucks and strictly exclusive to electric-drive models. A trolley-assist mining truck draws its power from overhead power lines that are run on haul segments where the largest benefits can accrue, such as where the loaded truck operates on a positive grade. The truck is fitted with a pantograph that acts as a conduit between the line and the truck's electric-drive distribution system. As the truck approaches the line, the operator lifts the pantograph until it contacts the line. When the two engage, the operator removes his or her foot from the throttle and continues to steer while the truck draws power from the line. Power is fed to the wheel motors, temporarily replacing the diesel engine and generator. Trolley assist has several benefits:

- Decreased fuel consumption, achieved by running the engine at idle for the length of the line. Depending on the length of the ramp, fuel savings can be as high as 50%.
- Increased productivity per cycle, achieved by using the excess power capacity of the wheel motors. The power rating of a wheel motor is almost twice the engine gross horsepower, in order to meet the technical requirements for continuous operation under diesel power. That is the primary reason for the significant speed-on-grade performance in retarding, when the wheel motors use their full potential. The same principle applies during trolley assist, when power from the diesel engine is replaced by power from the overhead line. The result is an increase of up to 80% in speed-on-grade performance. Depending on the haul cycle, this can translate into an increase in production of up to 10%.
- Increased diesel engine lifetime. The heaviest toll on an engine in a haul cycle normally occurs when the truck is fully loaded on a grade – the very point at which trolley assist kicks in. The life of a diesel engine is calculated in terms of the total quantity of fuel consumed during the design life. With the engine operating at idle on grade rather than at maximum, the life can be extended significantly, potentially eliminating one complete engine rebuild over the life of the truck.

Trolley assist also involves additional operational costs and constraints, including the following:

- Relative costs of electric power and diesel. This is one of the single largest variables to consider when evaluating trolley assist. The cost for diesel fuel can be enormous for

Figure 12.35. Trolley-equipped 860E truck. Komatsu (2008).

a medium to large fleet, but the ultimate question is whether the savings in fuel can offset the cost of electric power.

- Capital cost of trolley wayside equipment. This consists of mine power distribution, substations, masts, and wire.
- Capital cost of truck trolley equipment. This consists of a pantograph, auxiliary cooling, and truck controls.
- Mine plan. Trolley assist does not allow for operational flexibility. After the equipment is in place, it typically is not moved until doing so makes economic sense, often 5 to 10 years from the initial installment. Therefore it is critical to evaluate the long-term mine plan and determine whether or not a permanent main haul road is possible.
- Haul profiles. Determining which haul cycle benefits from trolley assist is one of the most critical pieces to the evaluation. A long haul segment with a grade that the truck travels loaded is the best choice.
- Capital cost for additional mine-support equipment. Costs associated with additional motor graders or wheel dozers may need to be included. Haul roads where trolley assist is used must be kept in pristine condition, since spillage and rutting can cause the truck to lose connection with the overhead line.

The industry will continue to support trolley assist. Technology improvements now under consideration include concepts such as auto-control when ascending a grade and regeneration of power when retarding during a return cycle."

12.6.2 *Trolley-equipped Komatsu 860E truck*

Figure 12.35 shows the trolley-equipped Komatsu 860E truck in the plant.

Figure 12.36. Truck operating under trolley power. Komatsu (2009a).

Figure 12.36 shows a trolley-equipped Komatsu 860E truck operating in an iron mine. In this case the trolley line is operated at 1800 volts DC.

12.6.3 *Cycle time calculation for the Komatsu 860E truck with trolley assist*

Figure 12.37 shows the performance curves for the Komatsu 860E truck with and without trolley assist.

In reviewing the path shown in Figure 12.34 it will be assumed that a trolley line has been constructed over segments 3 and 4. Assuming the truck has been loaded to its rated capacity, the haul time using the Komatsu Travel Time Simulator is now determined to be 7.1 minutes compared to a time of 12.4 minutes without trolley assist (Komatsu, 2012a). The interested reader is encouraged to check this calculation using the methods described in the following section.

12.7 CALCULATION OF TRUCK TRAVEL TIME – HAND METHODS

12.7.1 *Introduction*

Figure 12.38 shows a path to be taken by the Terex 33-09 truck in going from the shovel to the dump and back. It is required to determine the time needed for the truck to complete the haul and return portions.

This section presents two hand methods by which this can be done.

1. Equation of Motion Method
2. Speed Factor Method

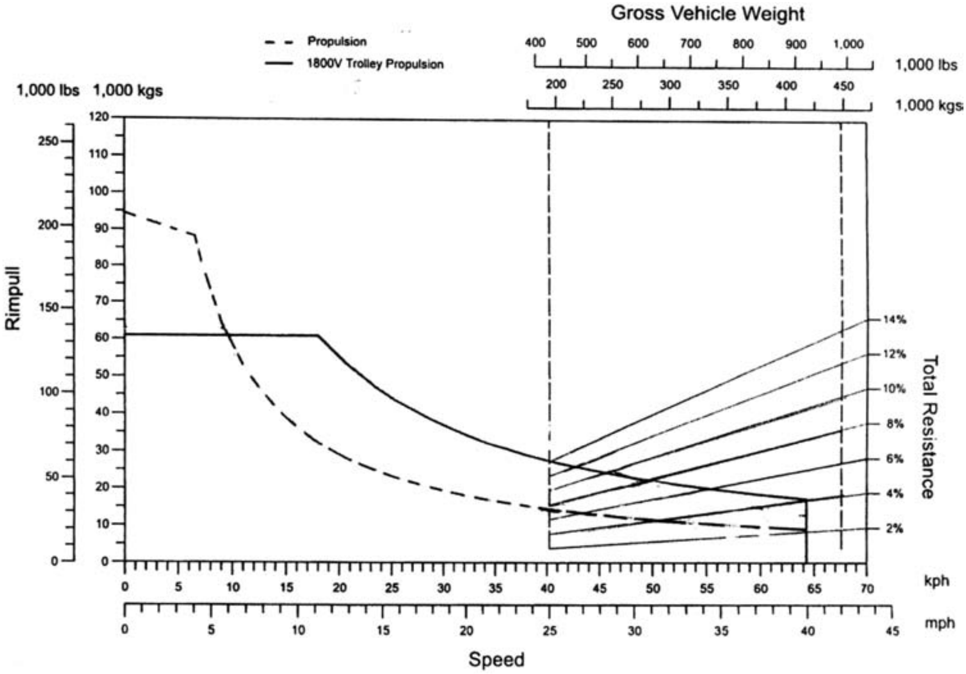
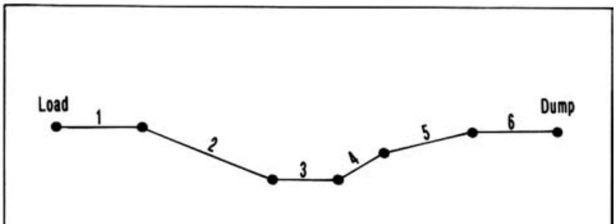


Figure 12.37. Performance curve for the Komatsu 860E truck with and without trolley assist. Komatsu (2012a).



	Segment Number	Length (ft)	Grade Resistance (%)	Rolling Resistance (%)	Total Resistance (%)
Haul	1	1000	0	3	3
	2	1500	-5	3	-2
	3	750	0	5	5
	4	500	8	3	11
	5	1000	3	4	7
	6	1000	0	6	6
Return	6	1000	0	6	6
	5	1000	-3	4	1
	4	500	-8	3	-5
	3	750	0	5	5
	2	1500	5	3	8
	1	1000	0	3	3

Figure 12.38. Path taken by the Terex 33-09 truck.

12.7.2 *Approach 1 – Equation of motion method*

In this section, four examples will be presented to illustrate how the performance and retardation curves can be used to calculate

- the time required to traverse a given road segment
- the exit speed
- the average speed over a segment

In the process of working the examples, the basic formulas will be introduced. With this as background it is hoped that the reader can complete the calculations for the remaining road segments. The examples will deal with the following segments:

Loaded Haul

Segment 1: Truck accelerating from rest

Segment 4: Entrance speed greater than that which can be maintained

Segment 6: Truck coming to a stop

Table 12.19. Summary of the coordinates for the performance curve.

	Velocity (mph)	Rimpull (lbs)
Gear 1	0	64000
	3.0	50000
	4.5	40000
	6.0	28500
	6.01	31500
	7.30	28000
Gear 2	7.31	25000
	8.90	19500
	8.91	21000
	10.6	19000
Gear 3	10.61	17500
	11.70	14500
	11.71	16000
	14.20	14000
Gear 4	14.21	11800
	17.50	10000
	17.51	10600
	21.30	9600
Gear 5	21.31	8200
	23.70	7000
	23.71	7800
	28.50	7000
Gear 6	28.51	6300
	35.40	4900
	35.41	5200
	42.50	4800
	42.41	0

Empty Return

Segment 4: Downhill segment with a speed limit

The loaded and empty weights of the truck are 203,200 lbs and 93,200 lbs respectively. The first step in the process is to have digitized versions of the performance and retardation curves. Table 12.19 gives the major coordinates of the performance curve shown in Figure 12.39.

Table 12.20 provides similar values for the retardation curve (Fig. 12.40).

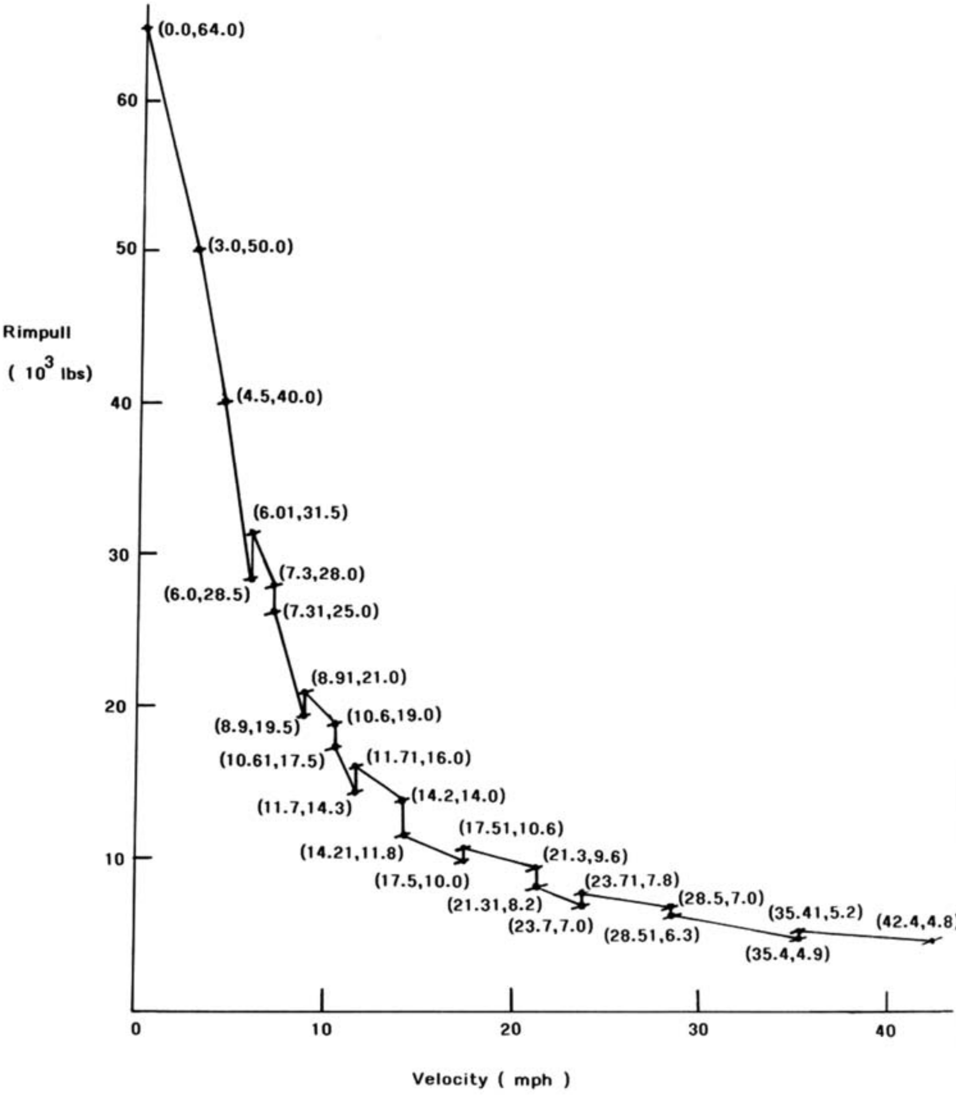


Figure 12.39. Performance curve for the Terex 33-09 truck with the coordinates given.

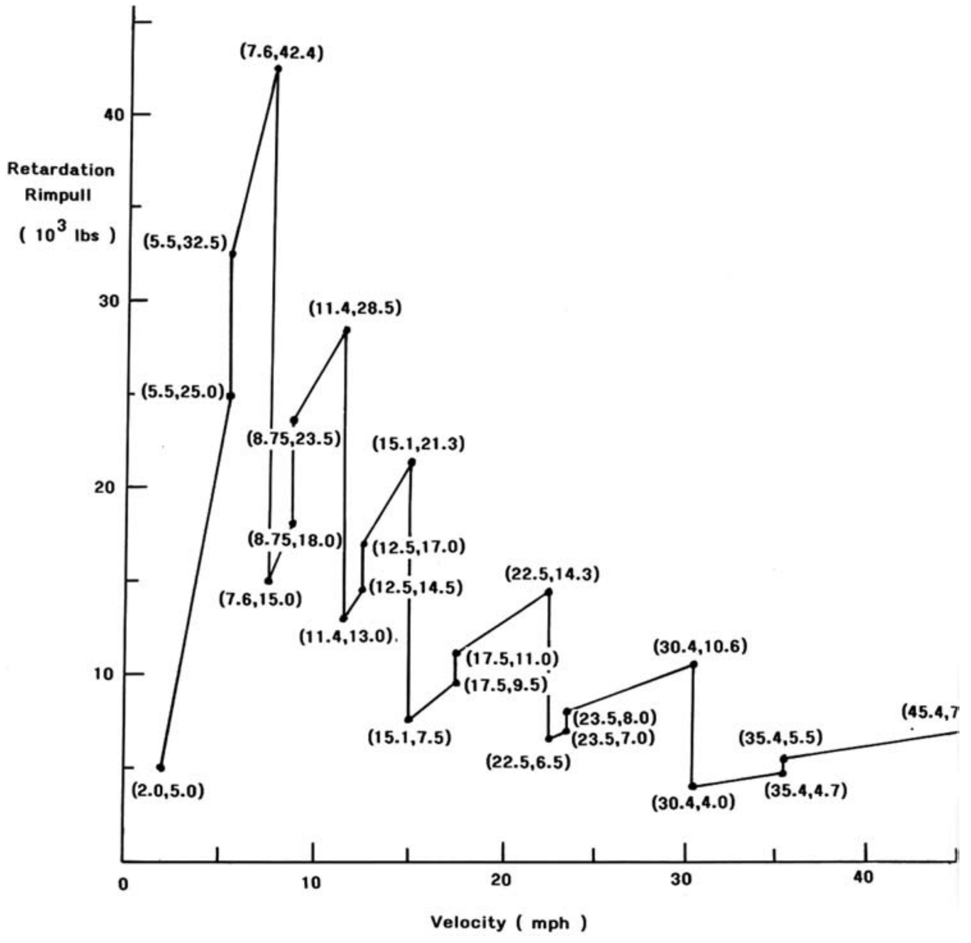


Figure 12.40. Retardation curve with the coordinates given.

Table 12.20. Retarder curve coordinates for the Terex 33-09 truck.

Velocity (mph)	Rimpull (lbs)	Velocity (mph)	Rimpull (lbs)
2	5000	15.10	7500
5.50	25000	17.50	9500
5.50	32500	17.50	11000
7.60	42400	22.50	14300
7.60	15000	22.50	6500
8.75	18000	23.50	7000
8.75	23500	23.50	8000
11.40	28500	30.40	10600
11.40	13000	30.40	4000
12.50	14500	35.40	4700
12.50	17000	35.40	5500
15.10	21300	45.40	7100

Loaded haul calculations

Segment 1. Truck Accelerating from Rest
The length of this segment is 1000 ft, the grade resistance is 0%, the rolling resistance is 3%, and the total resistance is 3%. The tractive effort required to overcome grade and rolling resistance (and to maintain a constant velocity) is

$$TE_{min} = 0.03(203200) \geq 6100\,\text{lbs}$$

For this example, it will be assumed that the coefficient of traction is 0.6. The weight on the rear (drive) tires is 135400 lbs. The maximum rimpull which can be exerted without spinning the wheels is

$$TE\ (\text{max}) = 0.6(135400) = 81240\,\text{lbs}$$

As read from the Performance Curve, the maximum available rimpull is 64,000 lbs, hence this is the controlling value. To simplify the discussion, the performance curve has been divided into a series of regions each denoted by a Roman numeral which will be handled in the forthcoming integration process.

In Figure 12.41, the minimum rimpull line intersects the performance curve at a speed of 29.5 mph. This is the maximum speed which can be achieved by the truck over this segment.

The difference in force between that given by the performance curve and the minimum (6100 lbs) is available to accelerate the truck. In this example, it will be assumed that the available force is fully utilized for acceleration.

Over segment I, (From 0 to 3 mph), the average rimpull is

$$\text{RP (ave)} = \frac{64000 + 50000}{2} = 57000\ \text{lbs}$$

The rimpull available for truck acceleration is

$$\text{RP (acc)} = 57000 - 6100 = 50900\,\text{lbs}$$

Since the mass of the truck is

$$\text{M (truck)} = \frac{\text{W}}{\text{g}} = \frac{203200\ \text{lbs}}{32.2\ \text{ft/sec}^2} = 6311\frac{\text{lbs-sec}^2}{\text{ft}}$$

the average acceleration in this portion of the performance curve is

$$a = \frac{\text{force}}{\text{acceleration}} = \frac{50900}{6311} = 8.07\,\text{ft/sec}^2$$

Knowing that the initial velocity (v_i) is 0 mph and the final velocity (v_f) is 3 mph the time can be calculated using

$$t = \frac{v_f - v_i}{a} \tag{12.19}$$

Thus,

$$t = \frac{(3-0)}{8.07}\left(\frac{5280}{3600}\right) = 0.545\,\text{seconds}$$

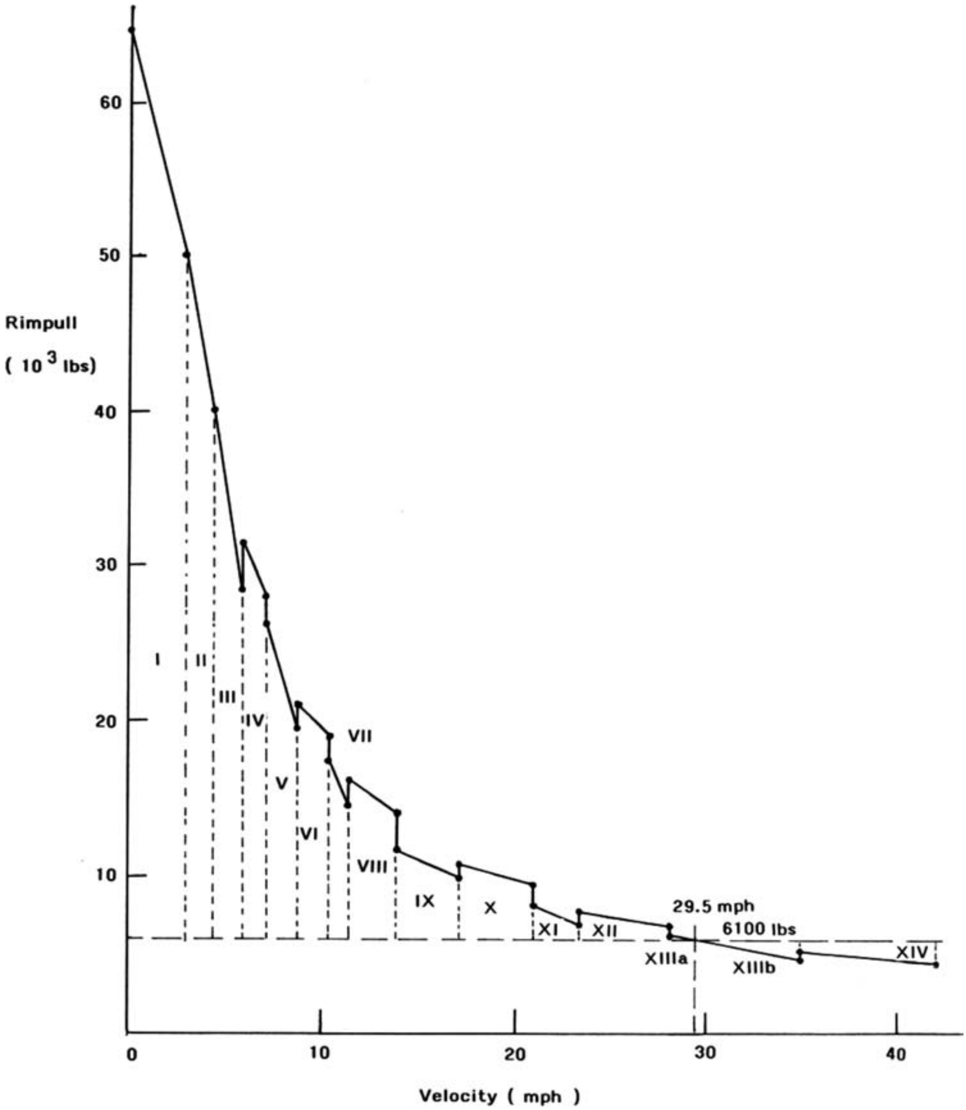


Figure 12.41. The regions involved in accelerating from rest (Segment 1).

The distance which would be traveled in this time can be calculated using either of the following formulas

$$x = \frac{v_f^2 - v_i^2}{2a} \tag{12.20}$$

$$x = \left(\frac{v_f + v_i}{2}\right)t \tag{12.21}$$

Table 12.21. Summary of the hand calculated results for segment 1 based on using the equations of motion

Region	V0 (MPH)	V1 (MPH)	RPnet (lbs)	acc (ft/sec^2)	time (sec)	distance (ft)
I	0	3	50900	8.07	0.55	1.20
II	3	4.5	38900	6.16	0.36	1.96
III	4.5	6	28150	4.46	0.49	3.80
IV	6.01	7.3	23650	3.75	0.50	4.93
V	7.31	8.9	16150	2.56	0.91	10.83
VI	8.91	10.6	13900	2.20	1.13	16.10
VII	10.61	11.7	9900	1.57	1.02	16.67
VIII	11.71	14.2	8900	1.41	2.59	49.20
IX	14.21	17.5	4800	0.76	6.34	147.53
X	17.51	21.3	4000	0.63	8.77	249.61
XI	21.31	23.7	1500	0.24	14.75	486.80
XII	23.71	28.5	1300	0.21	34.11	1305.80

Substituting the appropriate values, one finds that

$$x = \frac{v_f^2 - v_i^2}{2a} = \frac{(3^2 - 0^2)}{2(8.07)}\left(\frac{5280}{3600}\right)^2 = 1.20\,\text{feet}$$

$$x = \left(\frac{v_f + v_i}{2}\right)t = \left(\frac{3+0}{2}\right)\left(\frac{5280}{3600}\right)0.545 = 1.20\,\text{feet}$$

This process is continued until either the maximum velocity of 29.5 mph or the segment length of 1000 ft is reached. Table 12.21 summarizes the results.

At the end of Region XI, the truck has traveled 988.6 ft and hence the travel distance of 1000 ft falls in Region XII. The final velocity would be

$$v_f^2 = v_i^2 + 2ax = 23.7^2 + 2(0.21)(11.4)\left(\frac{3600}{5280}\right)^2$$

$$v_f = 23.75\,\text{mph}$$

The time required to accelerate from 23.71 mph to 23.75 mph is 0.28 seconds and the total time required for the segment would be 37.59 sec. The average velocity would be

$$\bar{v} = \frac{1000\,\text{ft} \times 60\,\text{sec/min}}{37.59\ \text{sec}} = 1596\,\text{ft/min}$$

or

$$\bar{v} = 18.2\,\text{mph}$$

Segment 4. Entrance speed greater than that which can be maintained

This example will cover the situation when the truck has an exit speed of 23.83 mph from segment 3 and encounters segment 4 which is 500 ft in length and the total resistance is 11%. For this segment, the minimum rimpull requirement is 22352 lbs and the velocity which can be maintained is 8.2 mph as read from the performance curve (see Fig. 12.42).

The truck has an entrance speed much higher than can be sustained on such a segment. The momentum of the truck however allows it to travel a considerable distance up the segment before the steady state speed is reached. In this example, the exit speed, time to traverse

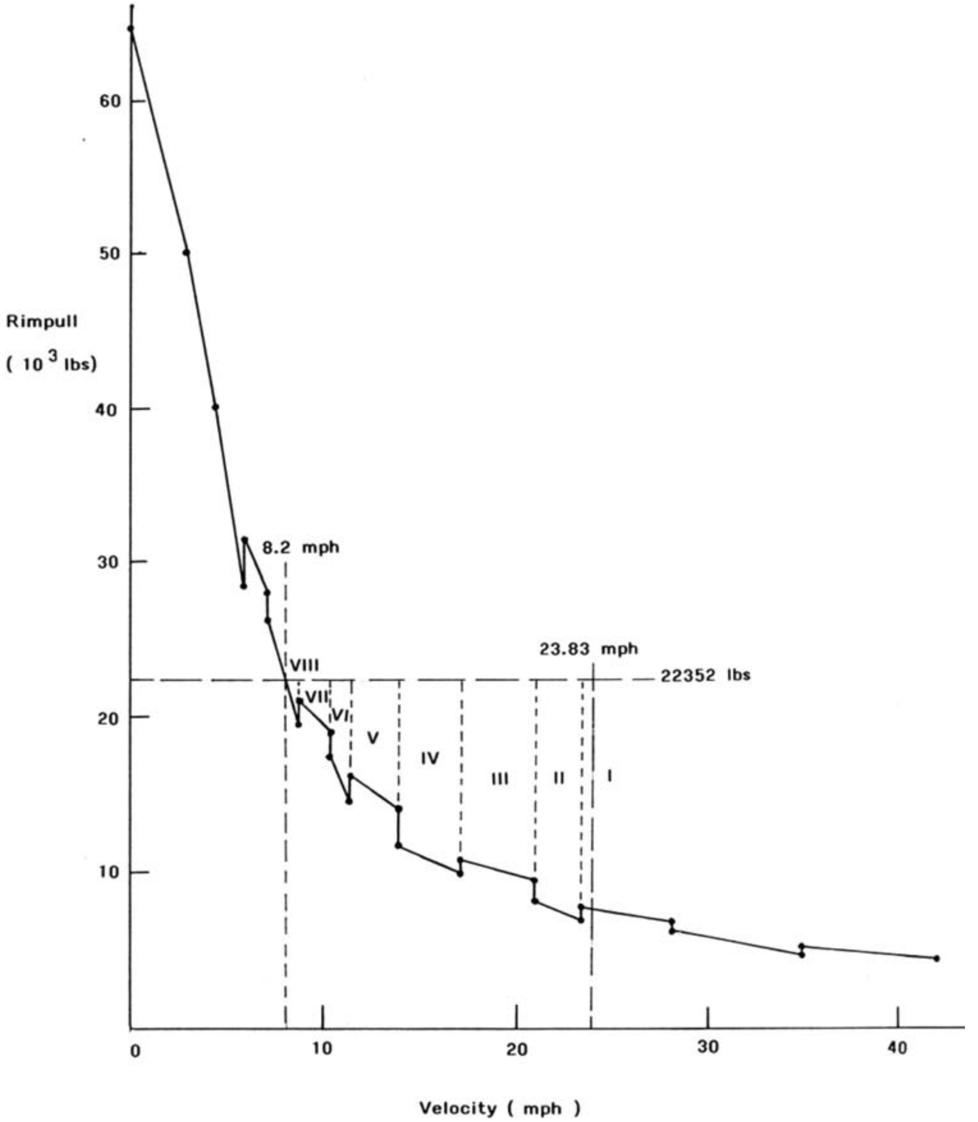


Figure 12.42. The regions involved in Segment 4.

the segment, average speed and the speed factor will be calculated. As can be seen from Figure 12.42, there are eight segments which must be evaluated. Over segment II (from 23.7 to 21.3 mph) the average rimpull force is

$$RP\,(ave) = \left(\frac{8.2 + 7.0}{2}t\right)10^3 = 7600\,\text{lbs}$$

The difference between 7600 lbs and 22352 lbs is the deceleration force on the truck

$$RP\ (\text{deceleration}) = 7600 - 22352 = -14752\,\text{lbs}$$

Table 12.22. Results for Segment 4 using the equations of motion.

Region	Speed Range (mph)	Ave Decel ft/sec^2	Time To Complete (sec)	Dist.	Total Dist.
I	23.83 → 23.7	−2.31	0.76	2.9	3
II	23.7 → 21.3	−2.34	1.51	49.6	53
III	21.3 → 17.5	−1.94	2.88	82.1	135
IV	17.5 → 14.2	−1.77	2.74	63.8	199
V	14.2 → 11.7	−1.16	3.17	60.4	259
VI	11.7 → 10.6	−1.01	1.61	26.4	285
VII	10.6 → 8.9	−0.37	6.70	96.0	381
VIII	8.9 → 8.2	−0.23	4.55	57.0	438

The deceleration is

$$a = -\frac{14752}{6311} = -2.34\ \text{ft/sec}^2$$

and the time required to go from 23.7 mph to 21.3 mph is

$$t = \left(\frac{21.3 - 23.7}{-2.34}\right)\frac{5280}{3600} = 1.51\ \text{sec}$$

The distance traveled in that time is

$$x = \left(\frac{21.3 + 23.7}{2}\right)\left(\frac{5280}{3600}\right)(1.51) = 49.8\ \text{ft}$$

The process is repeated for the other regions in the same way. The results are given in Table 12.22.

Thus only a stretch of 62 ft would be required to be traveled at 8.2 mph which is the exit speed. The time required for this short stretch would be 5.16 seconds and the total time for the segment would be 29.08 seconds. The average speed becomes

$$v_{ave} = \frac{500(60)}{29.08} = 1032\ \text{ft/min}$$
$$v_{ave} = 11.72\ \text{mph}$$

Segment 6. Truck Coming to a Stop

The exit speed from segment 5 is 13.9 mph and as can be read from the performance curve, the rimpull is 14224 lbs. Segment 6 has a length of 1000 ft, a grade resistance of 0% and a rolling resistance of 6%. At the end of segment 6 the truck is stopped. For 6% total resistance, the required tractive effort is

$$\text{TE (req)} = 0.06 \times 203{,}200 = 12{,}192\ \text{lbs}$$

and the maximum velocity (in gear 3) is 14.2 mph.

Therefore the truck can increase its speed from 13.9 mph to 14.2 mph before beginning to stop. Using the equations as before one finds that

F = (14224 − 12192) = 2032 lbs
a = 0.322 ft/sec
t = 1.37 sec
x = 28 ft

Thus in segment 6, there remains 972 ft for travel at 14.2 mph and stopping. The question to be answered is the stopping distance and time required. Three ways:

Case 1: Truck slowing due to rolling resistance only
Case 2: The truck slowed using the retarder and stopped using the service brakes
Case 3: The truck is slowed by following the full retarder curve.

will be examined.

Case 1: Truck slowing due to the rolling resistance only. If the truck is simply allowed to slow down because of the 6% rolling resistance, then

$$\begin{aligned} \text{F (decel)} &= -12192\,\text{lbs} \\ \text{a} &= -1.93\,\text{ft/sec}^2 \\ \text{t} &= \left(\frac{0-14.2}{-1.93}\right)1.47 = 10.82\,\text{sec} \\ \text{x} &= 112\,\text{ft} \end{aligned}$$

Case 2: The truck is slowed using the retarder and stopped with the service brakes.

In this case it will be assumed that the retarder force is maintained such that an average deceleration rate of 15000 ft/min/min (4.17 ft/sec^2) is maintained. Since the rolling resistance provides a deceleration rate of 1.93 ft/sec^2 by itself, the retarder is required to contribute 2.24 ft/sec^2. A minimum retarder force of 14136 lbs would be required. As seen from the retarder curve (Fig. 12.39), the minimum peak value is 21300 lbs. It could be maintained, except for the speed range of 8.75 to 7.5 mph where it drops to 13000 lbs at a speed of 3.5 mph. At this point (or some point earlier) the service brakes would be applied. If the deceleration rate of 4.17 ft/sec^2 is applied from 14.2 to 0 mph then

$$\begin{aligned} \text{t} &= 5.0\,\text{sec} \\ \text{x} &= 52\,\text{ft} \end{aligned}$$

Case 3: The truck is slowed by following the full retarder curve.

In this case, the retarder curve is followed in the same way as was done with the performance curve. The curve represents the maximum retarding situation with the retarder full of oil. The initial speed of 14.2 mph falls within sector I shown on Figure 12.43.

The peak force is 19811 lbs. The average force due to the retarder in going from 14.2 to 12.5 mph is 18406 lbs. This must be added to the retardation due to rolling resistance (12192 lbs) to yield a total retardation force of 30598 lbs. The deceleration, time and distance become

$$\begin{aligned} \text{a (decl)} &= -4.85\,\text{ft/sec}^2 \\ \text{t} &= \left(\frac{12.5-14.2}{-4.85}\right)1.47 = 0.52\,\text{sec} \\ \text{x} &= \left(\frac{12.5^2-14.2^2}{-2(4.85)}\right)2.15 = 10.06\,\text{ft} \end{aligned}$$

The process described above is repeated for each of the other sectors. The results are given in Table 12.23.

For Region VII, the same deceleration as Region VI was used.

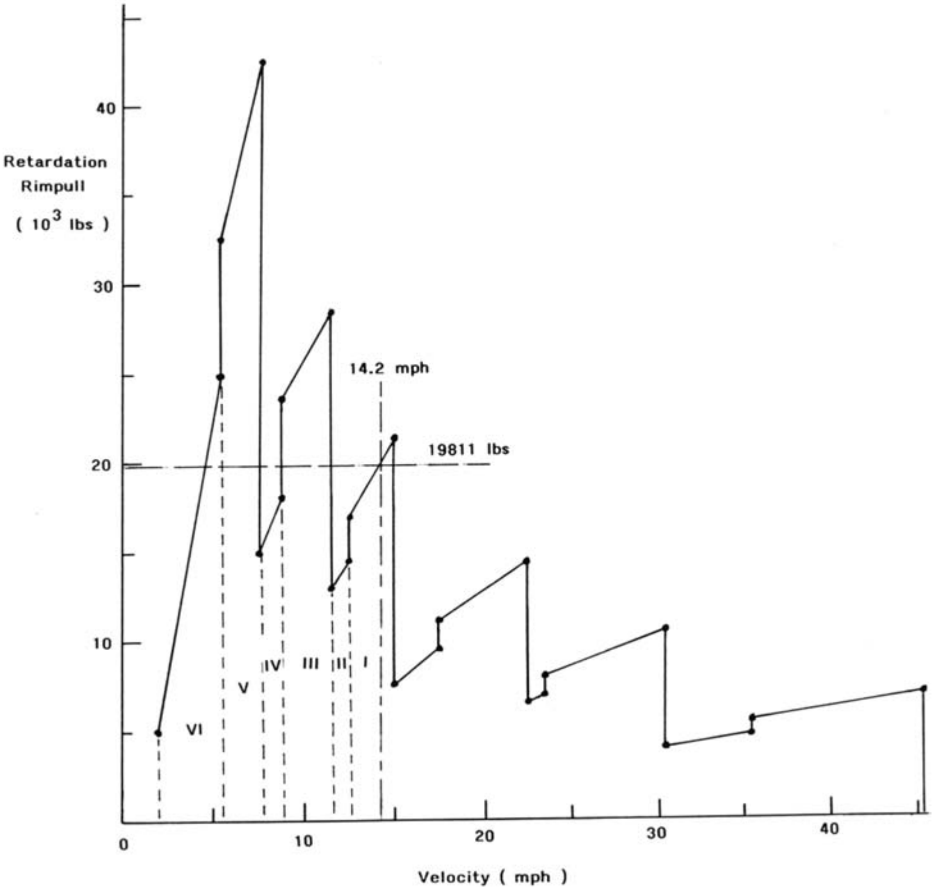


Figure 12.43. Application of the retardation curve.

Table 12.23. Results for Segment 6, truck coming to a stop, using the equations of motion approach.

Region	Velocity Range (mph)	Total Retarding Force (lbs)	Deceleration (ft/sec^2)	Time (sec)	Distance (ft)
I	14.2 → 12.5	30598	−4.85	0.52	10.06
II	12.5 → 11.4	25942	−4.11	0.39	6.88
III	11.4 → 8.75	38192	−6.05	0.64	9.49
IV	8.75 → 7.6	32756	−5.19	0.33	3.89
V	7.6 → 5.5	49642	−7.87	0.39	3.78
VI	5.5 → 2	27192	−4.31	1.19	6.55
VII	2 → 0	27192	−4.31	0.68	1.00

The total distance traveled is 41.65 ft and the time required is 4.14 sec. In this particular case the rate of deceleration varies from 4.11 ft/sec^2 up to 7.87 ft/sec^2. In practice, the operator would try to maintain a smooth retardation action. This is discussed in the following case.

Table 12.24. Summary of the hand-calculated results based on using the equations of motion.

Segment Number	Length (ft)	Imposed Speed Limit (mph)	Entrance Speed (mph)	Exit Speed (mph)	Average Speed (mph)	Time (sec)	Accum. Time (min)
Haul							
1	1000	–	0	23.75	18.2	37.45	0.62
2	1500	–	23.75	30.00	29.7	34.40	1.20
3	750	30	30.00	23.83	26.6	19.17	1.52
4	500	–	23.83	8.2	11.7	29.08	2.00
5	1000	–	8.2	13.9	12.8	53.26	2.89
6	1000	–	13.9	0	13.5	50.54	3.73
Return							
6	1000	–	0	28.9	22.0	31.04	0.52
5	1000	–	28.9	22	34.3	19.90	0.85
4	500	22	22	22	22	15.50	1.11
3	750	–	22	30.44	27.2	18.80	1.42
2	1500	–	30.44	26.9	28.5	35.84	2.02
1	1000	–	26.9	0	25.3	26.90	2.47

The third approach will be adopted here and a deceleration rate of 4.17 ft/sec^2 will be applied to all of the retarder situations in this book.

Since the acceleration from 13.9 to 14.2 mph requires 1.37 sec and consumes 28 ft and the deceleration from 14.2 mph to 0 requires 5 sec and 52 ft, the middle stretch of 920 ft will be traveled at 14.2 mph. This requires 44.17 seconds. The total time for the segment is 50.54 seconds and the average speed is 13.5 mph.

Empty return calculations

Segment 4: Downhill segment with speed limit
On this downhill empty return segment, the speed limit is 22 mph. The total resistance is -5% and the length is 500 ft. The rimpull required to keep the truck from accelerating is

$$RP = 0.05(93200) = 4660\,\text{lbs}$$

Using a segment entrance speed of 22 mph, the retardation rimpull available is 14500 lbs (4th gear). Therefore, no difficulty would be expected to maintain the 22 mph. The time for this segment would be 15.50 seconds.

Table 12.24 presents the results for all of the segments done by hand.

12.7.3 *Approach 2 – Speed factor method*

In the previous section, a hand procedure to calculate truck velocities and travel times using well known force/acceleration/velocity/distance equations and truck performance/retardation curves was described. The results of applying this process for the Terex 33-09 truck for the path shown in Figure 12.38 have been given in Table 12.24. The process is very time consuming and cumbersome. If the route or truck is changed, the entire process would have to be repeated. One alternative to this is the computerization of the equations. Several

Table 12.25. Maximum speeds as read from the performance (P) and retardation curves (R).

Segment	Speeds Read From Performance (P)/Retardation (R) Curves (mph)
Haul	
1	29 (P)
2	42.4 (R)
3	20 (P)
4	8.4 (P)
5	14.0 (P)
6	14.4 (P)
Return	
6	32.5 (P)
5	42.4 (P)
4	30.5 (R)
3	42.4 (P)
2	27.0 (P)
1	42.4 (P)

Table 12.26. Average speed as determined from the hand calculations.

Segment	Average Speed (mph)
Haul	
1	18.2
2	29.7
3	26.6
4	11.7
5	12.8
6	13.5
Return	
6	22.0
5	34.3
4	22.0
3	27.2
2	28.5
1	25.3

such programs are commercially available. Various truck manufacturers also provide cycle calculations as a service to their customers. There is, however, another hand procedure which can be followed yielding useable results rather quickly. This procedure based upon the use of *speed factors* is described in this section. The theory will be illustrated through discussion of the Terex 33-09 truck example.

Knowing the total resistance and the vehicle weight, the maximum speeds which the truck is capable of achieving over a given segment can be read directly from the appropriate performance/retardation curve. Such values for the example are given in Table 12.25.

The *average* speed values over these same segments as determined using the equation of motion approach are given in Table 12.26.

Table 12.27. Calculated speed factors.

Segment Haul	Calculated Speed Factor	Segment Return	Calculated Speed Factor
1	0.61	6	0.69
2	0.98	5	0.81
3	1.33	4	1.00
4	1.44	3	0.65
5	0.92	2	1.05
6	0.94	1	0.60

The ratio of the average speed (Table 12.26) to the performance/retardation values (Table 12.25) can be easily found.

$$SF = \frac{\text{average segment speed/curve (mph)}}{\text{performance/retardation speed (mph)}} \tag{12.22}$$

This ratio (called the speed factor (SF)) is given in Table 12.27.

To obtain the desired segment time, the average speed values (Table 12.26) are applied to the corresponding segment lengths. If, instead of calculating speed factors as a last step in the process, one could read the values from a table, then the step of determining average velocity could be greatly simplified. It would become

$$\text{average velocity} = \text{SF (Table)} \times \text{Performance/Retardation Curve Value} \tag{12.23}$$

Clearly the key to this process is having a general table of speed factors from which one could select the needed values for any combination of the following operating conditions:

- empty and loaded truck situations
- starting from rest
- coming to a stop
- various segment lengths
- level, uphill and downhill segments entered while in motion.

Table 12.28 presents speed factors which can be used for this purpose.

Some explanation dealing with this table will now be presented.

Various truck loading conditions (empty, loaded, partially loaded) are handled through the use of a weight to power ratio (WPR).

$$WPR = \frac{\text{Vehicle weight (lbs)}}{\text{Flywheel (brake) horsepower}} \tag{12.24}$$

Through this calculation, one determines which of the three sections of the table should be used. For loaded and empty hauls one would generally be in different sections of the table.

For the fully loaded Terex Truck

$$\text{Weight} = 203200\,\text{lbs}$$

$$\text{HP} = 624$$

and therefore

$$WPR = \frac{203200}{624} = 326\,\text{lbs/hp}$$

Table 12.28. Published speed factors. Terex (1984).

under 300 lbs/hp Maul Road Length in Feet	Level Haul Unit Starting from 0 MPH	Unit in Motion When Entering Haul Road Section		
		Level	Downhill Grade	Uphill Grade Factor
0–200	0–.40	0–.65	0–.67	1.00
201–400	.40–.51	.65–.70	.67.–72	(Entrance speed greater than
401–600	.51–.56	.70–.75	.72–.77	maximum attainable speed
601–1000	.56–.67	.75–.81	.77–.83	on section)
1001–1500	.67–.75	.81–.88	.83–.90	
1501–2000	.75–.80	.88–.91	.90–.93	
2001–2500	.80–.84	.91–.93	.93.–95	
2501–3500	.84–.87	.93–.95	.95.–97	
3501 & up	.87–.94	.95–	.97–	
300—380 lbs/hp				
0–200	0–.39	0–.62	0–.64	1.00
201–400	.39–.48	.62–.67	.64–.68	(Entrance speed greater
401–600	.48–.54	.67–.70	.68–.74	than maximum attainable
601–1000	.54–.61	.70–.75	.74–.83	speed on section)
1001–1500	.61–.68	.75–.79	.83–.88	
1501–2000	.68–.74	.79–.84	.88–.91	
2001–2500	.74–.78	.84–.87	.91–.93	
2501–3500	.78–.84	.87–.90	.93–.95	
3501 & up	.84–.92	.90–.93	.95–.97	
380 & up lbs/hp				
0–200	0–.33	0–.55	0–.56	1.00
200–400	.33–.41	.55–.58	.56–.64	(Entrance speed
401–600	.41–.46	.58–.65	.64–.70	greater than maximum
601–1000	.46–.53	.65–.75	.70–.78	attainable speed on section)
1001–1500	.53–.59	.75–.77	.78–.84	
1501–20OO	.59–.62	.77–.83	.84–.88	
2001–2500	.62–.65	.83–.86	.88–.90	
2501–3500	.65–.70	.86–.90	.90–.92	
3501 & up	.70–.75	.90–.93	.92–.95	

For the empty Terex Truck

$$\text{WPR} = \frac{93200}{624} = 149\,\text{lbs/hp}$$

The table distinguishes between vehicles which are starting from rest and those already in motion when entering the segment.

For a vehicle accelerating from rest along a level (grade resistance = 0) surface, the first column in the table is used. Assuming WPR = 326, the speed factor for a segment length of 200 ft is 0.39. If the length had been 800 ft instead, interpolation yields a speed factor of 0.575. There is often confusion regarding the correct column to be used when the unit is in motion. The authors have found the following procedure to eliminate much of the difficulty. The total resistance values are examined for each segment. This is illustrated in Table 12.29.

Table 12.29. Resistance values and speed factors for selected segments.

	Resistance			
Segment	Grade	Rolling	Total	Speed Factor Column
1	0	2	2	Level
2	8	2	10	Uphill
3	5	2	7	Downhill

If the grade resistance for the current segment is zero, then the level column is used. When the grade resistance is not zero, then the total resistance (TR) of the previous segment and the current segment are compared. In determining the proper speed factor column for segment 2, one can see that

$$TR(2) > TR(1)$$

and thus the uphill factor applies.

Since the grade resistance of segment 3 is 5%, the truck would continue to gain elevation with distance. Technically it would be labeled as an uphill haul segment. However

$$TR(3) < TR(2)$$

and thus the truck would react as if it were going downhill as compared to the previous segment.

The effective grade change (EGC) defined as

$$EGC = TR(i) - TR(i - 1) \tag{12.25}$$

should be used in deciding which column is appropriate. In this case

$$EGC = -3\%$$

which is downhill compared to the previous segment. Hence the downhill column would be used.

In general, as the length of the segment is increased, the vehicle will spend more time at its maximum speed. This means that for vehicles starting from rest, moving on the level, or traveling on downhill grades, the speed factor tends toward 1.0 with increasing segment length. However for trucks encountering an uphill segment, the speed factor may increase or decrease with segment length. If the vehicle speed on entering the segment is greater than that which can be maintained, then the speed factor decreases to 1.0 with distance. If the entrance speed is less than that which can be maintained on the grade, then the speed factor gradually increases to 1.0. As can be seen, a constant value of 1.0 is used in the chart.

This illustrates an important limitation to the speed factor chart which is that conditions existing in preceding or following segments are not considered. The "momentum effects" are important when entrance speeds are greater than those which can be maintained over a long segment and the segments are short in length.

For a vehicle in motion coming to a stop, the first column is used once again. However in this case the lowest WPR section (under 300 lbs/hp) should be applied irrespective of the true WPR value. This means that for a vehicle having WPR = 326 stopping at the end of a 200 ft long level segment, a speed factor of 0.40 should be used.

Table 12.30. Results of applying the speed factor approach by hand.

Segment	Segment Length (ft)	Speed Limit (mph)	Performance/ Retardation Curve Speed (mph)	Speed Factor Used	Average Speed (mph)	Segment Time (min)
Haul						
1	1000	–	29.0	0.61	17.7	0.64
2	1500	30	30.0 (42.4)	1.0 (0.88)	30.0	0.57
3	750	–	19.0	0.72	13.7	0.62
4	500	–	8.4	1.0	8.4	0.68
5	1000	–	14.0	0.83	11.6	0.98
6	1000	–	14.4	0.67	9.6	1.18
						4.67
Return						
6	1000	–	32.5	0.67	21.8	0.52
5	1000	–	42.4	0.83	35.2	0.32
4	500	22	22.0 (30.5)	1.0 (0.745)	22.0	0.26
3	750	–	42.4	0.77	32.6	0.26
2	1500	–	27.0	1.0	27.0	0.63
1	1000	~	42.4	0.67	28.4	0.40
						2.39

Haul time = 4.67 min
Return time = 2.39 min
Total time = 7.06 min

A complication in this approach is the treatment of speed limits. If the maximum speed which can be achieved over the segment is less than the speed limit, the normal process is applied. If, however, it is greater than the speed limit, a constant velocity equal to the speed limit is used and the speed factor is 1. For the segment preceding the speed limit section, no speed correction (reduction) is made. The results of applying the speed factor approach to the Terex example are given in Table 12.30.

12.8 CALCULATION OF TRUCK TRAVEL TIME – COMPUTER METHODS

Over the years, different computer programs to calculate truck travel times have been developed. In this section, two different approaches will be described:

1. Based on the equation of motion
2. Based on the speed factor approach

The results of applying both methods to the Terex 33-09 example discussed earlier will be presented here.

12.8.1 *Caterpillar haulage simulator*

The results of applying the Caterpillar Fleet Production and Cost Simulator are given in Table 12.31.

As can be seen, the results are similar to those given in Table 12.30 and particularly to those given in 12.24.

Table 12.31. Computer generated output for the Terex 33-09 example.

Course–HAUL SEGMENT
Initial Vehicle Speed = 0.00
Haul Road

Seg No.	Dist (ft)	Roll Res (%)	Grade Res. (%)	Vel Limit (mph)	Max SS Vel (mph)	Top Vel (mph)	Last Vel (mph)	Ac cum Time (mph)
1	1000	3.0	0.00	0.00	29.01	23.97	23.97	0.62
2	1500	3.0	−5.00	30.00	42.41	30.00	30.00	1.20
3	750	5.0	0.00	0.00	19.47	30.00	23.53	1.52
4	500	3.0	8.00	0.00	8.37	23.53	8.94	1.97
5	1000	4.0	3.00	0.00	14.20	14.20	14.20	2.82
6	1000	6.0	0.00	0.00	14.21	14.21	0.00	3.65

Course–RETURN SEGMENT
Initial Vehicle Speed = 0.00
Return Road

Seg No.	Dist (ft)	Roll Res (%)	Grade Res (%)	Vel Limit (mph)	MaxSS Vel (mph)	Top Vel (mph)	Last Vel (mph)	Accum Time (mph)
6	1000	6.0	0.00	0.00	30.55	28.55	28.55	0.52
5	1000	4.0	−3.00	0.00	42.41	41.47	22.0	0.85
4	500	3.0	−8.00	22.00	42.41	22.00	22.00	1.11
3	750	5.0	0.00	0.00	37.95	30.45	30.45	1.43
2	1500	3.0	5.00	0.00	25.97	30.45	26.69	2.04
1	1000	3.0	0.00	0.00	42.40	36.26	0.00	2.46

Haul time = 3.65 min
Return time = 2.46 min

12.8.2 *Speed-factor based simulator*

Loy (1985c) has presented a computer program to calculate truck travel times based upon the use of speed factors. Equations relating speed factor to distance have been developed for each of the weight/horsepower groups and operating conditions given in the speed factor chart. They have the form

$$SF = ad^{b} \tag{12.26}$$

where
a,b = constants
SF = speed factor
d = distance (ft).

Table 12.32 presents the values of a and b which have been used in the program.

The results for the Terex 33-09 truck are given in Table 12.33.

Some differences are observed in the results between Tables 12.30 and 12.33. These are due primarily to momentum effects and the way that speed limits are handled. Adjustments can be made in the starting and end points of the segments to at least partially eliminate this effect.

Table 12.32. Values used by Loy (1985c) in his speed factor-based computer program.

Condition	WPR < 300 lbs/HP		300 ≤ WPR < 380		WPR > 380	
	a	b	a	b	a	b
Start from Stop	0.089	0.292	0.094	0.272	0.082	0.268
Level Segment	0.298	0.146	0.304	0.133	0.193	0.193
Downhill Segment	0.309	0.144	0.287	0.153	0.230	0.177
Uphill Segment	1.0	0.0	1.0	0.0	1.0	0.0

12.9 AUTONOMOUS HAULAGE

The concept of "autonomous" haulage – the use of "driverless" trucks – is not new. According to Shelton (2010)

> "In 1970, Unit Rig started designing a system that would allow a haulage truck, working in an open-pit mine, to operate driverless. Unit Rig teamed up with a Swedish firm, Saab-Scania, to develop and test a driverless truck. The driverless, "The Hands-Off Truck", was shown at the AMC mining show in Las Vegas in 1974. In 1976 Unit Rig operated a fleet of five M-100's, utilizing the driverless system, at Kennecott's Chino Mine near Silver City, New Mexico. An interesting side note was that in order to get the labor union at the Chino mine to agree to the driverless test, Unit Rig had to agree not to sell the system to this mine for ten years. The test was successful and lasted several months and then was disassembled. The system did work but just was not practical with the technology available at that time."

The expression "Hands-Off Truck" was somewhat of an exaggeration. Truck guidance was based on the truck (without driver) following a cable buried in the haul road ramp. However, at the lower end of the cable line, control of the truck was taken over by an operator who moved the truck to the shovel and after loading drove the truck back to the cable line. At the upper end of the line, an operator drove the truck to the dumping point, dumped and returned the truck to the top of the line. It was an important first step.

Zoschke (2001) has summarized some of the important milestones along the way toward automation:

- Early 1970's — Cable – laid systems in haul road
- Early 1980's — Radio Transmitters
- 1980's — Ground beacons, optical fiber gyros
- Early 1990's — GPS operational system (prototype)
- Mid 1990's — Refined GPS development
- 1997 — Integration of a supervisory system
- 1998 — Full cycle demonstration using AT supervisory systems
- 1999, 2000 — Integrated development and operational testing of a "production capable" system
- 2001 — Real autonomous production

According to the Meeting Minutes from the Annual Meeting of the Western Mining Electrical Association (Anonymous, 2001).

Table 12.33. Computer output using the speed factor approach.

SIMULATION PATH = LOADED HAUL FROM SHOVEL TO DUMP
TRUCK UEIGHT = 203200 LBS.

SEGMENT	DISTANCE (FT)	GRADE (%)	ROLLING (%)	TOTAL RES (%)
1	1000	0	3	3
2	1000	−5	3	−2
3	750	0	5	5
4	500	8	3	11
5	1000	3	4	7
6	1000	0	6	6

SEGMENT	SPD-LIMIT (MPH)	SPEED (MPH)	S-FACTOR	SEG-TIME (MIN)	TOT. TIME (MIN)
1	45	30	.615	.615	.615
2	30	30	.878	.647	1.262
3	45	19	.733	.611	1.872
4	45	8	1	.71	2.582
5	45	13	.825	1.059	3.641
6	45	14	.668	1.215	4.656

SIMULATION PATH = RETURN EMPTY FROM DUMP TO SHOVEL
TRUCK WEIGHT = 93200 LBS.

SEGMENT	DISTANCE (FT)	GRADE (%)	ROLLING (%)	TOTAL RES (%)
1	1000	0	6	6
2	1000	−3	4	1
3	500	−6	3	−5
4	750	0	5	5
5	1500	5	3	8
6	1000	0	3	3

SEGMENT	SPD-LIMIT (MPH)	SPEED (MPH)	S-FACTOR	SEG-TIME (MIN)	TOT. TIME (MIN)
1	45	33	.668	.515	.515
2	45	44	.838	.306	.622
3	22	22	.758	.34	1.162
4	45	33	.783	.329	1.49
5	45	27	1	.631	2.121
6	45	43	.668	.395	2.516

"Modular Mining working with Komatsu has developed a very repeatable and reliable system of operating a haul truck fleet sans the operator. Les Zoschke walked us through a pictorial as well as technical overview of the 4 truck fleet they have operating at a mine in Australia. The trucks are provided with GPS as well as inertial guidance systems operating in tandem, providing a guidance system that is so repeatable the trucks always follow the exact same tire track, braking and accelerating at the exact same point. Whole truck monitoring is provided as is radar and other sensory systems which protect workers and other

> equipment which ventures onto the haul road. Modular is so satisfied with the current state of their technology that "babysitter operators" are no longer used on the trucks. Les demonstrated the haul dump characteristics and single backup loading capability. He stated that double backup loading and highwall dumping is something yet in the future. This is a technology still in the making."

Nagai (2010b) of Komatsu stated

> "In 2002, we got the official 'go' in the company for our development plan for AHS (Automated Haulage System), and kicked off the development of AHS for use in large mines. After developing and testing the system for about three years, in December 2005, we began testing a fleet of five 930E-AT driverless dump trucks at CODELCO's Radomiro Tomic mine. In January 2008, we formally delivered 11 driverless dump trucks to CODELCO's Gaby (Gabriela Mistral) copper mine, marking the world's first commercialization of AHS. On December 9, 2008, they held the opening ceremony to which Dr. Michelle Bachelet, then President of Chile, was invited. Under stable operation, 11 units transported 48.5 million tonnes (53.5 million U.S. tons) of material [24.5 million tonnes (27 million U.S. tons) of ore], and contributed to the production of 148,000 tonnes (163,140 U.S. tons) of copper for the fiscal year ended March 31, 2009. When I heard this news, I felt all our hard work had totally paid off. Note: As of June 30, 2010, 12 driverless dump trucks were working there".

The early results from the fleet of driverless trucks at the Radomiro Tomic mine are presented below (Meech, 2012):

- Tonnage hauled 2006 = 8,222,000 t (32,000 tpd for 256 days)
- Mechanical availability = 90.5% AHS vs 80.2% for total fleet
- Effective utilization = 84.2%
- Daily haulage time = $24 \times 0.0905 \times 0.842 = 18.2$ hrs
  - percentage gain = 25.4%
  - potential to 20.5 hrs
- Accidents = none (2 in 2007)
- Cost per tonne = U.S. $ 1.36/t which is much greater than the US $0.58/t normal
- Maintenance reduction = 7%
- Depreciation decrease = 3%
- Impact on mine design:
  - Increased slope angle
  - Decreased road width
- Significant increase in safety

Briefly, the AHS functions in the following way (Anonymous, 2010a):

- AHS is an integrated system to operate super-large driverless dump trucks autonomously by taking advantage of information and communication technologies (ICT) such as the high-precision global positioning system (GPS), obstacle detection sensors, a wireless communication network system and a fleet management system.
- The fleet management system operates driverless dump trucks. Information concerning their hauling routes and speed is sent wirelessly from the fleet management system to the driverless trucks while they travel as they ascertain their position by using GPS information.

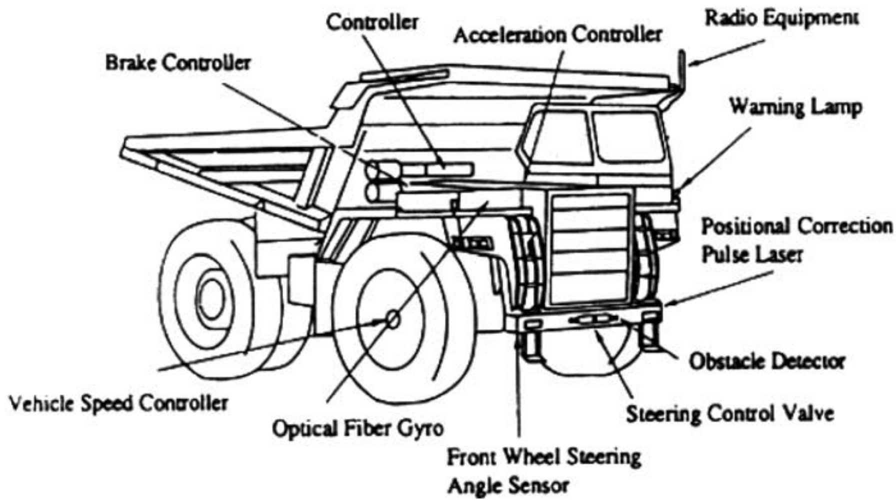


Figure 12.44. Sub-systems of an autonomous haulage truck (AHT). Parreira and Meech (2010).

- For loading, the fleet management system guides the trucks to the loading site, based on the position of the bucket of the GPS-fitted, man-operated hydraulic excavator or wheel loader. After loading, the fleet management system directs the trucks with ore along the route to the dump site for unloading.
- Via GPS and the wireless network, the fleet management system controls all equipment in the mines, including other equipment and vehicles that are man-operated, to prevent collisions of driverless dump trucks. If the built-in obstacle detection sensors detect another vehicle or person inside the haulage area during autonomous operation, the trucks will stop immediately.

Figure 12.44 shows the various sub-systems of an autonomous truck considered in the simulations performed by Parreira et al. (2010).

The results from the AHT simulations and those obtained from simulations of a manually operated truck system (truck drivers working 12-hour shifts on a 2-week-on/2-week-off schedule with all trucks running 24/7) were compared and expressed in the form of Key Performance Indicators (KPI). The results are shown below:

- Investment cost per truck +30%
- Truck haulage cycle times −7%
- Fuel consumption −10%
- Tire wear −12%
- Mechanical Availability +8%
- Increased productivity +5%
- Maintenance costs −14%
- Increased truck life +12%
- Labour costs −5%
- Improved Safety/Reliability to be determined

The authors note that

> "The expected improvement in labor costs is not particularly high. Despite there being no further need for four truck drivers (two on each cross shift cycle) for each truck, additional personnel will be needed to maintain the trucks and their sensor and control sub-systems. These employees will be much more skilled intellectually and thus, will have increased salaries."

Rio Tinto (2012) has recently inaugurated a substantial and vital initiative, "The Mine of the Future (MOF)", to help them find new ways to:

1. Explore better
2. Maximize their resources
3. Go underground – deeper, faster and better
4. Recover more from increasingly difficult deposits
5. Do all the above safely

Overall, it is aimed at finding advanced ways to extract minerals deep within the Earth while reducing environmental impacts and further improving safety. One of the key themes is to achieve a massive efficiency in surface bulk mining through autonomy. One important part of the move to greater autonomy is the introduction of a fleet of new driverless trucks, trains, drills and remote controlled equipment. By this means, productivity and safety are to be improved while enabling the employees to work from a site which is distant from the mine site. It should be noted that the program is not aimed at a single mine site but rather, once the technology has been proven, to be introduced across Rio Tinto. The initial automation program has been established in Rio Tinto's Iron Ore business in Western Australia. Working from their state-of-the-art Operations Center (OC) in Perth, employees will eventually supervise automated drills, loaders and haul trucks across the Pilbara sites.

Grad (2010) has provided the following description of some elements of the program:

1. The West Angelas mine has become a de facto trial site for innovation such as autonomous truck operation, automated drilling, and automated logistics applications. Pit A at the West Angelas mine has been designated as a pioneer site for MOF trials.
2. Under an alliance with Komatsu, Rio Tinto has been testing Komatsu's FrontRunner Autonomous Haulage System (AHS) with driverless trucks. There are presently five Komatsu 930E autonomous dump trucks (see Fig. 12.45) in operation there. Each haul truck, capable of carrying a 290-mt payload, is equipped with radars, lasers, high-precision GPS, an obstacle detection system and a wireless network communications. The trucks are operated and controlled through a supervisory computer at an operations center at the mine. In a few month's time, they will be operated from the OC in Perth.
3. Artificial intelligence onboard the truck "learns" the geography of the mine and "draws" a map of the mine in the truck's memory. Together with onboard sensors such as radars and lasers, this makes it possible for the truck to move quickly, efficiently and safely through the mine. The truck keeps sending the information it acquires to the supervisory center, which in turn processes and analyzes the data, and then sends instructions such as target course and speed to the truck. The truck always moves according to the instructions from the center, which coordinates all activities within the mine. The trucks' top speed is 50 km/h. The fleet control system prevents collisions with other trucks or other equipment. The obstacle detection system will cause vehicles to reduce speed or stop when necessary (see Fig. 12.46).

Figure 12.45. A Komatsu 930E truck equipped in service at West Angelas. Grad (2010a,b).

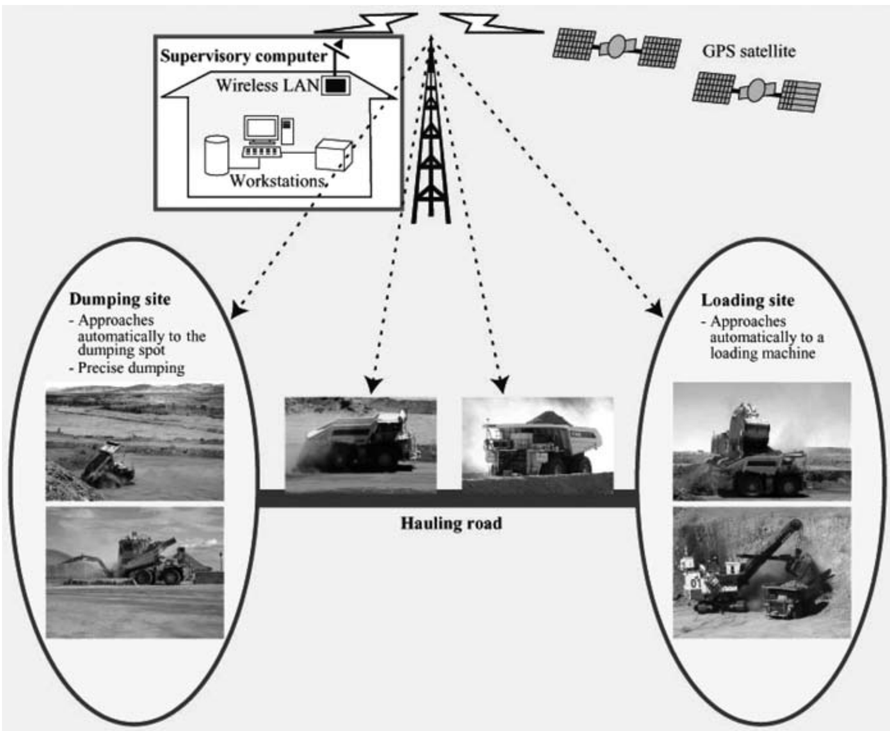


Figure 12.46. Diagrammatic representation of the Komatsu FrontRunner system. Anonymous (2010a).

4. The truck is automatically guided to the loading spot after computing the position of the bucket of the GPS fitted hydraulic excavator or wheel loader. The truck interacts with manned equipment including excavators, graders, wheel dozers, bulldozers and light vehicles. Manned equipment is also fitted with GPS and communications software.
5. An operator interacts with the truck via a screen. The truck "requests" permission to approach or pass. The excavator operator indicates where he wants to load the truck

by raising his bucket and the truck moves to that point autonomously. The task of loading the trucks is still performed by skilled human operators, but this will also see increasing automation in future.

6. A control system divides a manually defined dumping area into dumping nodes and tells the truck exactly where to dump each load. After ore is delivered from the trucks to the crusher, an impact hammer automatically fragments oversized rocks for crushing. This task was already performed remotely by an operator in the OC in Perth late last year.
7. In the future all mining operations will be controlled from an operations center (OC) in Perth, 1500 km away; however, a lot of the mine information will be kept on site, in centralized databases in the Pilbara.
8. When fully realized, the MOF vision – one of the world's largest civilian robotics projects – will see all vehicles and equipment at the mines self-guiding and self-controlling with minimal direct human intervention. Transportation and loading of iron ore on trains will also be fully automated, as will be the trains' travel and downloading of the ore on ships at Dampier and Cape Lambert.

It is obvious that the successful introduction of the ambitious MOF program has widespread implications, not only for Rio Tinto but for all of surface mining.

REFERENCES AND BIBLIOGRAPHY

Aerden, T. 2009. Equipment – Liebherr truck design: Designing monster mining trucks. Mining Magazine. October. pp. 12–15.

Alvarado, V.E. 1992. Trolley-assist to optimize hauling cycles with electric trucks. Proceedings, Surface Mining 2002. SAIMM Symposium Series S30. pp. 207–214.

Anonymous. 2001. Meeting Minutes. Western Mining Electrical Association Annual Meeting, Calgary, Alberta. May 11. www.wmea.net/technical%20Papers/driverless%20trucks%20rev%202.pdf

Anonymous. 2008. Equipment – AC drives: AC in ascendancy. Mining Magazine. March. pp. 26–28.

Anonymous. 2009a. Autonomous haulage system – Komatsu's pioneering technology deployed at Rio Tinto mine in Australia. www.komatsu.com/ce/currenttopics/v09212/index.html.

Anonymous. 2009b. Slow speed object detection for haul trucks. E&MJ. Nov. pp. 42–43.

Anonymous. 2010. Upgrading drives. Mining Magazine. July/August. pp. 29–32.

Anonymous. 2010a. Komatsu autonomous haulage system makes an evolutionary change in mining operations. Komatsu Views. No. 3. Special Edition. pp. 2–3.

Anonymous. 2010b. Behind the scenes of AHS development. Interview with T. Nagai. Komatsu Views. No. 3. Special Edition. p. 4.

Anonymous. 2011. Haul trucks – AC drive trucks: Powering up. Mining Magazine. May. pp. 48–53.

Anonymous. 2011. 'Load and Haul' round table discussion. Mining Magazine. September. pp. 77–101.

Aouad, N., & S. Frimpong. 2011. Research and engineering solution for dump truck vibration problems in high-impact shovel loading operations. Paper 11-102. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Atkinson, T. 1992. Selection and sizing of excavating equipment. In SME Mining Engineering Handbook, 2nd Edition. Vol. 2. (H.L. Hartman, editor). SME. Littleton, CO. pp. 1311–1333.

Awuah-Offei, K., Checkel, D., & Askari-Nasab. 2009. Evaluation of belt conveyor and truck haulage systems in an open pit mine using life cycle assessment. CIM Magazine. August. pp. 59–64.

Bealko, S. 2008. Mining haul truck cab noise: An evaluation of three acoustical environments. Paper 08-002. SME Annual Meeting. Feb 24–27. SLC, UT.

Bealko, S.B. 2009. Mining haul truck cab noise: an evaluation of three acoustical environments. Mining Engineering. 61(10): 36–42. October.

Beebower, A.G., & J.P. Nadzam. 1971. Off-highway haulage vehicle diesel electric drive performance. Society of Automotive Engineers. Automotive Engineering Congress. Detroit, MI. Jan 11–15. Paper 710154. 16 pp.

Berezan, J., Joseph, T.G., & V. de Valle. 2004. Monitoring whole body vibration effects on ultra-class haulers. CIM Magazine. September. pp. 9–12.

Best, R. 2010. Tire maintenance requires an entire team effort. Mining Engineering. 62(2): 32–33. February.

Birkhimer, E. 2011. Chapter 10.4 Selection and Sizing of Excavating, Loading and Hauling Equipment. SME Mining Engineering Handbook 3rd Edition (P. Darling, editor). SME. pp. 931–939.

Blackwell, G.H. 1999. Estimation of large open pit haulage truck requirements. CIM Bull. 92(1028): 143–149.

Bozorgebrahimi, E., Hall, R.A., Blackwell, G., & M. Scoble. 2003. Economies of scale in surface mining: A case study of the impact of haul truck size on maintenance cost. Paper 03-20. SME Annual Meeting. Feb 24–26. Cincinnati, OH.

Brown, G.M., Ebacher, B.J., & W.G. Koellner. 2001. Increased productivity with AC drives for mining excavators and haul trucks. Surface Mining. 53(No. 1 January/March): 45–54.

Burton, A.K. 1975a. Off-highway trucks in the mining industry: Part 1 of a five-part series. Mining Engineering. 27(8): 28–34.

Burton, A.K. 1975b. Off-highway trucks: A guide to engines and transmissions: Part 2 of a five-part series. Mining Engineering. 27(10): 41–47.

Burton, A.K. 1975c. Off-highway trucks: Tires, brakes, and suspensions are important cost-performance components: Part 3 of a five-part series. Mining Engineering. 27(11): 26–32.

Burton, A.K. 1975d. Off-highway trucks: How to calculate truck fleet requirements: Part 4 of a five-part series. Mining Engineering. 27(12): 36–43.

Burton, A.K. 1976. Capital and operating parameters for off-highway trucks: Part 5 of a five-part series. Mining Engineering. 28(1): 38–41.

Butler, J.M., & D.R. Buerschinger. 1972. WABCO's 200 ton truck electro-mechanical drive system. Society of Automtive Engineers. Milwaukee, WI Sept 11–14. Paper 720754. 8 pp.

Carter, R.A. 1999. Haul more, haul faster, haul longer. E&MJ. July.

Carter, R.A. 2007. Diesel developments. E/MJ. 208(3): 78–83. April.

Carter, R.A. 2008. Body of work: Selection of the right dump body for the job requires careful thinking both inside and outside the box. E/MJ. 209(5): 46–52. June.

Carter, R.A. 2008. Diesel engine suppliers map emission-reduction strategies for 2011 and beyond. E/MJ. 209(3): 36–42. April.

Carter, R.A. 2010. Fleet dispatch systems accelerate productivity. E/MJ. 211(1): 30–33. January/February.

Carter, R.A. 2010. Latest IPCC systems provide improved operational flexibility, higher capacity: New project developers as well as mature mine operators are showing renewed interest in in-pit crushing and conveying as a viable alternative to conventional truck-shovel mining. E/MJ. 211(3): 42–45. April.

Carter, R.A. 2010. OEMs upshift as haul truck market picks up speed. E&MJ. Nov. pp. 28–36.

Carter, R.A. 2010. OEMs upshift as haul truck market picks up speed. E/MJ. 211(9): 28–36. November.

Carter, R.A. 2010. Truck wheels: Mine productivity rides on the rim. E/MJ. 211(7): 70–75. September.

Carter, R.A. 2011. Clean machines: Cut operating costs with contamination control. E&MJ. Jan/Feb. pp. 42–47.

Carvajal, E., & J. Johnsen. 2011. Haul truck capacity assurance. Paper 11-116. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Casteel, K. 2008. New focus on in-pit crushing systems. E/MJ. 209(5): 46–52. June.

Caterpillar. 1981. Handbook of Earthmoving. 24 pp.

Caterpillar. 2012. Caterpillar Performance Handbook. Edition 42.

Chadwick, J. 2001. Operating at high altitude. Mining Magazine. Aug. pp. 64–68.

Chadwick, J. 2003. U.S. mining technology. Mining Magazine. Feb. pp. 54–75.

Chadwick, J., & D. Gleeson. 2009. Trucking along. International Mining. June. pp. 16–24.

Cutler, T. 2007. Management in action – Tyres: Rounding up the market. Mining Magazine. August. pp. 16–21.

Czaplicki, J.M., & V.A. Temeng. 1989. Reliability investigation of a shovel-truck system at the Nchanga open-pit mine. Off-Highway Haulage in Surface Mines (S. Golosinski and V. Srajer, editors). A.A. Balkema, Rotterdam. pp. 105–110.

Dessureault, S.D. 2011. Development of a real-time fleet cost tool as part of an integrated remote mine control centre of an integrated remote mine control centre. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30.

Diaz Bórquez, R.A. 2010. Haul truck operator training at the Esperanza mine project. E/MJ. 211(4): 48–51. May.
FiatAllis. 1981. Performance Handbook. 340 pp.
Fiscor, S. 2009. Cat unveils the new F series haul trucks. E/MJ. 210(9): 36–40. November.
Fiscor, S. 2009. Cat unveils the new F Series Haul Trucks. E&MJ. November. pp. 36–40.
Fiscor, S. 2010. Tire makers get a chance to restock and retrain. E/MJ. 211(7): 66–68. September.
Fiscor, S. 2011. Haulage & loading preview. E/MJ. 212(3): 38–45. April.
Fiscor, S. 2011a. Haulage & loading 2011 preview. E&MJ. April. pp. 38-45.
Fiscor, S. 2011b. A new world order for mining OEMs. E&MJ. October. pp. 94–100.
Frimpong, S., & Y. Li. 2007. Cable shovel health and longevity in formation excavation. Paper 07-114. SME Annual Meeting. Feb 25–28. Denver, CO.
Frimpong, S., Y. Li, & S. Agarwal. 2007. Frontier research in dynamic control, vision and collision avoidance for truck haulage. Paper 07-115. SME Annual Meeting. Feb 25–28. Denver, CO.
Fytas, K., & R.K. Singhal. 1987. Computer program matches trucks with shovels. Coal Age. 92(5): 48–53.
General Electric. GE 150AC Drive System for 150-ton Truck Applications. GE Transportation Systems.
General Electric. GE Motorized Wheel for Off-Highway Vehicles. GE Transportation Systems.
General Electric. Typical 787/788 Motorized Wheel for Off-Highway Vehicles. GE Transportation Systems.
Gessel, R.C. 1977. Chapter 5: The Basic Principles of Estimating. Mineral Industry Costs (J.R. Hoskins and W.R. Green, editors). Northwest Mining Association, Spokane. pp. 91–111.
Gleason, W. 2011. Liebherr looks to the future; New technology could lower body weight. Mining Engineering. 63(2): 31–33. February.
Golosinski, T.S. 1989. Trolley assisted off-highway haulage. Off-Highway Haulage in Surface Mines (S. Golosinski and V. Srajer, editors). A.A. Balkema, Rotterdam. pp. 211–215.
Grad, P.S. 2010a. Running with robotics. E&MJ. Jan/Feb. pp. 34–36.
Grad, P.S. 2010b. Running with robotics – Step by step: Rio Tinto's concept of highly autonomous mining is materializing in Western Australia's Pilbara iron ore district. Womp 2010 Vol 01. www.womp-int.com
Grad, R.S. 2010. Running with robotics: Step-by-step Rio Tinto's concept of highly autonomous mining is materializing in Western Australia's Pilbara iron ore district. E/MJ. 211(1): 34–36. January/February.
Gresham, R.B., & R.D. Morey. 1974. Electric-drive trucks at Canadian Johns-Manville's Jeffrey Mine. CIM Bulletin. Feb. pp. 47–50.
Hagenbuch, L. 2002. Off-highway truck body capacity: Why can't I get rated payload on my off-highway trucks without hungry boards and tail extensions? Paper 02-137. SME Annual Meeting. Feb 26–28. Phoenix, AZ.
Hagenbuch, L.G. 2000. Adapting the off-highway truck body volumetric process to real world conditions. Society of Automotive Engineers. Paper 2000-01-2652. 9 pp.
Harcus, M. 2010. Tyres – Radial returns. Mining Magazine. December. pp. 24–28.
Harcus, M. 2011. Vehicle health monitoring – Taking a pulse. Mining Magazine. January/February. pp. 19–23.
Hayes, A. 2011. Haul trucks – Trolley assist: Assisted performance. Mining Magazine. May. pp. 41–46.
Holman, P.J. 1989. Caterpillar mechanical drive off-highway trucks. Off-Highway Haulage in Surface Mines (S. Golosinski and V. Srajer, editors). A.A. Balkema, Rotterdam. pp. 119–130.
Humphrey, J. 2007. All you have ever wanted to know about truck weights – but were afraid to ask. Paper 07-121. SME Annual Meeting. Feb 25–28. Denver, CO.
Humphrey, J.D., & J.D. Wagner. 2011. Mechanical extraction, loading, and hauling. Chapter 10.3 3rd SME Mining Engineering Handbook (P. Darling, editor). SME. pp. 903–929.
International Pay Line Division. 1975. Earthmoving Principles: A Guide to Production and Cost Estimating. 110 pp.
Jackson, M., & L. Zoschke. 2000. Experience to date with unmanned haul trucks in open pit mines. Presented at the 102nd annual meeting of the CIM, Toronto. May 1–5.
Jamasmie, C. 2009. Lonely Trucks in a Lonely Place: Autonomous Trucks Debut in Chile's Desert. Retrieved Feb. 2, 2010 http://magazine.mining.com/Issues/0904/KomatsuAutonomousTrucksDebutChileDesert.pdf.
Johnson, G.D. 1997. Driving force: Trends in AC/DC electric drive systems in shovels, drills, draglines and trucks. World Mining Equipment. May. www.wme.com/wme0597/drive.htm.

Johnson, J.W. 1989. Application of drive systems for mine haulage systems. Off-Highway Haulage in Surface Mines (S. Golosinski and V. Srajer, editors). A.A. Balkema, Rotterdam. pp. 131–137.

Joseph, T.G., & A. Chamanara. 2012. Hauler body payload balance. CIM Journal. 3(1): 24–32.

Kaufman, D.D. 1980. Mechanical versus electrical drive. E&MJ. Aug. pp. 66–73.

Kecojevic, V., & D. Komljenovic. 2010. Haul truck fuel consumption and CO_2 emission under various engine load conditions. Mining Engineering. 62(12): 44–48. December.

Kecojevic, V., & D. Komljenovic. 2011. Haul truck fuel consumption and CO_2 emission under various engine load conditions. Paper 11-002. SME Annual Meeting. Feb 27–Mar 2. Denver, CO.

Komatsu. 2006. Siemens cooperates with Komatsu on drive systems for mining trucks. www.komatsu.com/CompanyInfo/press/2006110214270222514.

Komatsu. 2008. Komatsu to launch 860E super-large dump truck powered by Siemens' AC motor. Press release. Sept. 24.

Komatsu. 2009a. Komatsu, 860E-1K Specification Sheet AUSS791-01. Nov. 16 pp.

Komatsu. 2009b. Specifications and Application Handbook. Edition 30. Dec.

Komatsu. 2010a. Komatsu autonomous haulage system makes an evolutionary change in mining operations. Komatsu Views – Promoting Customer Communication. No. 3. pp. 2–3.

Komatsu. 2010b. Behind the scenes of AHS development. Komatsu Views – Promoting Customer Communication. No. 3. pp. 4–8.

Komatsu. 2012. Personal communication with Komatsu America Corporation staff, in particular, Nathan Priegnitz. June–July.

Lang, R. 2010. Truck body design – Perfect bodies. Mining Magazine. December. pp. 10–13.

Lewis, S.D. 1989. Experience with 200-ton trucks on a 23-mile haul. American Mining Congress Mining Convention. Sept. 17–20. pp. 327–344.

Lovejoy, C. 2009. Equipment – Cat F series: The F series. Mining Magazine. November. pp. 10–13.

Lovejoy, C. 2010. Equipment – Mid-size trucks: Highly flexible. Mining Magazine. April. pp. 46–53.

Loy, M.D. 1985a. Microcomputers in the mining industry – A case study. Preprint 85–76. Presented at the SME-AIME Annual Meeting, New York, NY, Feb. 24–28. 5 pp.

Loy, M.D. 1985b. Determining front-end loader productivity and costs. Coal Mining. July. pp. 46–47.

Loy, M.D. 1985c. Estimating truck cycle time and costs. Coal Mining. Aug. pp. 42–43.

Loy, M.D. 1985d. Using the computer to plan your loader/truck fleet requirements. Coal Mining. Sept. pp. 66–67.

Loy, M.D. 1986. Selection of software for the surface mining industry. Chapter 48, Use of Computers in the Coal Industry (Y.J. Wang, R. L. Grayson, & R.L. Sanford, editors). A.A. Balkema, Rotterdam. pp. 427–432.

MacIntosh, O., & V. Lofftus. 2010. Selection, design and construction of retaining structure to support the primary crusher relocation project at Round Mountain Mine. Mining Engineering. 62(1): 44–48. January.

MacMillan, R.A. 1989. A large electric drive haulage truck – The 240-ton model 830E Haulpak. Off-Highway Haulage in Surface Mines (S. Golosinski and V. Srajer, editors). A.A. Balkema, Rotterdam. pp. 161–169.

Mazumdar, J. 2009. Regeneration energy management in AC mining drives for providing control power. In Proceedings of the IEEE Industry Applications Society Annual Meeting. pp. 1–5.

Mazumdar, J. 2011. The case for trolley assist. E/MJ. 212(9): 30–33. November.

Mazumdar, J. 2011. The case for trolley assist. E&MJ. Nov. pp. 30–33. Or www.womp-int.com/story/2011vol09/story024.htm.

Mazumdar, J., Koellner, W., & A. Holweck. 2007. Design and implementation issues of active front end based systems in mining draglines. In Proceedings of the IEEE Industry Applications Society Annual Meeting. pp. 1760–1765.

Mazumdar, J., Koellner, W., & R. Moghe. 2010. Trolley assist to counter high fuel costs and enhanced operational productivity. In Proceedings of the IEEE Applied Power Electronics Conference and Exhibition (APEC). pp. 1158–1165.

McCaughey, D. 2008. Improved transmission fluids meet needs of today's equipment. Mining Engineering. 60(5): 139–140. May.

Md-Nor, Z.A., Kecojevic, V., Komljenovic, D., & W. Groves. 2008. Risk assessment for haul truck-related fatalities in mining. Mining Engineering. 60(3): 43–49. March.

Mechefske, C.K., & C. Campbell. 2007. Estimating haul truck duty meters using operational data. CIM Magazine. May. pp. 85–89.

Meech, J.A. 2012. Autonomous Haulage Trucks. APSC 150 – Engineering Case Studies. Case Study 4: Sustainable Mining. Part III: Automation inMining. The University of British Columbia. http://mining.ubc.ca/faculty/meech/APSC150/lecture%204.9%20-%20Autononomous%Haulage%20Trucks.pdf.
Moore, P. 2008. Equipment – Rigid trucks. Mining Magazine. July. pp. 18–30.
Moore, P. 2008. Equipment- Trolley-assist trucks: Trolley renaissance. Mining Magazine. August. pp. 16–25.
Moore, P. 2008. Management in action – Tyre initiatives: Tyre taskforces. Mining Magazine. December. pp. 24–26.
Moore, P. 2009. Equipment – Truck bodies: Body beautiful. Mining Magazine. December. pp. 10–17.
Moore, P. 2009. Equipment –Truck update: Haul truck future. Mining Magazine. October. pp. 16–20.
Moore, P. 2009. Management in action – Collision avoidance: Camera ready. Mining Magazine. March. pp. 34–35.
Moore, P. 2009. Management in action – Truck monitoring: Healthy and wise. Mining Magazine. January/February. pp. 20–23.
Moore, P. 2010. Interview – haulage revolution. Mining Magazine. December. pp. 22–23.
Morgan, W.C., & L.L. Peterson. 1968. Determining shovel – truck productivity. Mining Engineering. 20(12): 76–80.
Morrison, D.J. 2007. The Use of Continuous Materials Handling Equipment in an Open Cut Iron Ore Mine, AUSIMM Conference Proceedings, Iron Ore Conference Perth, WA, August 2007 pp. 325–331
Nadzam, J.P. 1971. In growing numbers, electric trucks make their mark at open pit mines. E&MJ. Aug. pp. 55–61.
Naft, M.H. 1977. Integration of component design for a 170 ton off-highway truck. Society of Automotive Engineers. Off-Highway Vehicle Meeting and Exhibition. Milwaukee, Sept. 12–15. Paper 770741. 11 pp.
Nel, S., Kizil, M.S., & P. Knights. 2011. Improving truck-shovel matching. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 381–391.
Neul, M.C. 1966. The effect of weight-to-horsepower ratios on off-highway haulage trucks. Mining Engineering. 18(3): 62–65.
Nienhaus, K., & R. Winkel. 2007. Enabling mining safety & automation by electronically scanning microwave radar systems on surface mining machines. Paper 07-117. SME Annual Meeting. Feb 25–28. Denver, CO.
Nienhaus, K., Hahn, M., Berg, J., & R. Winkel. 2011. Radar-based surface scanning with the world's first fully digital beam forming technology sensor especially developed for mining productivity and safety. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 857–863.
Nieto, A., Sun, E., & ZhongXue Li. 2010. Real-time assisted driving in open pit mining operations using Google Earth. Mining Engineering. 62(2): 21–26. February.
Parreira, J., & J. Meech. 2010. Autonomous vs manual haulage trucks – How mine simulation contributes to future haulage system developments. CIM Meeting, Vancouver, BC. 13 pp.
Phelps, R.W. 1998. Haulage quality – A mine/manufacturer partnership pays off. E&MJ. Dec.
Plaistowe, R.H.A., & M.B. Algeo. 1979. The determination of haulage-truck requirements for an open-pit operation. JSAIMM. Feb. pp. 185–190.
Porter, D. 2005. Opinion – Equipment trends: Caterpillar – is the future electric? Mining Magazine. December. pp. 28–30.
Porter, D. 2008. Equipment – Komatsu 960E: In pursuit of perfection. Mining Magazine. March. pp. 30–35.
Porter, D. 2008. Management in action – Titan tires: A Titan triumph. Mining Magazine. October. pp. 48–51.
Porter, D. 2011. Traffic management – Steering a safe course. Mining Magazine. January/February. pp. 24–29.
Renström, A. 2007. Truck Fleet Utilisation and Fuel Saving in Aitik, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 107–111.
Richards, M. 2009. Slow speed object detection for haul trucks. E/MJ. 210(9): 42–43. November.
Rio Tinto. 2012. The Mine of the Future. www.mineofthefuture.com.au. 14 pp.
Ruff, T., & J. Steele. 2004. Recent advances in proximity warning technology for surface equipment. Paper 04-03. SME Annual Meeting. Feb 23–25. Denver, CO.
Schaum, A., Nieto, A., Scharfrik, S., & M. Karmis. 2008. Haul truck safety and virtual environments training. Mining Engineering. 60(8): 56–60. August.
Sharma, D.K., & L. Ekka. 1989. Shovel-truck productivity in open-pit copper mines. Off-Highway Haulage in Surface Mines (S. Golosinski and V. Srajer, editors). A.A. Balkema, Rotterdam. pp. 111–116.

Shelton, J.A. 2010. The Unit Rig Story. http://utahrails.net/bingham/trucks-unit-rig.php

Siemens. 2008. Pictures of the future: Raw materials – Mining electrification.

Siemens. 2012a. More drive, more productivity: Mining power to the max – with AC drives for haul trucks. www.industry.usa.siemens.com/verticals/us/en/mining/mobile-mining-solutions/documents/brochure_SRMINE_TR_en.pdf.

Siemens. 2012b. Personal communication with Joy Mazumdar. June–July.

Singhal, R.K., & K. Fytas. 1999. Developments in Canadian open pit mining. Paper 99-026. SME Annual Meeting. Mar 1–3. Denver, CO.

Subtil, R.F., Silva, D.M., & J.C. Alves. 2011. A practical approach to truck dispatch for open pit mines. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 765–777.

Terex. 1974a. Terex 33-09 Hauler. Specifications. General Motors. October.

Terex. 1974b. Operator's Handbook for Terex 33-09 Series Haulers. Published by Parts – Service Publications. Terex Corporation. Hudson, Ohio. 74SP10Rev. 2.

Terex. 1978. Production and Cost Estimating of Material With Earthmoving Equipment. General Motors. Form No. 357. 79 pp.

Terex. 1981. Production and Cost Estimating of Material With Earthmoving Equipment. Terex Corporation – A member of the IBH Group. Form 358R. 82 pp.

Terex. 2011. Terex Rigid Truck TR100. Terex Construction. www.terex.com.

Terex. 2012a. Terex Rigid Dump Trucks – Product Range. 27 pp.

Terex. 2012b. Terex Rigid Truck TR100. Specifications. 6 pp.

Terex. 2012c. Personal communication with Jacqueline Anderson. June–July.

Terex Mining. MT 3300 AC Mining Truck – 150 tons (136 mt) Payload Capacity.

Thote, N.R., & R.K. Pandey. 2010. Latest technological developments in mining trucks. MinTECH '10. The Indian Mining & Engineering Journal. pp. 140–142.

Unit Rig. 2011. Unit Rig Mining Trucks Product Line. www.minng.cat.com.

Vallejo, M. 2008. Komatsu Chile achieves milestone delivery of 200th 930E electric drive dump truck. www.komatsu/ce/currenttopics/v08213/index.html.

Walker, S. 2010. Mobile crushers graduate from quarries to mines. E/MJ. 211(9): 38–42. November.

White, L. 1997. Mine haul trucks – Fleet management and performance. E&MJ. 198(2). Feb. pp. 16II–16OO.

Winkel, R., Augustin, C., Nienhaus, K., & M. Hahn. 2011. Productivity enhancement driver assistance for mining machinery based on unique two-dimensional radar sensor fusion concept. Proceedings of the 35th APCOM. Wollongong NSW, Australia. September 24–30. pp. 751–759.

Woof, M. 2001. The biggest load – Payload is King. World Mining Equipment. April. pp. 26–32.

Zoschke, L., & M. Jackson. 2000. Experiences to date with unmanned haul trucks in open pit mines. Annual General Meeting, Canadian Institute of Mining, Metallurgy and Petroleum.

Zoschke, L.T. 2001. Recent experiences with autonomous trucks in open pit mines. PowerPoint presentation. Western Mining Electrical Association Annual Meeting, Calgary, Alberta. May 11. www.wmea.net/technical%20Papers/driverless%20trucks%20rev%202.pdf.

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It must be clearly stated that any misinterpretation of source information and mistakes in understanding are clearly the responsibility of the authors.

REVIEW QUESTIONS AND EXERCISES

1. What is the general rule-of-thumb concerning matching of shovel loaders and haulers?

2. Table 12.1 presents some capacity values extracted from a specification sheet. The truck is designed and built to carry a specific payload (tons). The amount of material which is actually placed in the box depends on a number of factors. In the past the average value for converting volume to weight has been the assumption of 3000 lbs/yd^3. In reviewing the three volume types, which one appears to best correspond to the rule-of-thumb?
3. What is the relevance of the angle of repose values given in Table 12.2?
4. What process would you go through in selecting the correct box to be chosen for the trucks to be used at a particular property?
5. What is meant by "rimpull"? What is meant by "tractive effort"?
6. What is meant by the "rolling radius"?
7. Show the derivation of equation (12.2). Write equation (12.2) for use with SI units.
8. Discuss the transfer of power from the engine to the wheel.
9. What does a torque converter do?
10. What is meant by the term "lock-up"?
11. For the mechanical drives in the 1970's and 1980's, the total power transmission efficiency varied between 75% to 85% depending on whether it was in "lock-up" or not. Using the text material or material you find on the web, expand this brief explanation.
12. The Terex 33-09 Hauler used as a principal example in the text was designed and built by General Motors in the 1970's. Since that time, the brand has gone through a series of owners. Terex trucks are still sold today. Using help from the web, try to trace the Terex brand over the years.
13. What is meant by a "Performance" curve? For what is it used?
14. What is meant by "Gross Horsepower"? What is meant by "Flywheel Horsepower"? What is meant by "Brake Horsepower"? What are the differences, if any, and the reasons for the differences?
15. Show the development of equation (12.3). Show the equivalent form in SI units.
16. Reproduce the theoretical performance curves shown in figures 12.6, 12.8 and 12.9.
17. What is the difference between "available" rimpull and "usable" rimpull?
18. What is the typical "empty" and "loaded" truck weight distribution on the front and drive wheels of a two axle hauler of the Terex 33-09 type?
19. Why is knowing the "coefficient of traction" important? If you have had personal experience in driving under icy conditions, perhaps you can explain the response based on the values in Table 12.9.
20. Discuss the concept of rolling resistance.
21. Sometimes the rolling resistance is expressed in terms of % and sometimes in terms of lbs/ton. If the rolling resistance is 80 lbs/ton, what would be the equivalent value in %? What is the equivalent value of 80 lbs/ton expressed in SI units?
22. What is meant by the "grade resistance"?
23. What is meant by the "total resistance"? Assume that the rolling resistance is 3% and the grade resistance is 8%. What is the total resistance assuming that with respect to the truck motion it is an upward grade? What is the total resistance assuming that the grade resistance is negative? Note: If the total resistance is positive, the performance curve is used to determine the maximum speed. If the total resistance is negative, the retardation curve is used.
24. Retarders are used in mechanical drive systems to slow vehicles. In the cab of the vehicle, there are normally three pedals, the accelerator, the retarder pedal and the brake pedal. When the retarder pedal is depressed explain what physically happens.

25. Explain how the retardation curve shown in Figure 12.17 is applied for both a loaded truck and an empty truck.
26. The performance curve for the Terex TR100 truck is given in Figure 12.24. Compare the shape to the performance curve for the Terex 33-09. What obvious changes have taken place over the intervening 40 years.
27. For the Terex TR100 truck, develop the theoretical rimpull-velocity curve using the approach described in the chapter assuming an efficiency of 100%. Apply an efficiency factor so that your theoretical curve provides a good fit to the actual curve. Conclusion?
28. Compare the retardation curves for the Terex 33-09 and the Terex TR100. Conclusion?
29. Discuss the different types of mechanical braking systems used on large mining trucks.
30. Summarize the components involved in an electric drive system for large haulage trucks.
31. What are the advantages/disadvantages of electrical drive trucks compared to mechanical drive trucks?
32. Discuss how the retardation system works in an electrical drive truck.
33. The performance and retardation curves for a large electric drive truck are given in Figures 12.32 and 12.33, respectively. Compare the curves to those applicable for a mechanical drive truck.
34. Develop the theoretical rimpull-velocity performance curve for the Komatsu 860E truck using the SI unit system. Compare it to the actual performance curve. Conclusion?
35. In Figure 12.32, for a fully loaded truck, what is the developed rimpull and resulting speed assuming a total resistance of 10%. Do the calculation by hand (without the assistance of the nomograph) and then do it using the nomograph. Hopefully, you will get the same answer. If not, try again.
36. Discuss the "trolley-assist" concept. Summarize the advantages/disadvantages. In what places in the world is it currently being used?
37. Repeat the example hand calculation of truck travel time for the Terex 33-09 truck included in your text. It is important that you understand this! It is based on first principles and concepts learned in physics but possibly never applied until now.
38. Repeat the example speed-factor truck travel time calculation for the Terex 33-09 truck included in your text.
39. Apply the hand calculation method to calculate the travel and return times for the Komatsu 860E truck traversing the route shown in Figure 12.34. Compare your answers to those given.
40. Apply the speed factor method to calculate the travel and return times for the Komatsu 860E truck traversing the route shown in Figure 12.34. Compare your answers to those given.
41. Apply the hand calculation method to calculate the travel and return times for the Komatsu 860E truck with trolley assist traversing the route shown in Figure 12.34. Compare your answers to those given.
42. Apply the speed factor method to calculate the travel and return times for the Komatsu 860E truck with trolley assist traversing the route shown in Figure 12.34. Compare your answers to those given.
43. Autonomous haulage seems to be coming of age. Briefly discuss the application in Chile and Australia.

CHAPTER 13

Machine availability and utilization

13.1 INTRODUCTION

Today the basic method of equipment scheduling in the surface mining industry is to determine the equipment required to maintain the desired tonnage and to then schedule only the required equipment to do the job. In the event of a breakdown, production is maintained by the crew picking up another machine on standby or just out of repair. In order to properly schedule and size the overall equipment fleet, knowledge of a number of factors dealing with machine availability and utilization are needed. In this regard, one or more of the following terms are used:

- availability
- operational availability
- mechanical availability
- physical availability
- utilization
- use of availability
- working efficiency
- job efficiency
- operating efficiency
- effective utilization

The calculation of useful hours or minutes is accomplished by applying these factors to various periods of time. Time expressions such as

- scheduled hours
- annual hours
- total hours
- working hours
- shift hours
- operating hours
- efficiency hours

may be used. Unfortunately there is little consistency with which most of these terms are used and their values calculated in the mining industry today. 'Availability' to one mining company may not have the same meaning to another mining company or to a supplier of mining equipment. In this section, the authors have attempted to provide a consistent set of definitions and formulate an overall logical framework for these important concepts.

A number of different information sources have been drawn upon in this effort and some new terms have been applied. It is hoped that confusion will be reduced rather than increased with this approach.

13.2 TIME FLOW

A simple flow sheet showing the distribution of total hours for equipment at a mining operation is shown in Figure 13.1.

The definition of the terms is given in Table 13.1.

The time flow sheet in Figure 13.1 can be represented as a series of nodes (Figure 13.2) at which the flows are split into basically productive and non-productive streams from a production viewpoint.

The ratio of the productive flow out of a node to the total flow into the node can be considered an 'efficiency ratio' or a 'node efficiency'. The total time flow to any given point can then be determined by multiplying the flow in by the efficiencies of the nodes in between.

Node 1:

$$\text{operational availability} = \frac{\text{up-time}}{\text{total time}} \tag{13.1}$$

Node 1*:

$$\text{mechanical availability} = \frac{\text{working hours}}{\text{working hours} + \text{downtime}} \tag{13.2}$$

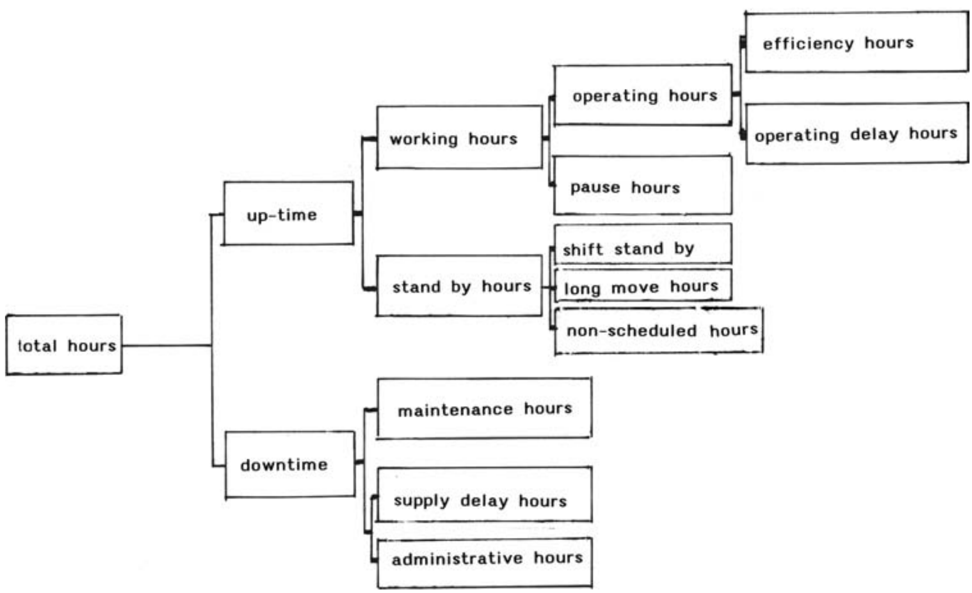

Figure 13.1. Total hour distribution with respect to mining machinery. Source unknown.

Node 2:

$$\text{use of availability} = \frac{\text{working hours}}{\text{up time}} \tag{13.3}$$

Table 13.1. Definition of the terms given in Figure 13.1. Source unknown.

Term	Definition
Total hours	Total of up-time and downtime. This is equal to the scheduled hours for the pit.
Up-time	Time during which machine is able to perform its specified function. It consists of working hours and stand by hours.
Working hours	Time that an operator or crew is assigned to the piece of equipment and it is in workable condition. It is expressed as clocktime and not in terms of the service meter reading. Working hours include all operating delay hours.
Pause hours	That pan of the working hours during which the equipment could be cycling.
Efficiency hours	That portion of the operating time that the machine is in the cycling mode. For a shovel, it would include the time spent in the dig, hoist and swing, dump, and swing and lower actions.
Operating delay hours	This time would include delays such as • short move time • daily maintenance, fuel, lubrication • wait time • time lost due to weather conditions
Stand-by hours	Time during which an operational machine is not used on account of weather, the work schedule, long moves, etc.
Long mote hours	Time during which the machine is operational but scheduled for long moves.
Non-scheduled hours	Portion of the total hours which the operational machine is not scheduled to work.
Downtime	Time during which a machine cannot perform its specified function. It consists of maintenance hours, supply delay hours and administrative hours.
Maintenance hours	Time necessary to carry out preventive and corrective maintenance.
Supply delay hours	Time during which maintenance work is not possible owing to lack of immediate availability of parts and materials necessary to perform maintenance.
Administrative hours	Downtime minus maintenance hours and supply delay hours. 1 Administrative hours represent the time necessary to report machine failure, give work directions for maintenance, etc.
Overtime maintenance hours	Overtime maintenance hours represent maintenance conducted during those hours that the machine would not have been scheduled to work. For a machine scheduled to work day shift, this would describe maintenance performed on another shift.
Daily maintenance hours	Time required to perform daily maintenance. It is included in working hours. Going by the strict definition from the field of reliability engineering, daily maintenance hours should be included in maintenance hours. Here, however, it is included in working hours in consideration of common practice in the mining industry.
Mean time between failure (MTBF)	Average value of operating time between successive failures of a product, device, component or part.
Mean time to repair (MTTR)	Average length of time necessary to repair a failure. It does not include supply delay hours or administrative hours as it assumes that repair is performed in an ideal support environment.

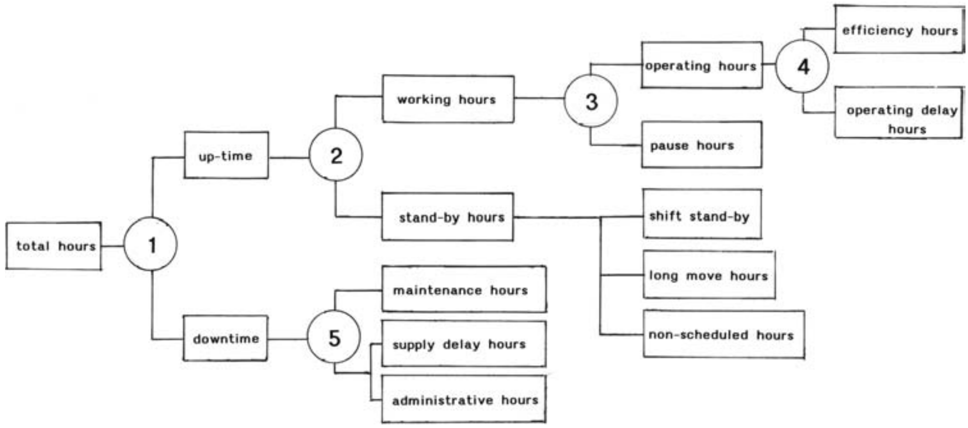

Figure 13.2. Flow sheet showing the distribution of total hours with nodes added.

Node 3:

$$\text{working efficiency} = \frac{\text{operating hours}}{\text{working hours}} \tag{13.4}$$

Node 4:

$$\text{job efficiency} = \frac{\text{efficiency hours}}{\text{operating hours}} \tag{13.5}$$

Node 5:

$$\text{maintenance efficiency} = \frac{\text{maintenance hours}}{\text{down time}} \tag{13.6}$$

13.3 AVAILABILITY – NODE 1

An important factor in equipment scheduling is the availability of the various units. It is defined (Table 13.2) as that proportion of the time during which the machine is capable of performing the required function.

There are three different availabilities:

- Operational Availability (or physical availability)
- Mechanical Availability
- Inherent Availability

which are commonly calculated. Node 1 corresponds to operational availability which is the most commonly used and often abbreviated to just ‘availability’. This is also the most convenient to use in calculations. The more meaningful availability is however mechanical availability which relates working hours to working hours plus downtime hours. Table 13.3 gives typical mechanical availabilities for some mining machines.

If the standby hours equal zero then the operational and mechanical availabilities are equal. Figure 13.3 is a plot of the ratio of the two availabilities and the standby to working hour ratio.

Table 13.2. Definitions and formulas for calculating availability. Source unknown.

Term	Definition
Availability:	The proportion of time during which the machine is capable of performing the required function. Availability includes operational availability, inherent availability, and so on. Remark 1: Often the "availability" Is used to denote operational availability.
Operational availability:	The proportion of time in which a machine is in an operational conditions. It is defined by the following equation: $\text{Operational availability} = \dfrac{\text{Up-time}}{\text{Up-time} + \text{Downtime}}$ Remarks 1: Because this is the most widely used kind of indication, it is often abbreviated to just "availability". 2: This indication does not express the reliability of the machine alone, but rather of the overall system including the support system.
Mechanical availability:	Obtained by subtracting the stand-by hours from the denominator and numerator in the equation defining operational availablity. It is defined by the following equation: $\text{Mechanical availability} = \dfrac{\text{Working hours}}{\text{Working hours} + \text{Downtime}}$ Remarks 1: The influence of stand-by hours (not related to machine reliability) is deducted from operational availability. 2: It is mainly used by the supplier of the machine. 3: The mechanical availability is always lower than the operational availability.
Inherent availability:	An indication which expresses the inherent reliability of the machine. It is defined by the following equation: $\text{Inherent availability} = \dfrac{\text{Mean time between failure (MTBF)}}{\text{Mean time between failure (MTBF)} + \text{Mean time to repair (MTTR)}}$ Remarks 1: Takes into account only corrective maintenance and excludes preventive maintenance. 2: Takes into account only maintenance hours and excludes supply delay hours and administrative hours.

Table 13.3. Typical mechanical availability figures for some mining machines. Source unknown.

Machine	Typical mechanical availability	
	Range	Average
Rotary Drills	0.80 to 0.90	0.85
Electric Shovels	0.70 to 0.80	0.75
Trucks	0.60 to 0.90	0.75

It can be seen that the relationship is not too sensitive to standby time. However, if the operational availability is 5% or more above the mechanical availability (30% standby time), the latter should then be used. The use of true mechanical availability will give a conservative result.

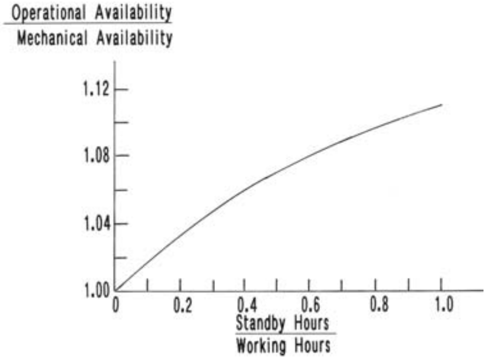

Figure 13.3. Ratio of operational availability to mechanical availability versus the ratio of standby hours to working hours.

Table 13.4. Definition of utilization.

Term	Definition
Utilization (Use of Availability)	The ratio of working hours to the up-time. It is defined by the following equation: $= \dfrac{\text{Working hours}}{\text{Up-time}}$

Although some remarks have been included in Table 13.2, some additional ones are also appropriate.

- Operational availability is basically a historical record of a machine showing what use was made of previous time. If it is considerably higher than mechanical availability, the equipment is not being used to capacity and a thorough study of the operation would be desirable.
- Mines on one- or two-shift and five- or six-day operation could possibly schedule all repair work to off (standby) shifts. This repair time must however be added to that performed on scheduled shift time if true mechanical availability is to be calculated. Failure to do this will produce high value of mechanical availability not reflective of the actual machine. As can be seen from Figure 13.1, standby repair has the effect of transferring "up-time" into "down-time" hours.
- Using working hours and downtime hours one can predict the future performance of a machine with reasonable accuracy.

13.4 UTILIZATION – NODE 2

Availability does not indicate the percentage of the time that a machine is actually being used. This is done through the "utilization" or the "use of availability" ratio. It is described in Table 13.4.

Utilization provides a measure of how efficiently available equipment are utilized in an operation. If the ratios for individual units are low, operational problems such as

- poor machine performance
- machine incapable of task assigned
- machine located in an area not scheduled for use.

might be suggested. On the other hand it might simply reflect

- a need to cover a widespread work area for blending, etc.
- the decision to retain obsolete but useable equipment as backup.

Overly high values could indicate a shortage of equipment or an excess of operators. If it is the former, then equipment breakdown could lead to production shortfalls. If the latter then the resulting labor cost per ton would be higher. Adding equipment or rescheduling may be called for.

13.5 WORKING EFFICIENCY – NODE 3

There are various ways in which non-productive time in the working hours such as lunch and travel time can be included. One method is simply to include these in the operating delay portion of the operating hours. However as will be seen, the delays are not of the same type. The term pause hours is used to denote non-operating time that is fixed for every shift. For example during each shift a lunch break may be included as well as the travel time to and from the parking lot. Although this time could simply be subtracted from the working hours to get operating hours,

$$\text{operating hours} = \text{working hours} - \text{pause hours} \tag{13.7}$$

it is better from several viewpoints to treat it as an efficiency

$$\text{Working efficiency} = \frac{\text{operating hours}}{\text{working hours}} \tag{13.8}$$

The first reason is that it reflects the efficiency changes which occur from changing the shift time. If lunch and travel times are fixed at 30 minutes and 20 minutes, respectively, and the working shift is increased from 8 to 10 hours, the working efficiency increases from 0.90 to 0.92. The second reason is that it simplifies computation.

13.6 JOB EFFICIENCY – NODE 4

The job efficiency factor is used to reflect delays which occur in the normal work day such as

- equipment adjustment
- lubrication, fueling or service
- stops by the operators
- short machine moves
- road maintenance
- traffic
- blasting, etc.

It is defined in Table 13.5.

Table 13.5. Definition of job efficiency and efficiency hour.

Term	Definition
Job Efficiency	An index which Cakes account of the various factors, such as operator's skill, ease or difficulty of work, daily maintenance hours, and so on, which influence production. $\text{Job efficiency} = \dfrac{\text{efficiency hours}}{\text{operating hours}}$

Table 13.6. Efficiency hour values for various mining machines (International, 1975).

Conditions	Crawlers	Front-End Loaders	Elevating Scrapers	Haulers & Open Bowl Scrapers	Excavators Hydraulic
Favorable	55	50	57	50	55
Average	50	45	50	45	50
Unfavorable	45	40	45	40	45

This has given rise to the common use of the term "efficiency hour" which is

$$\text{Efficiency hour (minutes)} = \text{Job Efficiency} \times \text{Operating hour (60 minutes)} \tag{13.9}$$

A job efficiency of 0.83 for example gives rise to the so-called 50 minute hour. The concept of an efficiency hour and how it is calculated is extremely important since it leads to production rate estimates.

Table 13.6 presents values of the efficiency hour that might be assumed under favorable, average, and unfavorable conditions for crawlers, front-end loaders, elevating scrapers, haulers, open bowl scrapers, and hydraulic excavators.

13.7 MAINTENANCE EFFICIENCY – NODE 5

The overall term of "maintenance" can include

- Preventive maintenance
- Corrective maintenance
- Periodic maintenance
- Condition based maintenance
- Daily maintenance

The time for daily maintenance is included under "up-time" hours whereas the others fall under the category of "down time". The maintenance flow sheet is shown in Figure 13.4 and the terms are defined in Table 13.7.

Maintenance efficiency is defined at that proportion of the total down time that is used in maintenance. One would like this number to be as close to one as possible.

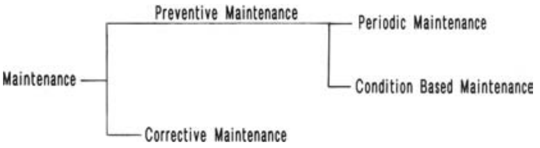

Figure 13.4. Flow Chart of Maintenance Functions.

Table 13.7. Definition of maintenance terms. Source unknown.

Term	Definition
Maintenance	All steps and activities carried out with a view to keeping machines in, or restoring them to, an operable condition.
Preventive maintenance	Maintenance performed systematically for the purpose of preventing a breakdown of a machine during operation or for keeping it an operable condition.
Corrective maintenance	Maintenance performed in order to restore a machine to an operable condition after the occurrence of a breakdown often simply called repair.
Periodic maintenance	Preventive maintenance performed on the basis of predetermined time intervals, i.e. oil change at every 250 hours, filter replacement at every 500 hours, etc.
Condition-based maintenance	Preventive maintenance performed by monitoring the condition of a machine and by detecting signs of possible breakdowns.
Daily maintenance	Maintenance within the scope of the Operation & Maintenance Manual which is performed either daily or by shifts, such as adding oil, greasing, inspecting, cleaning, tightening, and so on.

13.8 ESTIMATING ANNUAL OPERATING TIME AND PRODUCTION CAPACITY

An examination of Figure 13.2 reveals that the flow path from total hours (H) to efficiency hours (E) requires the traversing of 4 nodes. At each node the flow is split and only a fraction continues along each path. In estimating production time and the resulting production one is interested in the upper path. This is shown in Figure 13.5.

The transfer functions

N_1 = operational availability
N_1^* = mechanical availability
N_2 = utilization
N_3 = working efficiency
N_4 = job efficiency

summarized in Table 13.8 are also shown

The relationship between total hours and efficiency hours is given by

$$E = N_1 N_2 N_3 N_4 H \tag{13.10}$$

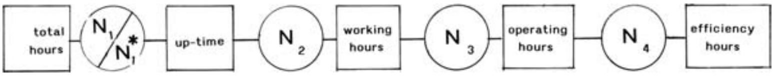

Figure 13.5. Production time flow sheet.

Table 13.8. Summary of the transfer functions.

Node 1:	Total hours = up-time + down lime
	$\text{Operating availability} = \dfrac{\text{up-time}}{\text{total hours}} = N_1$
Node 1*:	$\text{Mechanical availability} = \dfrac{\text{Working hours}}{\text{Working hours} + \text{downtime}}$
Node 2:	Up-time = working hours + standby hours
	$\text{Utilization} = \dfrac{\text{working hours}}{\text{up-time}} = N_2$
Node 3:	Working hours = operating hours + pause hours
	$\text{Working efficiency} = \dfrac{\text{Operating hours}}{\text{Working hours}} = N_3$
Node 4:	Operating hours = efficiency hours + operating delay hours
	$\text{Operating efficiency} = \dfrac{\text{Efficiency hours}}{\text{Operating hours}} = N_4$
Node 5:	Downtime = maintenance hours + supply delay hours + Administrative hours
	$\text{Maintenance efficiency} = \dfrac{\text{Maintenance hours}}{\text{downtime}} = N_5$

To illustrate the use of this consider the following example.

Example 1: Determine the efficiency hours based on the following data:
Total hours = 2000 hrs/year (50 wks/yr, 5 days/wk, 8 hrs/day)
Mechanical availability = 0.80
Utilization = 0.90
Working efficiency = 0.90 (1/2 hr lunch, 18 min. travel)
Operating efficiency = 0.83 (50 min. hour)
Thus,

$$E = 0.8 \times 0.9 \times 0.9 \times 0.83 \times 2000$$

$$E = 1076 \text{ hours}$$

One can also turn this around to determine the number of total hours that would be required to yield a given number of efficiency hours.

$$H = \frac{E}{N_1 N_2 N_3 N_4} \tag{13.11}$$

Example 2:

Efficiency hours required = 2000 hrs
Mechanical availability = 0.80
Utilization = 0.90
Working efficiency = 0.90
Operating efficiency = 0.83

$$H = \frac{2000}{0.80 \times 0.90 \times 0.90 \times 0.83} = 3719 \text{ hours}$$

In example 2, if the mine were to work 1–8 hr shift per day, 5 days/week, 50 weeks per year then the total hours available per machine would be 2000 hours. Hence 2 machines would have to be used. If two machines were purchased then the utilization would drop to that shown in Example 3.

Example 3:

Total hours = 4000
Efficiency hours = 2000
Mechanical availability = 0.80
Working efficiency = 0.90

$$\text{Utilization} = N_2 = \frac{E}{N_1 N_3 N_4 H}$$

$$N_2 = \frac{2000}{0.80 \times 0.90 \times 0.83 \times 4000}$$

$$N_2 = 0.84$$

The calculation of production (P) from a given machine follows directly if the cycle time (T) is known plus the production per cycle (P_o). The following definitions of cycles apply.

shovel: dig, hoist and swing, dump, swing and lower.
truck: spotting at loading machine, loading, loaded travel, turn and dump, travel
drill: collaring, drilling, flushing, retract, moving, leveling

The cycles per hour are calculated from

$$C = \frac{60}{T} \tag{13.12}$$

where
T = minute per cycle
C = cycles per efficiency hour

The total production would then be

$$P = \frac{E \times 60 \times P_o}{T} \tag{13.13}$$

or

$$P = \frac{N_1 N_2 N_3 N_4 H \times 60 \times P_o}{T} \tag{13.14}$$

An example of how this is used is the following:

Example 4:

Assume the 20 cy shovel in Example 1 has an average cycle time of 30 seconds (0.5 minute) and a production of 30 tons per cycle.

The expected yearly production would be

$$P = \frac{1076 \times 60 \times 30}{0.5} = 3{,}873{,}600 \text{ tons}$$

In the mining literature, the product of N_1 and N_2 is given the term "effective utilization".

$$\text{Effective utilization} = N_1 N_2 \tag{13.15}$$

Annual operating hours are obtained from total hours by

$$\text{Annual operating hours} = N_1 N_2 \times \text{total hours} \tag{13.16}$$

13.9 ESTIMATING SHIFT OPERATING TIME AND PRODUCTION CAPACITY

A flow sheet similar to that presented for the annual time breakdown can also be done for a single shift. This is shown in Figure 13.6.

The corresponding productive shift time flow sheet is given in Figure 13.7.

Although Figure 13.6 looks as though it could just be a part of Figure 13.2, it is not. Shift standby and major maintenance times have been shifted from their locations in Figure 13.2 to this flow sheet. The transfer functions become

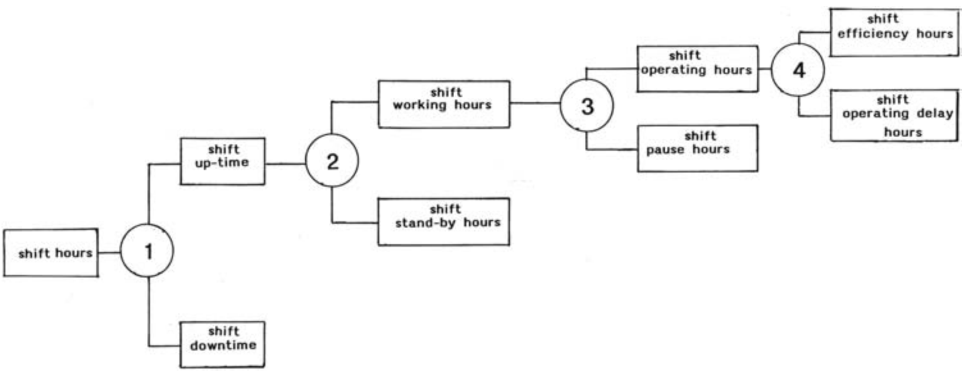

Figure 13.6. Flow sheet for shift hours.

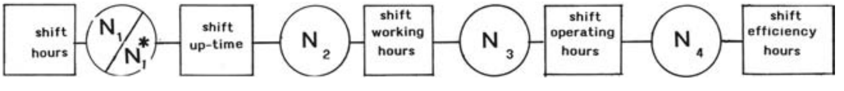

Figure 13.7. Productive shift time flow sheet.

$S_1^* =$ shift mechanical availability

$S_2 =$ shift utilization

$S_3 =$ working efficiency

$S_4 =$ job efficiency.

In this case the shift mechanical availability (S_1^*) will be greater than or equal to the overall mechanical availability. Shift standby hours are those caused by blasting, power cable moves, etc. The relationship between shift hours (H_S) and shift efficiency hours (E_S) is given by

$$E_S = S_1^* \, S_2 \, S_3 \, S_4 \, H_5 \tag{13.17}$$

It is noted that

$S_1 \geq N_1^*$

$S_2 \geq N_2$

$S_3 = N_3$

$S_4 = N_4$

The on-shift mechanical availability (S_1^*) will vary depending on when the maintenance is done and how well it is done. If major maintenance is to be done during the scheduled shifts then, assuming a good program of maintenance, S_1^* for most machines should be of the order of 0.8. If major maintenance is done entirely off shift, then $S_1^* = 1$. If a poor maintenance program is in effect, then $S_1^* < 0.8$. The degree of utilization in a shift will vary with the type of equipment (high for shovels and trucks (0.9 to 1.0) and lower for drills due to blasting shutdowns) and the organizational planning. The working efficiency (S_3) will depend upon the time required for travel and lunch. Job efficiency (S_4) can vary from 0.75 (poor, 45 min hour) to 0.92 (excellent, 55 min hour).

If data are available, then it is best to determine the individual factors and use them in equation (13.17). Sometimes, however, it is convenient for estimating purposes to use equation (13.18) given below

$$E_S = OE \times H_S \tag{13.18}$$

where

$$OE = S_1^* \, S_2 \, S_3 \, S_4$$

The term OE will be referred to as the *operating efficiency*. The expected magnitude of OE can be studied from two examples below.

Case I: *Excellent Conditions*

$S_1^* = 1$ (off − shift maintenance)

$S_2 = 1$ (no standby)

$S_3 = 0.9$ (30 min lunch + 20 min travel)

$S_4 = 0.92$ (55 min hour)

$OE = 1 \times 1 \times 0.9 \times 0.92 = 0.83$

Case II: *Poor Conditions*

$$
\begin{aligned}
S_1^* &= 0.8 \text{ (on-shift maintenance)} \\
S_2 &= 0.9 \text{ (some standby)} \\
S_3 &= 0.9 \text{ (30 min lunch + 20 min travel)} \\
S_4 &= 0.75 \text{ (45 min hour)} \\
OE &= 0.8 \times 0.9 \times 0.9 \times 0.75 = 0.49
\end{aligned}
$$

A low operating efficiency depends upon the magnitude of the non-productive flows in the flow sheet of Figure 13.6.

1. Shift down-time
2. Shift stand-by hours
3. Shift pause hours
4. Shift operating delay hours.

Each of these can be separated into those related to management conditions and those related to job conditions.

Shift down-time depends upon how well maintenance is scheduled and carried out. This is a management function. If the machines are poorly suited to the job to be performed, then a higher level of breakdown is expected. Machine selection is also a management function. If the operators are poorly trained in the operation of their equipment, then a higher breakdown frequency is expected. Such training is also a management function.

The amount of breakdown depends upon the operating (job) conditions under which the equipment is used. For example when digging in material which is blocky, poorly fragmented and highly abrasive, one would expect greater downtime than when digging in a free-flowing sand. When driving over poorly maintained haul roads one would expect a greater amount of time lost in tire and other repairs than on a well maintained road.

Shift stand-by hours are largely dependent upon how well management organizes the different aspects of the operation. If blasting times, cable shifts, etc are well thought out, then the stand-by can be minimized.

The effect of shift pause hours can be reduced, for example, by minimizing travel time and increasing shift lengths. Both of these are management conditions.

Shift operating delay hours are a function of both management and job conditions. Poor scheduling of trucks and shovels can lead to long wait times. A poorly thought out and designed job leads to delays, as well. Poor conditions (excessive rain, ice and snow) also contribute to poor job conditions.

A general check list of job conditions to be considered when selecting an operating efficiency factor are given in Table 13.9.

A general check list of management conditions to be considered when selecting an operating efficiency factor are given in Table 13.10.

Table 13.11 presents operating efficiency factors as a function of management and job conditions. Although specifically developed for mining shovels, it can be applied to other types of mining equipment as well.

From the table, it can be seen that excellent-excellent conditions are similar to the Case I result described earlier. Similarly the poor-poor operating efficiency value is in reasonable agreement with the Case II analysis.

Table 13.9. General job conditions for consideration in selecting an operating efficiency factor (Fiat Allis, 1981).

Conditions	Favorable	Unfavorable
Material	Loose earth, dry clay and loam, coal	Cemented bank materials, wet clay, rock
Excavation area ("cut")	Unobstructed, smooth, dry, well maintained	Wet, slippery or soft. Restricted area
Haul road maintenance	Grader in use full time	No maintenance
Dump or Fill area	No obstructions, dozer maintained	Restricted area, wet or soft, unmatntained
Traffic	Completely independent of public roads and railways	Haul along public of public roads, city traffic
Weather	Consistent	Liable to sudden change
Night shift working	None	Regular
Operator availability	Good	Very Few

Table 13.10. General management conditions for consideration in selecting an operating efficiency factor. Source unknown.

Conditions	Favorable	Unfavorable
Job layout	Well planned and organized	Poor planning, high interference of equipment
Supervision	Adequate in all areas	Very little
Job procedures	Well defined	No definition
Equipment selection	Good combination and arrangement of machines	Equipment inadequate for the job
Equipment operation and care	Operating personnel well trained. Daily maintenance, good condition of consumable parts	Poor training programs. Operating with badly worn consumables
Equipment repair	Good workshops, periodic maintenance, high availability	Inefficient workshops, poor availability

Table 13.11. Operating efficiency factors for mining machinery (Atkinson, 1971).

Job conditions	Management conditions			
	Excellent	Good	Fair	Poor
1. Excellent	.84	.81	.76	.70
2. Good	.78	.75	.71	.65
3. Fair	.72	.69	.65	.60
4. Poor	.63	.61	.57	.52

Values of operating efficiency expressed in terms of operating conditions and machine maintenance are given in Table 13.12.

The "goodness" of operating condition in the first column is to be selected with respect to:

- Suitability of the machine in regard to topography.
- Arrangement and combinations of machines.
- Job site environmental and conditions, such as size, weather and lighting.

Table 13.12. Operating efficiency as a function of operating conditions and machine maintenance. Source unknown.

Operating conditions	Maintenance of machine				
	Excellent	Good	Normal	Rather poor	Poor
Excellent	0.83	0.81	0.76	0.70	0.63
Good	0.78	0.75	0.71	0.65	0.60
Normal	0.72	0.69	0.65	0.60	0.54
Rather poor	0.63	0.61	0.57	0.52	0.45
Poor	0.52	0.50	0.47	0.42	0.32

Table 13.13. Typical rotary drill operating time table (After Heinen (1979)).

Item	Hours
1. Total Hours	8544
2. Maintenance and Repair	1440
3. Operational Restrictions	624
4. Long Drill Moves and Other Major Disruptions	216
5. Personnel Time	936
(a) travel (432)	
(b) lunch (432)	
(c) other (72)	
6. Other Non-Drilling Time	792
(a) lubrication and inspection (288)	
(b) short moves (72)	
(c) running repairs (216)	
(d) other (216)	
7. Net Drilling Time	4536

- Operating method and preparatory planning.
- Skill or experience of operator and supervisor in regard to the operation.

The following points should be considered when selecting the "goodness" level for maintenance of machine.

- Regularity of periodic maintenance.
- Maintenance appropriate for job condition.
- Condition of consumable parts, for example bulldozer cutting edges.

13.10 ANNUAL TIME FLOW IN ROTARY DRILLING

To illustrate the principles discussed in the previous sections, the rotary drilling data in Table 13.13 presented originally by Heinen (1979) will be analyzed.

Annual Working Days

Total calendar time = 365 days
Holidays = 9 days
Annual working days = 356 days

Table 13.14. Annual rotary drilling hours divided into flow sheet categories.

Item		Hours
1.	Total Hours	8544
2.	Downtime	1440
3.	Up-time	7104
4.	Standby	840
	(a) Shift standby (624)	
	(b) Long move (216)	
	(c) Non-scheduled* (0)	
5.	Working Hours	6264
6.	Pause Hours	864
7.	Operating Hours	5400
8.	Operating Delay Hours	864
9.	Efficiency Hours	4536

*No attempt was made to separate the non-scheduled standby from shift standby.

<u>Total Hours</u>

3–8 hour shifts/day = 24 hours/day
Total Hours = 8544 hours

Operational restrictions include;

- shutdown of drills during blasting,
- the need for drill fleets to have excess capacity to ensure a steady supply of blasted material for loading,
- scheduling problems,
- unscheduled maintenance failure of the loading equipment.

Long drill moves and other major interruptions are defined as occurrences when;

- the drill must be moved far enough on one level to require additional trail cable (this is approximately 750 ft),
- the drill mast must be lowered,
- the drill is moved to a new level.

The first task is to match the times with the categories given on the annual time flow sheet (Fig. 13.1). In this case, the following assignments will be made;

maintenance and repair	=	downtime
operational restrictions	=	long move standby
long drill moves	=	long move standby
travel + lunch	=	pause time
other non drilling time +other personnel time	=	operating delay hours

The resulting hours and categories are given in Table 13.14.

These values are presented graphically in Figure 13.8.

In estimating, it is clearly the upper line of boxes in Figure 13.8 which are of importance. Substituting these figures into the appropriate boxes on the productive time flow diagram (Fig. 13.9) one can easily calculate the transfer functions (efficiencies).

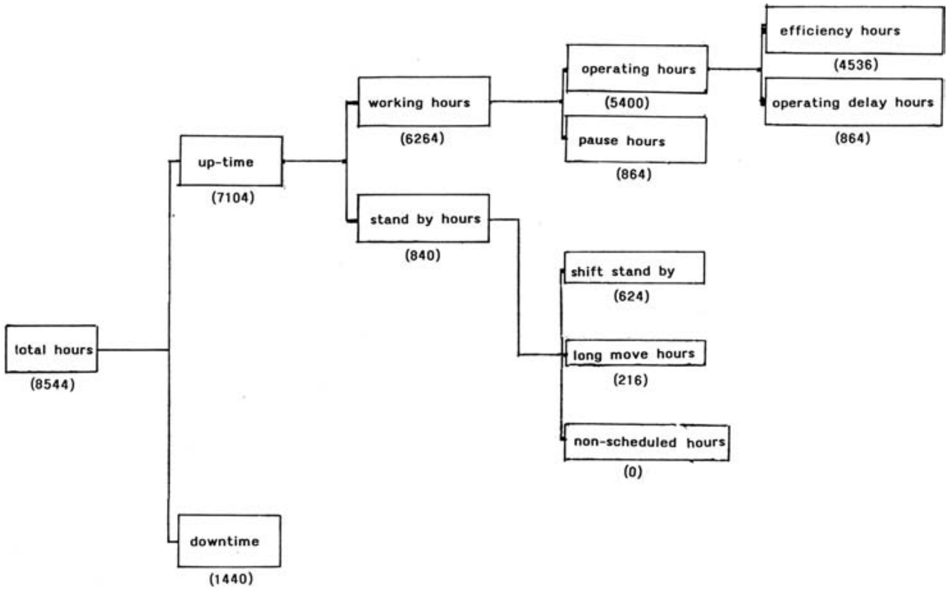

Figure 13.8. Annual time flow sheet for rotary drilling.

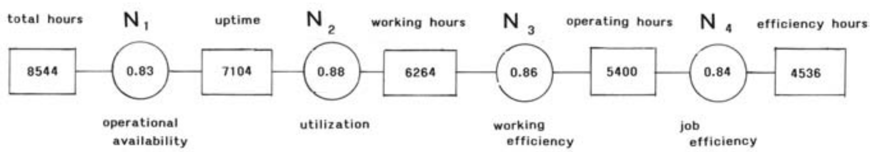

Figure 13.9. Productive time flow in rotary drilling.

In this case;

operational availability $(N_1) = 0.83$
utilization $(N_2) = 0.88$
working efficiency $(N_3) = 0.86$
job efficiency $(N_4) = 0.84$

The operating efficiency is

$$OE = N_1N_2N_3N_4 = 0.53$$

The mechanical availability from Figure 13.8 is

$$MA = \frac{\text{downtime}}{\text{working hours} + \text{downtime}} = 0.81$$

which is similar to the operational availability due to the relatively small amount of standby time. The job efficiency of 0.84 corresponds to a 50 minute hour. The working efficiency of 0.86 corresponds about to 1/2 hour travel and 1/2 hour lunch per shift. Using the following estimating values for a good operation;

mechanical availability $(N_1^*) = 0.8$
utilization $(N_2) = 0.85$

working efficiency (7 hr/8 hr shift) = 0.875
job efficiency (50 min/hr) = 0.83

one would get an annual operating efficiency of

$$OE = 0.49$$

This is within 8% of the actual values for this rotary drill.

Using the transfer functions, one can estimate the average number of efficiency hours in an 8-hour shift scheduled for operation. This presumes that maintenance is done in whole shifts (in this case 180) and the other whole shifts are used in production. Including all standby time, the efficiency hours per 8-hour shift would be

$$N_2N_3N_4 \times 8 = 0.88 \times 0.86 \times 0.84 \times 8 = 5.1 \text{ hours/shift}$$

The number of 8-hour shifts to which this would apply is 888. Scheduled long moves however occupy 27 shifts (216 hours) during the year and it is probable that little if any drilling would be done in those shifts. Including only shift standby time, the efficiency hours per 8-hour shift would be

$$N_2^*N_3N_4 \times 8 = 0.91 \times 0.86 \times 0.84 \times 8 = 5.25 \text{ hours/shift}$$

This would apply to 861 shifts. In this latter case, the breakdown by shift for the year would be

Total shifts = 1068 shifts
Maintenance + Repair = 180 shifts
Long moves = 27 shifts
Working shifts = 861 shifts

13.11 APPLICATION IN PREFEASIBILITY WORK (Schumacher, 2012)

In prefeasibility work, the process is more or less reversed. Instead of beginning with an accounting of hours, one begins with an assumed factor of availability and another for operating efficiency, part of which cannot necessarily be defined as hours, but rather as a measure of effectiveness, e.g. ability to cycle in a reasonable time, or ability to spot a truck for loading quickly and properly. These factors are then applied to productivity studies to determine (1) how many equipment units must be purchased, and (2) how many hours per day each unit must operate. Hence, the purpose of the presented charts and definitions with respect to prefeasibility studies would be to help understand what is being filtered out by the factors. It would be most helpful for prefeasibility studies, to show your first cut as a division between available hours and unavailable hours, and your second cut of available hours between efficient operating hours and non-cycling hours. The availability factor would then be defined as the ratio of available hours to total hours, and the efficiency factor as the ratio of efficient operating hours to available hours.

Unavailable hours would include maintenance hours, breakdown hours (non-functional hours), supply delay hours, administrative hours, and probably move time. Non-scheduled hours and stand-by hours should not be included here.

Non-cycling hours would include potty breaks, traffic delays, possibly weather delays, pauses to confer with supervisor, waiting time, etc.

A key consideration is that hourly operating costs are applied to all the operating hours, whether cycling or not.

REFERENCES AND BIBLIOGRAPHY

Aghajani, A., Osanloo, M., & M. Akbarpour. 2007. Optimising the Loading System of Gol-e-Gohar Iron Ore Mine of Iran by Genetic Algorithm, AUSIMM Conference Proceedings, Iron Ore 2005 Conference, Perth, WA, August 2007, pp. 211–215.

Anonymous. 1962. Measuring the cost of unavailability. E&MJ. 163(6):102–103.

Atkinson, T. 1971. Selection of open pit excavating and loading equipment. *Trans. Instn of Min and Met.* July. pp. A101–A128.

Atkinson, T. 1992. Chapter 13.3 Selection and sizing of excavating equipment. SME Mining Engineering Handbook (H.L. Hartman, editor). pp. 1311–1333.

Bernard, R. 2001. The path to profitability: Optimizing availability of your key critical aspects. Paper 01-13. SME Annual Meeting. Feb 26–28. Denver, CO.

Carter, R.A. 2006. Power play: New electric motors, drives and controllers designed for mine-duty service. E/MJ. 207(9):34–37. September.

Carter, R.A. 2011. Clean machines: Cut operating costs with contamination control. E/MJ. 212(1):42–47. January/February.

Carter, R.A. 2011. Remote possibilities: Mining fleet operators continually look for methods and technologies that lead to better asset management. Remote machine health monitoring provides a wide range of opportunity for improvement in this area. E/MJ. 212(2):42–48. March.

Caterpillar. 2003. Caterpillar Performance Handbook. Edition 34.

Caterpillar. 2012. Caterpillar Performance Handbook. Edition 42.

Chadwick, J. 2005. OEM components – From the ground up. Mining Magazine. February. pp. 12–20.

Chan, J., & M. Kuruppu. 2006. Mining Equipment Maintenance Management, AUSIMM Conference Proceedings, The AusIMM New Leaders' Conference Kalgoorlie, WA, April 2006, pp. 107–110.

Fiatallis. 1981. Performance Handbook. Sept. 340 pp.

Fiscor, S. 2007. New system enables fleets to tailor maintenance programs. E/MJ. 208(9):40–43. November.

Heinen, R.H. 1979. Chapter 18 Drill evaluation. Open Pit Mine Planning and Design (J.T. Crawford and W.A. Hustrulid, editors). AIME, New York. pp. 245–255.

International. 1975. Earthmoving Principles: A Guide to Production and Cost Estimating. International Pay Line Division. 111 pp.

Knights, P., Jullian, F., & L. Jofre. 2006. Building Business Cases for Maintenance Improvement Projects, AUSIMM Conference Proceedings, International Mine Management Conference, Melbourne, Vic, October 2006, pp. 173–178.

Knights, P.F., & P. Oyanader. 2005. Best-in-class maintenance benchmarks in Chilean open pit mines. CIM Magazine. June/July. pp. 325–330.

Komatsu. 1993. Specifications and Application Handbook. Edition 15.

Komatsu. 2009. Specifications and Application Handbook. Edition 30.

Lumley, G. 2009. How can you get mining equipment to work to its real capacity? AusIMM New Leaders Conference. Brisbane, QLD. April 29–30. pp. 17–23.

Morey, P.G., & K.J. Ashley. 1984. Open pit mine design. 1984 NWMA Short Course on Mine Feasibility. Dec 3–5. Spokane, WA. 54 pp.

Myntti, D.C. 1978. Elements of an advanced open-pit equipment maintenance program. Mining Engineering. 29(3):259–60.

Najor, J., & P.C. Hagan. 2007. Improvements in Truck and Shovel Scheduling Based on Capacity Constraint Modelling, AUSIMM Conference Proceedings, Large Open Pit Mining Conference Perth, WA, September 2007, pp. 87–91.

Nix, M. 2007. Location Aware Mine Machinery for Productivity and Safety Improvements, AUSIMM Conference Proceedings, First International Future Mining Conference Sydney, NSW, November 2008, pp. 175–178.

Rahne, F. 1970. Equipment performance. Proceedings, 31st Annual Mining Symposium, and 43rd Annual Meeting, Minnesota Section, AIME, University of Minnesota, Duluth. Jan 1–12. pp. 169–172.

Schumacher, O. 2112. Personal communication. July.
Sense, J. 1968. Chapter 10.2 Equipment scheduling – Including utilization and availability. Surface Mining (E.P. Pfleider, editor). AIME. New York. pp. 659–670.
Sense, J.J. 1964. Asarco's method of keeping its equipment fleet rolling at maximum efficiency. Mining Engineering. 15(3): pp. 35–38.
Terex. 1978. Production and Cost Estimating of Material Movement With Earthmoving Equipment. 79 pp.
Terex. 1981. Production and Cost Estimating of Material Movement With Earthmoving Equipment. 82 pp.
Tomlingson, P.D. 1982. Effective maintenance – An essential ingredient of successful mining operations. Mining Engineering. 33(12):1673–1676.
Tomlingson, P.D. 1985. Implementing and using a fully integrated maintenance management information system. Mining Engineering. 36(8):1047–1050.
Tomlingson, P.D. 2004. Maintenance – A gold mine of improvement opportunity. Paper 04-22. SME Annual Meeting. Feb 23–25. Denver, CO.
Tomlingson, P.D. 2005. Maintenance information: Value added? E/MJ. 206(8):54–56. October.
Tomlingson, P.D. 2006. Achieving world class maintenance status. Mining Engineering. 58(11):30–33. November.
Tomlingson, P.D. 2007. Achieving world class maintenance status. E/MJ. 208(2):38–40. March.
Tomlingson, P.D. 2007. Mine Maintenance Management Reader. SME. 96 pp.
Tomlingson, P.D. 2008. Maintenance program: a void with penalties or a path to effectiveness. Mining Engineering. 60(3):36–40. March.
Tomlingson, P.D. 2010. Equipment Management Workbook. SME. 152 pp.
Tomlingson, P.D. 2010. Equipment Management: Key to Equipment Reliability and Productivity in Mining, 2nd edition. SME. 324 pp.
Tomlingson, P.D. 2011. Managers and maintenance: Guidelines for optimizing performance. Mining Engineering. 63(1):45–52. January.
Werner, J., & M.W. Lewis. 2005. The integration of condition monitoring into the maintenance process. CIM Magazine. January/February. pp. 163–168.
Zaburunov, S.A. 1986. Eyeball engineering is easy to master if you know what to look for. Coal Mining. June. pp. 42–43.
Zywiel, J., Yuet, F., King, K., & J. Struble. 2005. Integrated inertial/GPS positioning system for open pit mine applications. Paper 05-062. SME Annual Meeting. Feb 28–Mar 2. SLC, UT.

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REVIEW QUESTIONS AND EXERCISES

1. Define the following terms:
   - Availability
   - Operational availability
   - Mechanical availability
   - Physical availability
2. Define the following terms:
   - Utilization
   - Use of availability
3. Define the following terms:
   - Working efficiency
   - Job efficiency

   - Operating efficiency
   - Effective utilization
4. Differentiate between the following:
   - Scheduled hours
   - Annual hours
   - Total hours
   - Working hours
   - Shift hours
   - Operating hours
   - Efficiency hours
5. Be able to reproduce the time flow diagram shown in Figure 13.1. Are the included components logical? Are the descriptions given in Table 13.1 clear? Why or why not?
6. Should a Node 6 be added in Figure 13.2? If so, how would it be drawn? If not, why not?
7. Study the nodal descriptions and be able to reproduce them.
8. What is meant by the abbreviations MTBF and MTTR?
9. What is meant by "inherent availability"?
10. Why is the "mechanical availability" always less than the "operational availability"?
11. How does the mechanical availability vary with the life of the machine?
12. How does your answer to problem 11 relate to the "typical" values given in Table 13.3?
13. Explain the significance of Figure 13.3.
14. Distinguish between availability and utilization.
15. Working efficiency is an important concept. What does it include? Does it depend on shift length? Why or why not?
16. Some mining operations employ "hot changes" where the shift changes occur at the machine location. Discuss the pro's and con's of this practice. Which time and/or efficiency term does practice this impact?
17. To keep the machines operating as much as possible during a shift, sometimes extra personnel take over during lunch breaks. Is this a good idea? Why or why not? Which time and/or efficiency term does practice this impact?
18. At some operations, the operators do not get a real meal break but eat their meals "on the fly". Discuss the pro's and con's of this practice. Which time and/or efficiency term does practice this impact?
19. What delay factors are included in "job efficiency"?
20. What is meant by the common term "efficiency hour"? What are some typical values? What might be a reason for a low value?
21. Define and be able to distinguish between the following terms:
   - Preventive maintenance
   - Corrective maintenance
   - Periodic maintenance
   - Condition-based maintenance
   - Daily maintenance
22. Work the four examples.
23. What things give rise to low operating efficiency?
24. Review in detail the rotary drilling annual time flow example.
25. Obtain information from a mining operation or from the literature concerning equipment availability/utilization. Apply the time flow approach to your data set.

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OPEN PIT MINE PLANNING & DESIGN
VOLUME 2 – CSMine and MicroMODEL SOFTWARE PACKAGES AND OREBODY CASE EXAMPLES

Contents

CHAPTER 14

The CSMine tutorial

Computers play an important role in most phases of the mining business today. Their increasing use in mine planning and design has meant that the engineers and planners can spend more time evaluating alternative designs and asking what if types of questions rather than in performing tedious hand calculations and drawing and redrawing the various maps by hand. There are a number of different mine design software packages commercially available and currently being used by both mining companies and mining universities throughout the world. Each package differs rather significantly from the next. Although becoming more user-friendly over time, all require a rather large time commitment to become proficient in their use. In the surface mine design courses taught at most universities, the total time spent in the classroom and laboratory is quite limited. Within this time frame both the fundamentals and the applications must be covered. To accomplish this demonstration of the principles within the available time, a special, very user-friendly commercial quality software package called CSMine has been developed. To demonstrate the use of the package, a drill hole data set and the topography maps for the Arizona Copper property have been included. The Arizona Copper property is described in Chapter 17.

Introduction to the software package is through the use of the CSMine tutorial included in this chapter. It is expected that the diligent student should be confidently running the software within 3 hours of first exposure. The CSMine users manual, included as Chapter 15, is intended to answer more detailed questions about the package.

By being able to design and plan in detail an actual open pit property, albeit on a somewhat reduced scale, through the use of the software package the student's learning experience is increased. With the principles firmly in hand, the step up to the larger, more powerful commercial software packages available in industry should be made easier.

WARRANTY

The software and accompanying written material are provided 'as is' without warranty of any kind. Further, the authors do not warrant, guarantee, or make any representations regarding the use, or the results of the use of the software or written materials in terms of the correctness, accuracy, reliability, currentness, or otherwise. The entire risk as to the results and performance of the software is assumed by you. If the software or written materials are defective, you and not the authors assume the entire cost of all necessary servicing, repair, or correction.

14.1 GETTING STARTED

The CSMine program included in edition 1 of this book was written for the DOS operating system. The program has been completely rewritten as a more modern Windows program using the Microsoft Visual C++, version 6.0, software development system. The program has been tested using a standard PC computer running the Windows XP operating system. Compatibility with previous or future versions of Windows cannot be guaranteed.

14.1.1 *Hardware requirements*

The CSMine program should run properly on any PC computer capable of running the Microsoft Windows XP operating system. Printers and plotters are supported through the Windows interface. A color printer is recommended but not required. Printing is currently limited to standard Letter (8.5 by 11 inches) or A4 (210 by 297 mm) sized paper.

14.1.2 *Installing CSMine*

CSMine comes on one standard CD disk that includes the program files and the demonstration data sets needed to run the tutorial. The following subdirectories and files should be present:

```
\Program_Files
    CSMine.exe
    MFC42D.DLL
    MFC042D.DLL
    MSVCIRTD.DLL
    MSVCRTD.DLL
\Data_Files
    Tutor.kon
    Ex1.cmp
    Ex2.cmp
    \Ariz_Cu
        ArizCu.kon
        ArizCu.dhf
        ArizCuElev.elv
```

A number of folders are present within the folder named 'Data_Files'. Each of these folders contains a data set that may be used with CSMine. Only the contents of the folder 'Ariz_Cu' are shown here.

The InstallShield installation program can be used to install CSMine. From the root directory on the CD:

SELECT: the 'SETUP.exe' program

Follow the instructions given by the InstallShield Wizard. It is recommended that the directory C:\\CSMine be selected as the destination location for the installation.

To manually install the program using Windows Explorer, copy the directory structure and files as shown above directly to your computer's hard disk. It is recommended that the program files be copied to the following directory:

```
C:\\CSMine
        \Program_Files
            CSMine.exe
            :
            :
        \Data_Files
            Tutor.kon
            :
            :
            \Ariz_Cu
                ArizCu.kon
                :
                :
```

Create a desktop shortcut to the program by right clicking on the program file CSMine.exe and dragging it to the desktop.

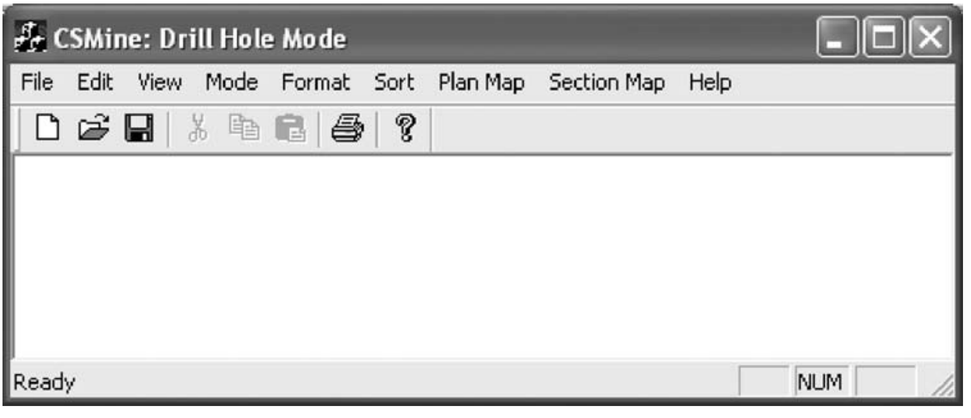
To uninstall CSMine, if the InstallShield program was used for installation, from the directory where the program was installed:

SELECT: the 'uninst.isu' program

Manually delete all the installed files if the program was installed manually.

14.1.3 *Running CSMine*

The program can be started by double clicking on the CSMine desktop icon, or by double clicking on the program CSMine.exe from Windows Explorer. Once started, the main program screen should be displayed.



14.2 THE ARIZONA COPPER PROPERTY DESCRIPTION

The Arizona Copper data set is used throughout the tutorial presented in this chapter. The Arizona Copper data set used throughout this chapter is from the Arizona Copper Property described in some detail in Chapter 17. It is a low-grade porphyry type of copper deposit typical of the southwest United States. There are 40 vertical exploration drill holes with an average length of 250 ft. The area covered by the exploration drilling is approximately 1400 ft × 1000 ft. Copper assays expressed in percent copper are given at an average spacing of 5 ft along the length of each drill hole. The Arizona Copper data set contains drill hole data for a low-grade porphyry type copper deposit typical of the southwest United States. There are 40 vertical exploration drill holes with an average length of 250 ft. The area covered by the exploration drilling is approximately 1400 ft × 1000 ft. Copper assays at an average spacing of about 5 ft are given along the length of each drill hole expressed in percent copper.

14.3 STEPS NEEDED TO CREATE A BLOCK MODEL

In this section, the steps required to create a block model are briefly outlined.

(a) Create a drill hole data file containing the collar coordinates of the exploration drill holes, the hole directions, and the assay values along the hole lengths. The file 'ArizCu.dhf' is included as an example drill hole file and will be used in this tutorial.

(b) Plot the locations of the drill holes in plan view. This has been done for the 'ArizCu.dhf' data set and the plan map is shown in Figure 14.1.

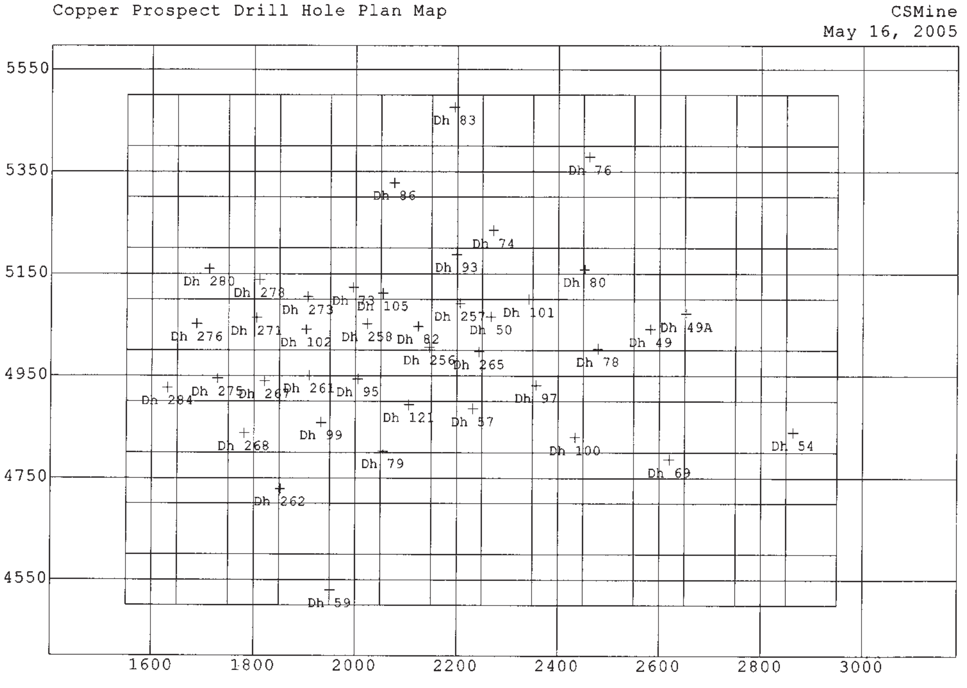


Figure 14.1. Example drill hole plan map.

(c) Plot vertical sections through lines of drill holes in order to examine probable pit locations. An example vertical section along the East-West line through North 4950 of Figure 14.1 is shown in Figure 14.2.

(d) Decide on the size and location of the blocks and the model grid. The height (B_z) of the blocks is usually set to the planned bench height. The block width (B_x) and (B_y) length dimensions are chosen arbitrarily (normally equal to one another) but a rule of thumb is that the block size (in plan) should not be less than one-fourth the average drill hole spacing. The location of the grid can be decided upon by examining the plan and section maps and locating the block grid such that the potential pit area (including a provision for the pit slopes) is covered. The block dimensions, their total number and their locations can be easily changed with CSMine. However, each time such a change is made, a new topography file must be created, which is a lot of work. Therefore the creation of the topography file should be done as a last step. With CSMine, the block grid must run North-South, East-West. The block grid selected for the tutorial is shown superimposed on Figure 14.1 and Figure 14.2. The blocks are 100 ft × 100 ft × 50 ft. There are 14, 10, and 12 blocks in the X, Y, and Z directions respectively.

(e) Create a composite file from the drill hole file. The drill hole file will usually contain assay values for core samples of varying length. In order to estimate block values, the assay samples must be regularized to form composites defined over core sections of equal length. It is common in open pit mining to choose the composite lengths to be equal to the bench height, and to calculate the composites such that their centers fall at the midpoints of the blocks. Figure 14.3 shows the section plotted in Figure 14.2 with 50 ft composites.

(f) Create a surface topography file. This file contains the surface elevation at the center (in plan) of each block. Figure 14.4 shows the chosen block grid superimposed over the surface

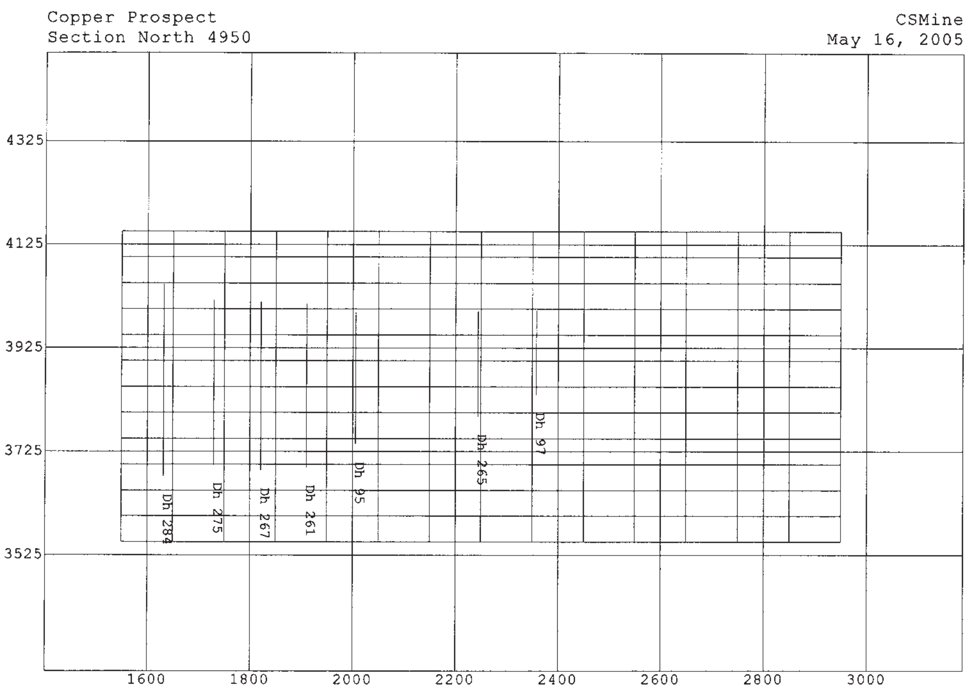


Figure 14.2. Example drill hole section map.

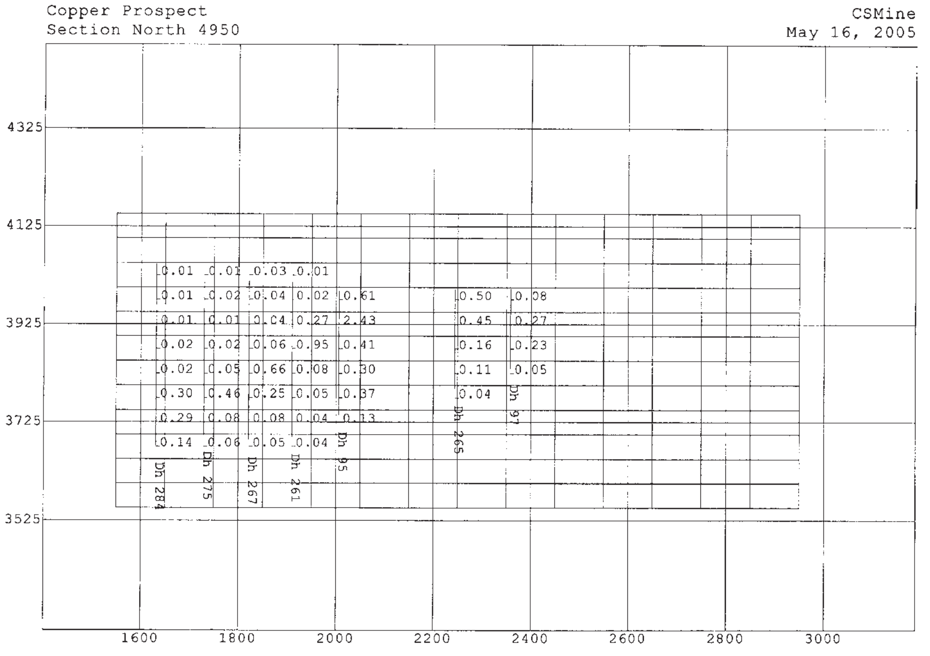


Figure 14.3. Section map showing composites.

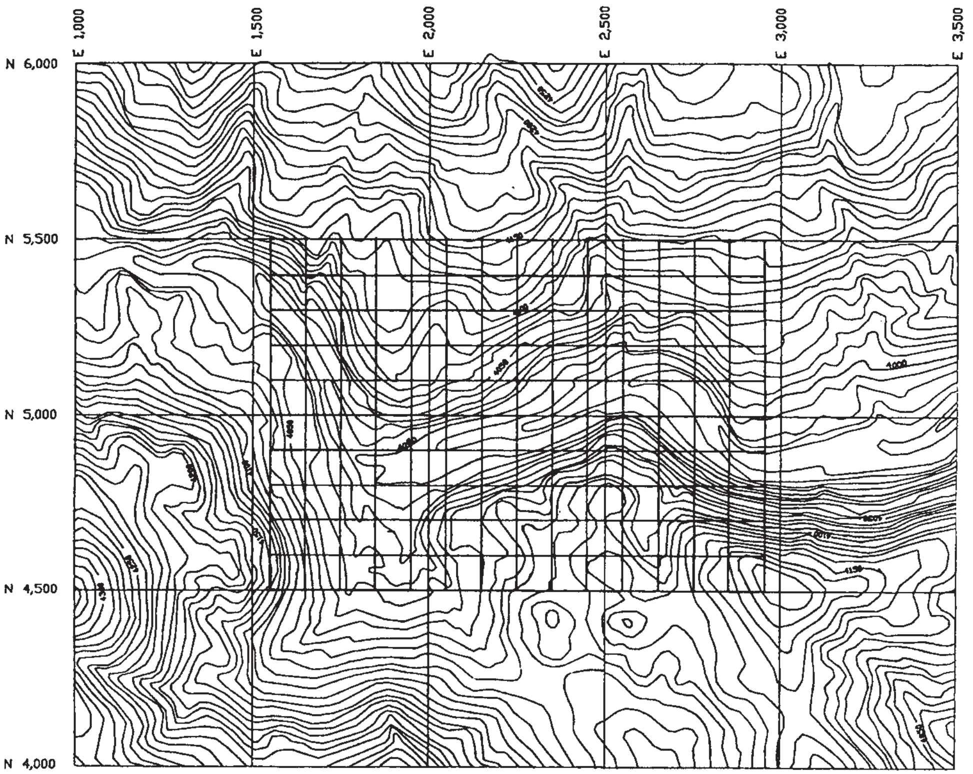


Figure 14.4. Surface topography map and block grid.

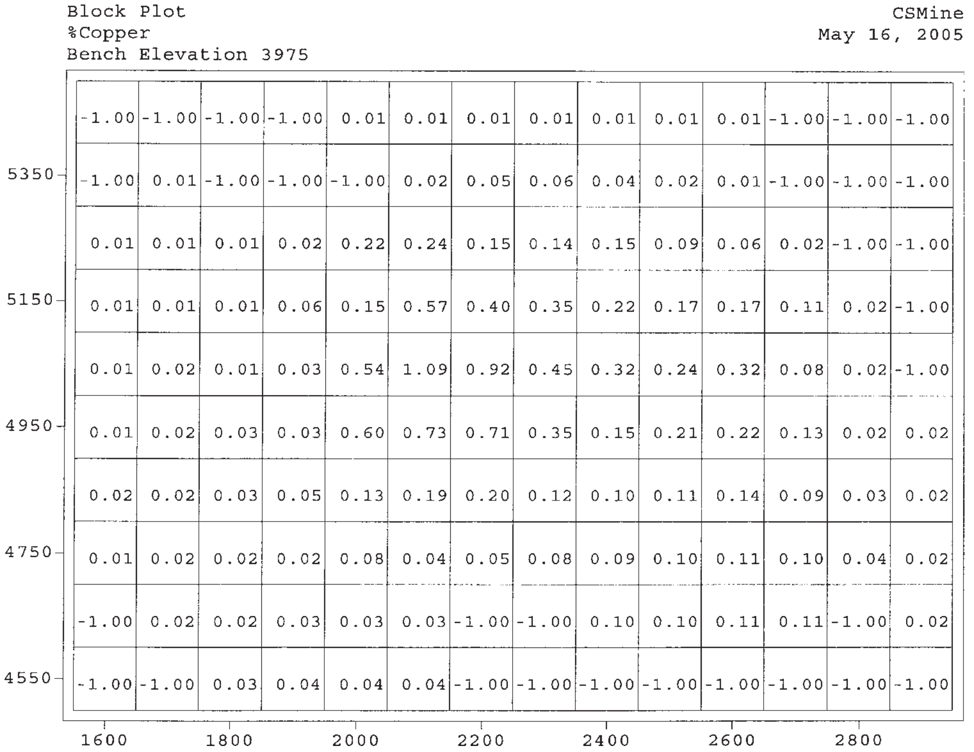


Figure 14.5. Horizontal block plot map showing the block grades for Bench 3975.

topography map of the property. The values needed for the surface topography file can be easily read off this map and entered into a file. The surface topography file is optional. The block model can be created with or without the use of surface elevations.

(g) Assign the block grades. After preparing the necessary data files and deciding on the size and location of the block model, a grade must be assigned to each block. In this tutorial, the inverse distance squared method will be used for assigning block values.

(h) Create horizontal plan maps and vertical section maps through the block model to be used for planning purposes. A horizontal plan map showing the block values for Bench 3975, is shown in Figure 14.5.

(i) Assign economic values to the blocks. From the block grades and estimated mining costs, an economic value for each block is assigned.

(j) Calculate the final pit limits. From the economic block values, the final pit limit shell may be calculated using one of several available optimization techniques. CSMine uses a simple Floating Cone algorithm. The final pit shell found for the tutorial example is shown in Figure 14.6.

14.4 DATA FILES REQUIRED FOR CREATING A BLOCK MODEL

Three basic types of data files are required for creating the block model:

(a) The drill hole file contains the X, Y, and Z coordinates and direction information along with the raw assay data for each drill hole in the deposit. The user must create the drill hole file before running CSMine.

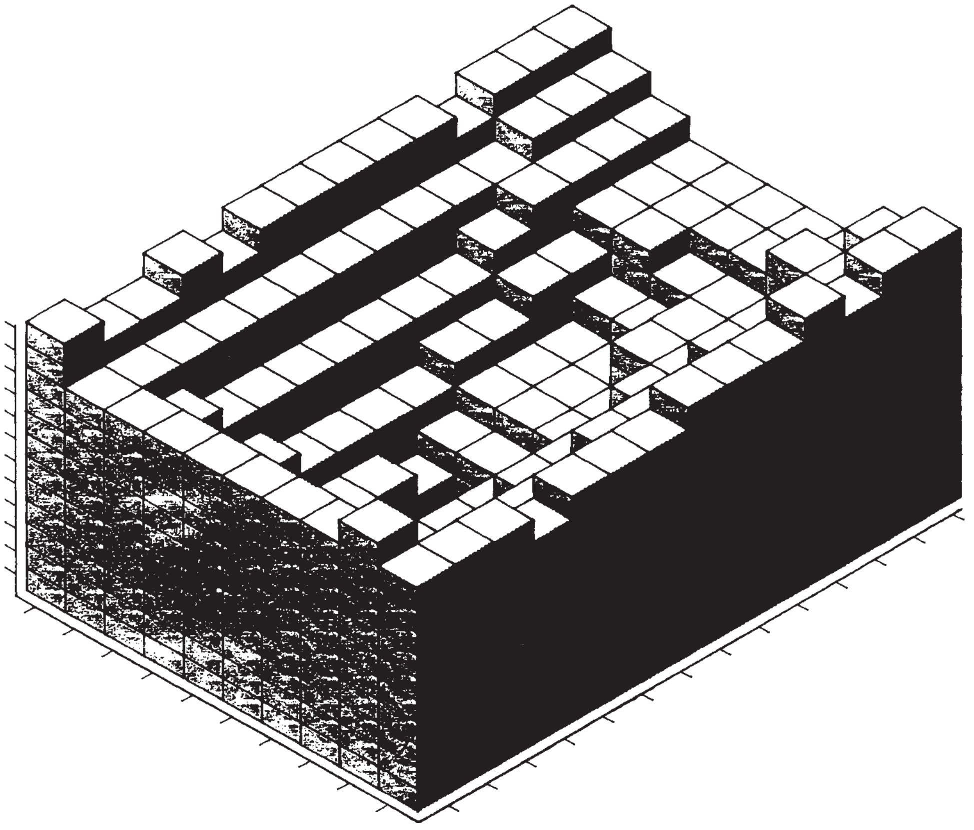


Figure 14.6. Isometric view of the final pit limit shell.

(b) The composite file contains the X, Y, and Z coordinates and the assay values for each regularized composite. CSMine creates this file from the drill hole data.

(c) The block file contains the block values for each block in the model. CSMine creates this file from the composite file.

(d) The surface elevation file (optional).

To summarize, before running CSMine, the user must create a drill hole file. The user may also create an optional surface elevation file. From the drill hole file, CSMine creates the composite file. From the composite file with or without the optional surface elevation file, CSMine creates the block file.

14.5 CSMine PROGRAM DESIGN OVERVIEW

The design of CSMine closely follows the three basic types of data the program is required to handle. The program is divided into three modes:

(a) The Drill Hole mode -used to read in the drill hole data file and plot plan and drill hole section maps.

(b) The Composite mode -used to create regularized composites from the raw drill hole data. Composite files may be stored on disk and read in at a later time from this mode.

(c) The Block mode -used to create the block model from the composite values. The Block mode contains by far the most program commands including those for assigning the block mineral values, the economic block values, the pit limits, and for generating a large number of plots including block value plots and contour maps.

Thus a typical project using CSMine would include the following basic steps:

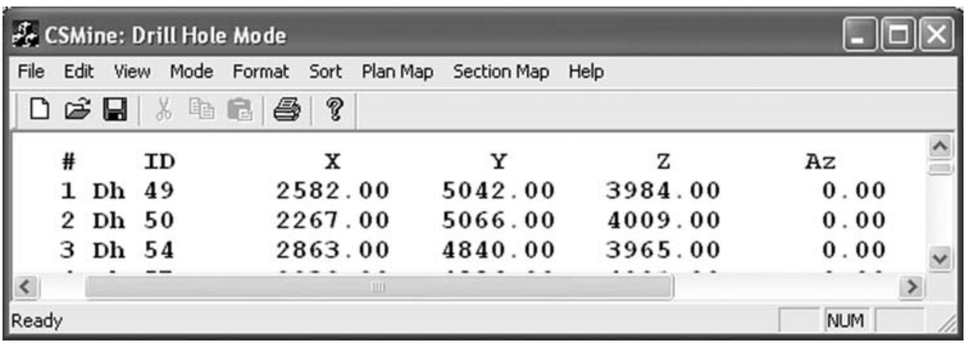
(a) Use the Drill Hole mode to read in the drill hole file.

(b) Use the Composite mode to create the regularized composites from the drill hole file.

(c) Use the Block mode to create the block grade model, the economic block model, and generate the final pit limits.

14.6 EXECUTING COMMANDS WITH CSMine

The 'Drill Hole Mode' main data window is typical of the program screens found in CSMine. At the top of the window is a menu with a short descriptive name for each command available.

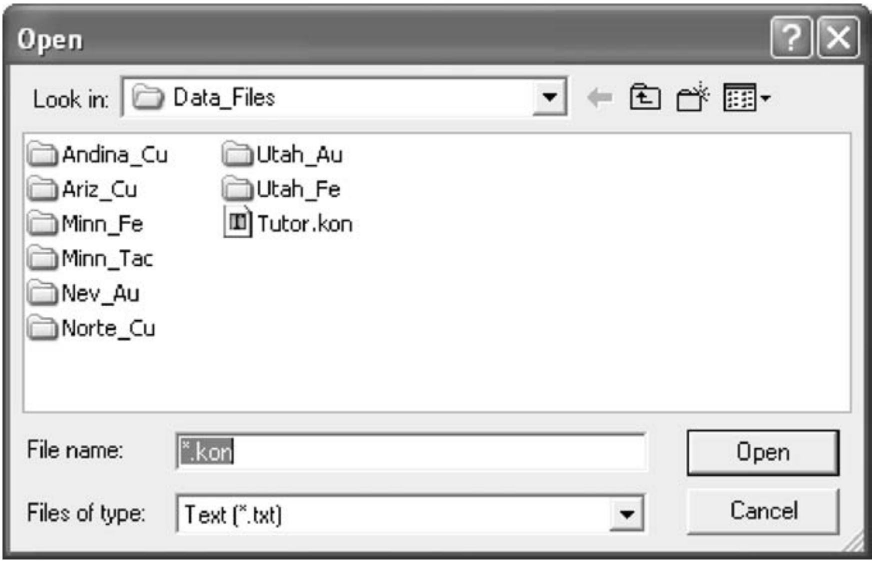


Commands are executed by either clicking the mouse on the command name, or by holding down the 'ALT' key and the first letter of the command at the same time. One of four things will occur when a command has been selected: 1) the command will execute immediately, 2) a drop down menu will appear from which a new command can be selected, 3) a dialog box will be opened, or 4) an error message will be displayed. Error messages usually inform the user that the data required for the command are not available. For example, one cannot plot a drill hole plan map until drill hole data have been read.

14.7 STARTING THE TUTORIAL

First start CSMine using one of the methods described in Section 14.1.3. The user can now proceed through the various steps of the program (from entering the drill hole data through to final pit generation) entering the required menu input values in the process. If upon completing the process (generating the various sections, etc.), the user simply exits the program, none of these menu values will be saved (stored). The next time the same problem is run, the user would have to enter these values again. Although not difficult, it is tedious and presumes that the user has noted all of the values used. To avoid this, a project file can be created and used to store program menu values. One obviously can not avoid entering the required menu input values the first time but once generated this file of input values can be stored as a project file. The next time the same problem (drill hole set) is run, this previously generated and stored project file can be recalled. The menu items will be set to those contained. The user therefore need not begin from the beginning each time.

For this tutorial a project file 'Tutor.kon' has been created and stored on disk. The first step in the tutorial is to enter this file into the program by utilizing the 'Load Project File' command found in the drop down menu that will appear when the 'File' command shown at the top of the program main window is selected. In this way the first-time user can concentrate on learning about how the program functions rather than in completing the various menu items. Obviously, when beginning the analysis of another drill hole data set, there will be no existing project file and hence no project file to load. However, the items forming the future project file will be automatically stored as the user enters the menu items. If the program is simply exited, then this file of values will be deleted. Therefore, the last step, prior to exiting the program, irrespective of how far along the user has come, should be to select the 'Save Project File' command found in the drop down menu under the 'File' command, and to store the menu values in a project file with the desired name and the extension ".kon". When returning to the problem, the recall of this project file is the first step. The project file "ArizCu.kon" has also been included. It is one possible way for evaluating the Arizona Copper data set.



To load the project file for the tutorial:

SELECT: the 'File' command

From the drop down menu:

SELECT: the ‘Load Project File’ command

The standard Windows file dialog box will appear.

SELECT: the file ‘Tutor.kon’

The ‘Tutor.kon’ project file will then be read and the contained menu values stored.

14.8 THE DRILL HOLE MODE

In this section of the tutorial, the commands associated with the Drill Hole mode are covered. This mode is used mainly for reading the drill hole file and for creating drill hole plan and section maps. Defining the block model grid is also covered.

14.8.1 *Reading the drill hole file*

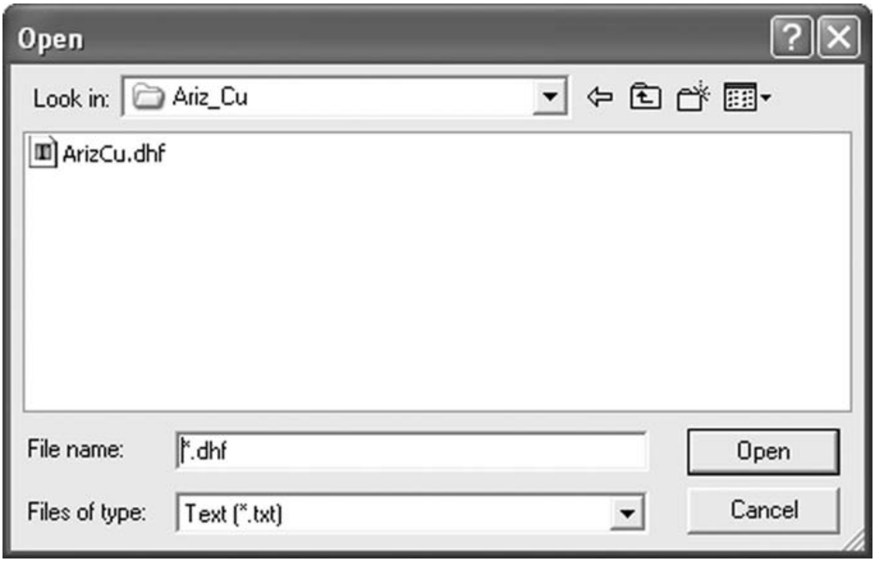
The next step is to read in the drill hole file “ArizCu.dhf”. From the ‘Drill Hole Mode’ menu:

SELECT: the ‘File’ command

From the drop down menu:

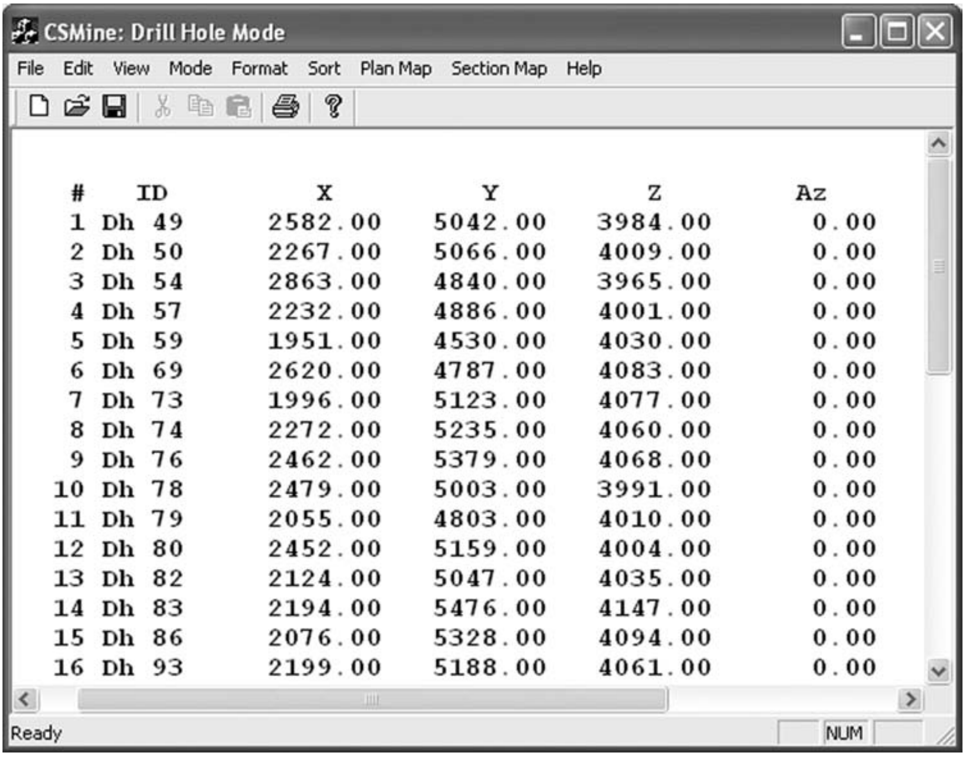
SELECT: the ‘Open’ command

The standard Windows file dialog box will appear.



SELECT: the file ‘ArizCu.dhf’

The drill hole file 'ArizCu.dhf' will then be read and the drill hole definition records displayed.



#	ID	X	Y	Z	Az
1	Dh 49	2582.00	5042.00	3984.00	0.00
2	Dh 50	2267.00	5066.00	4009.00	0.00
3	Dh 54	2863.00	4840.00	3965.00	0.00
4	Dh 57	2232.00	4886.00	4001.00	0.00
5	Dh 59	1951.00	4530.00	4030.00	0.00
6	Dh 69	2620.00	4787.00	4083.00	0.00
7	Dh 73	1996.00	5123.00	4077.00	0.00
8	Dh 74	2272.00	5235.00	4060.00	0.00
9	Dh 76	2462.00	5379.00	4068.00	0.00
10	Dh 78	2479.00	5003.00	3991.00	0.00
11	Dh 79	2055.00	4803.00	4010.00	0.00
12	Dh 80	2452.00	5159.00	4004.00	0.00
13	Dh 82	2124.00	5047.00	4035.00	0.00
14	Dh 83	2194.00	5476.00	4147.00	0.00
15	Dh 86	2076.00	5328.00	4094.00	0.00
16	Dh 93	2199.00	5188.00	4061.00	0.00

14.8.2 *Defining the block grid*

Before plotting a drill hole plan map, it is necessary to jump ahead and explain the architecture of the block model since the block grid can if desired be plotted on drill hole plan and section maps. The block model is described under the Block mode. To reach the model descriptor one changes to the Block mode, then chooses the 'Calculate' command and finally the 'Values by IDS' (Inverse Distance Squared) command. Residence of the block model descriptor under IDS is logical since this is where the block center coordinates are needed for block grade assignment. To reach this location from the 'Drill Hole Mode' menu:

SELECT: the 'Mode' command

From the drop down menu:

SELECT: the 'Block' command

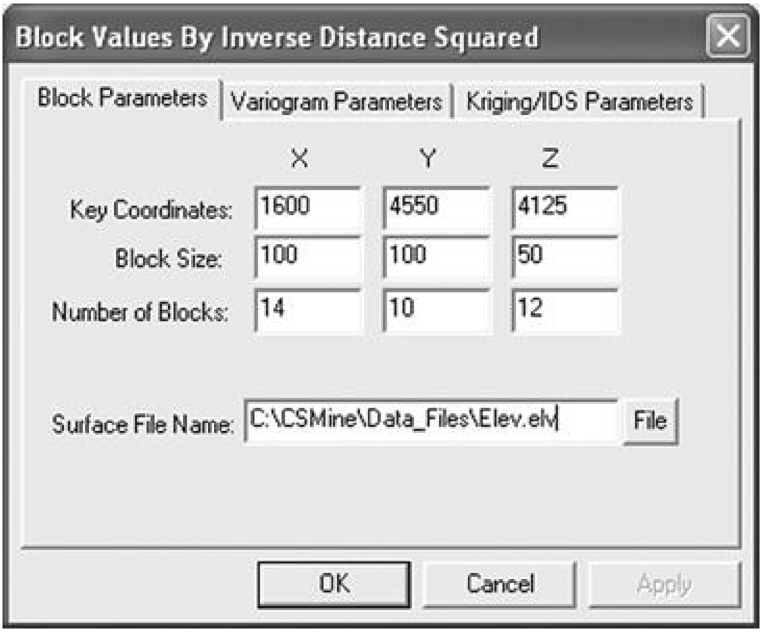
The data screen will then be cleared as there are no block values to display, and the screen will show 'Block Model Mode' along the top of the window.

SELECT: the 'Calculate' command

From the drop down menu:

SELECT: the 'Values by IDS' command

The 'Block Values By Inverse Distance Squared' dialog box will then be displayed.



The block parameters that define the block grid are defined under the 'Block Parameters' tab. Two types of coordinate systems are used when referencing the block grid. The first is the X, Y, Z system, which is used to provide coordinates in true space. The X-axis runs East-West, the Y-axis runs North-South, and the Z-axis vertically.

In addition to the X, Y, Z coordinate system, an *i*, *j*, *k* coordinate system is also used. The coordinates of the so-called 'Key Block' in the *i*, *j*, *k* system are (1,1,1). The *i* coordinate increases along the line of increasing X, the *j* coordinate increases along the line of increasing Y, and the *k* coordinate increases along the line of decreasing Z. The i, j, k coordinates of several of the corner blocks are shown in Figure 14.7.

The concept of the key block is very important to the construction of a block model. In plan, the key block is situated in the lower left (southwest corner) of the upper-most bench (Bench 1), (see Figure 14.7 and Figure 14.8). The key coordinates (K_x, K_y, K_z) define the center of this block. In the case shown, the center of the key block has coordinates (1600, 4550, 4125). Knowing (a) the key block center coordinates, (b) the dimensions of a block and (c) the chosen number of blocks along the X, Y and Z directions one can calculate the coordinates of the block centers for the remaining blocks in the model. This is exactly what the computer program does, so it is important that the user carefully select the key block. Furthermore, knowing the block center elevation and the block height, the top and bottom elevations at each bench are determined. These elevations are used when calculating and assigning the composite grades to each block. The elevation of the top surface of the uppermost block is the elevation of the key block plus half the block height. This is the starting elevation used for making bench composites. This is discussed in more detail in Section 14.9.

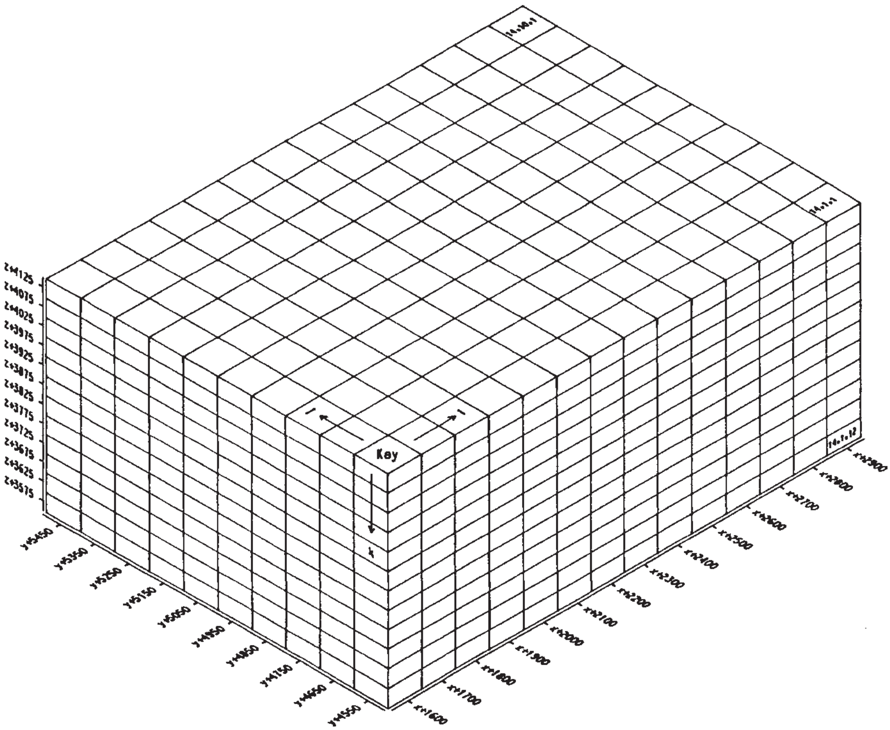


Figure 14.7. Isometric view of block model showing key block, the X, Y, Z and *i*, *j*, *k*, coordinate systems.

After studying the layout of the 'Block Values by Inverse Distance Squared' dialog box, return to the Drill Hole mode by the following steps:

SELECT: the 'Cancel' button {to exit the IDS dialog box}

SELECT: the 'Mode' command

From the drop down menu:

SELECT: the 'Drill Hole' command

The program should now be in the Drill Hole mode and the data window filled with the drill hole definition records.

14.8.3 *Creating a drill hole plan map*

Next the steps required to create a drill hole plan map are covered. From the main 'Drill Hole Mode' menu:

SELECT: the 'Plan Map' command

A plan map will be displayed using the control variables that were stored in the 'Tutor.kon' project file. To change the 'Plan Map' display control variables:

SELECT: the 'Configure' command

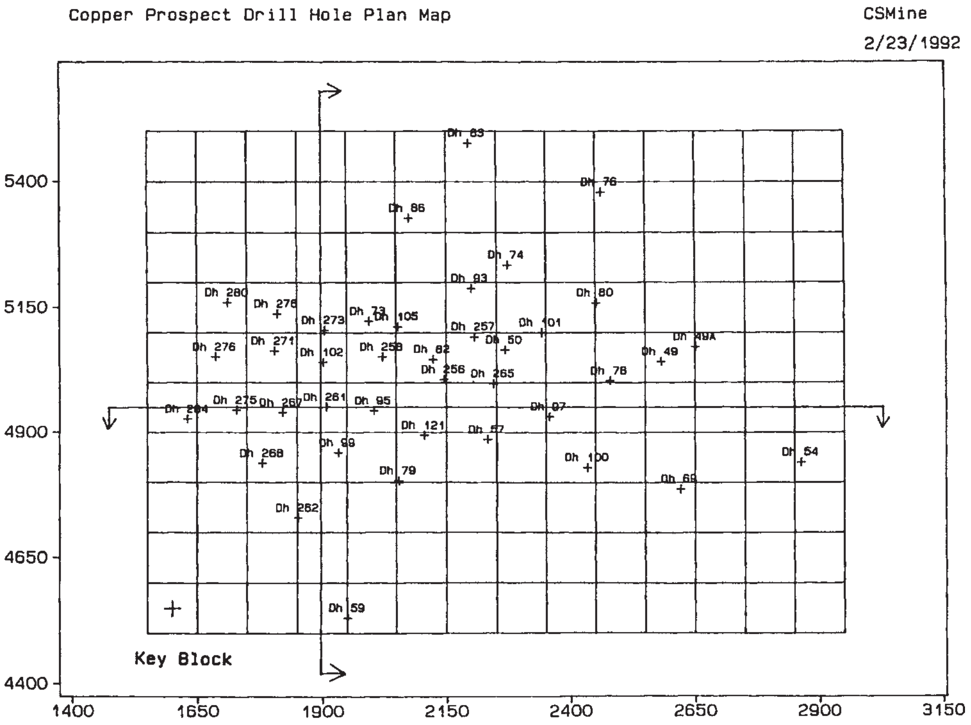
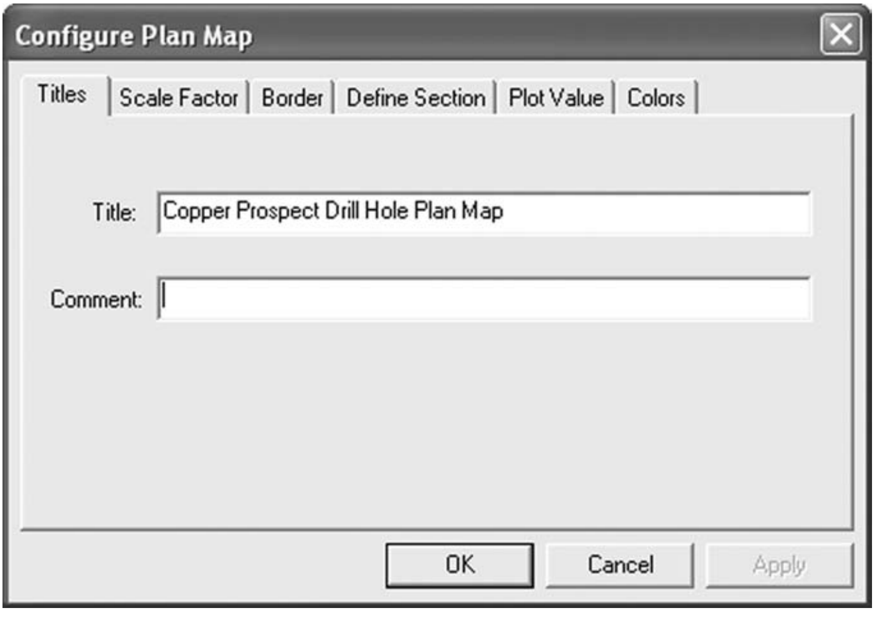
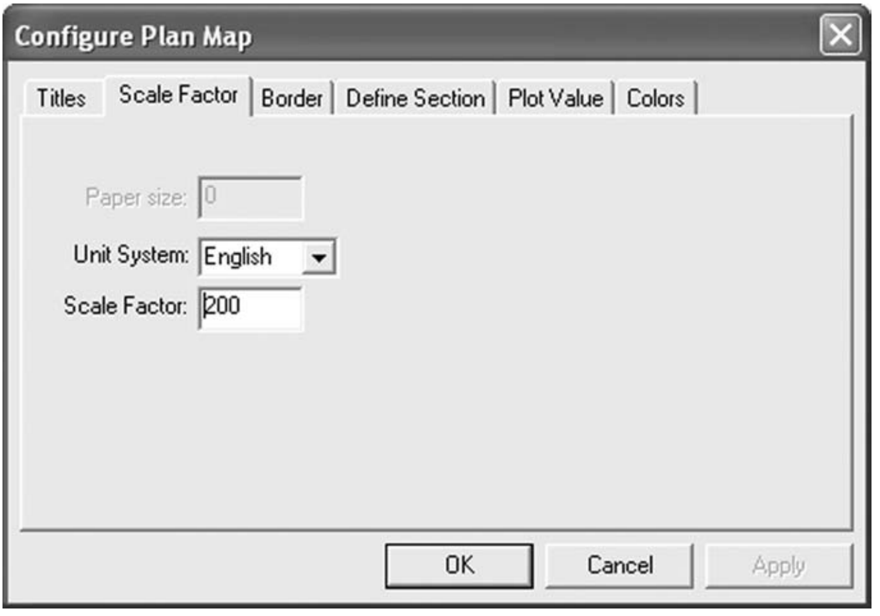


Figure 14.8. Drill hole plan map showing location of key block and location of vertical sections.

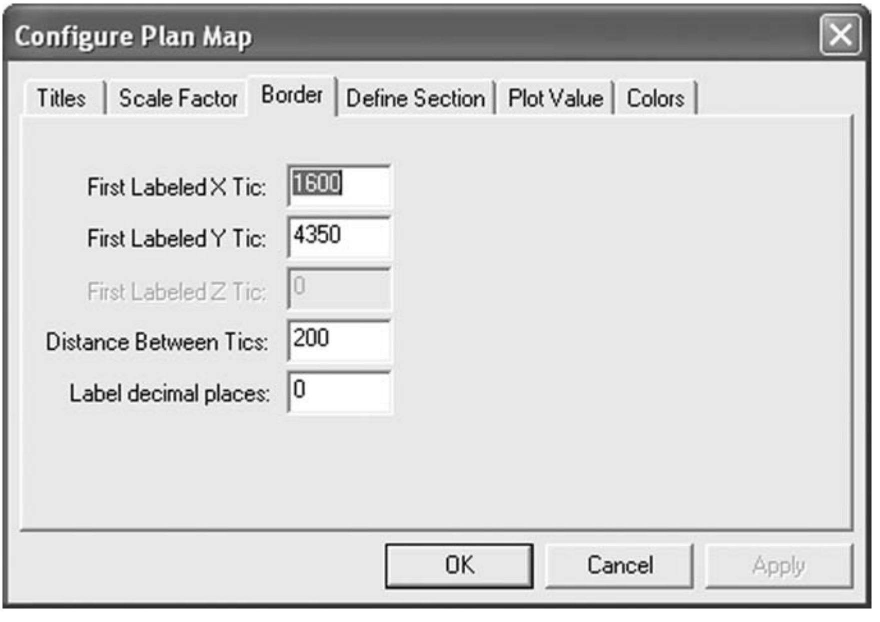
The ‘Configure Plan Map’ dialog box will then be displayed.



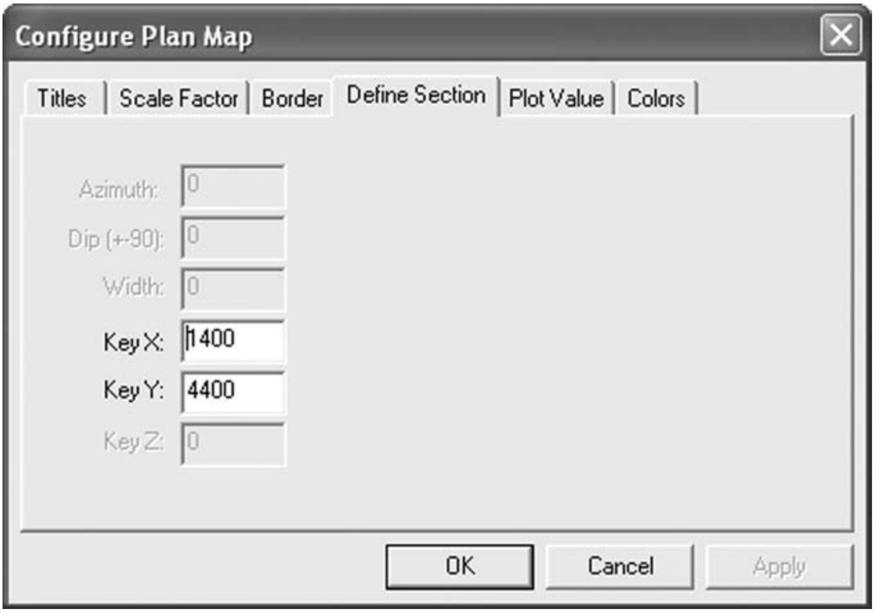
The dialog box consists of six tabs. All dialog boxes for plotting in CSMine have the same basic layout. The 'Titles' tab is used for entering the map title and comment. The 'Scale Factor' tab is used for entering the map scale factor and unit system (English or Metric Units).



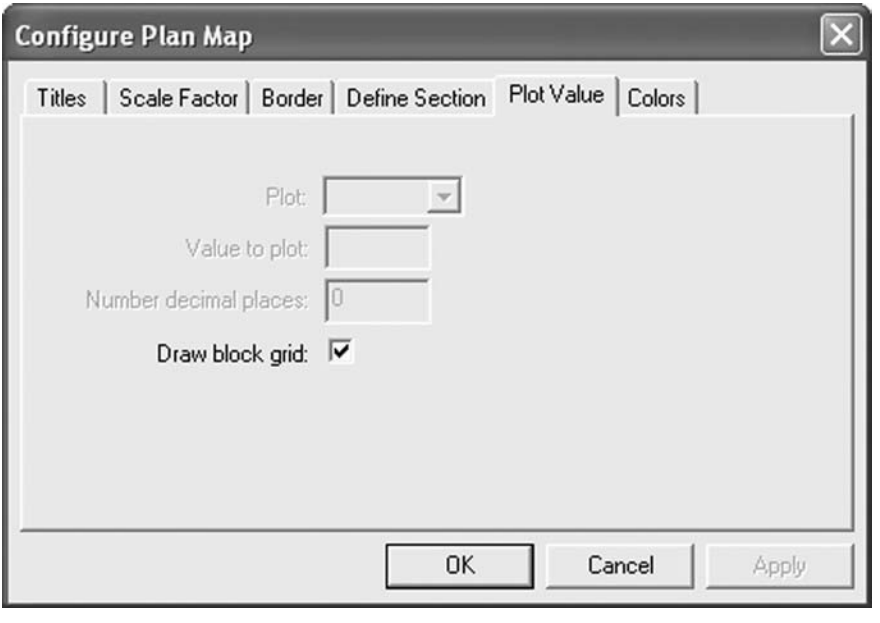
The 'Border' tab is used for entering the first labeled X and Y axis tics, the distance between labeled tics, and the number of decimal places to use with the labels.



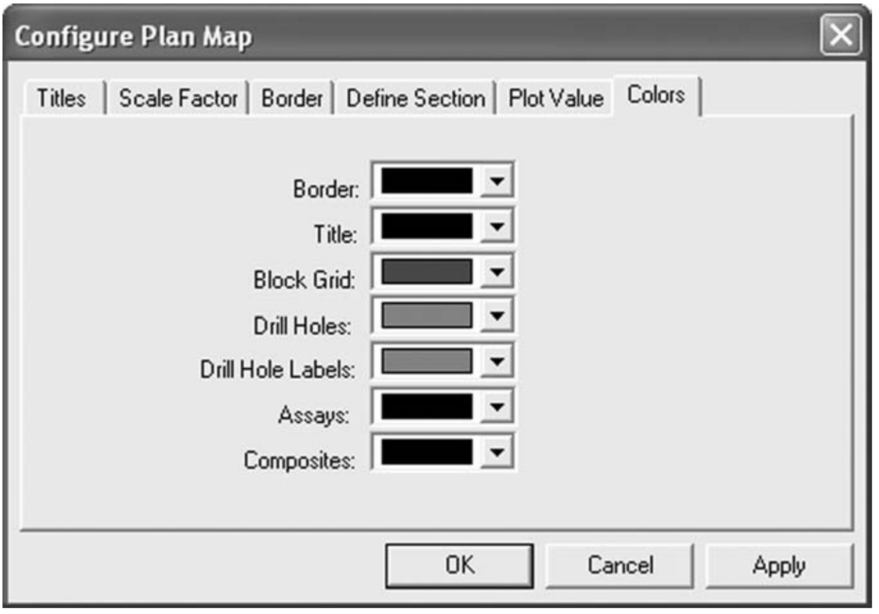
The 'Define Section' tab is used for defining the X and Y coordinates of the lower left corner of the plan map.



The 'Plot Value' tab is used to indicate whether the block model grid is to be drawn on the plan map.



The 'Colors' tab is used to indicate the color to use with the various items displayed graphically.



Once any dialog box item has been changed, the 'Apply' button is activated. Pushing the 'Apply' button will update the displayed plan map using the current dialog box values, and the dialog box will remain open. Any dialog values that were changed will be stored and used the next time the dialog box is opened. Pushing the 'OK' button will also update the displayed plan map and store the dialog box values, however the dialog box will be closed. The 'Cancel' button is used to close the dialog box and restore any dialog values that were modified to their previous values.

To print the currently displayed plan map:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Print' command

The standard Windows print dialog will be opened from which the user may select a printer for output and configure the print out according to the options available for the selected printer. Figure 14.8 shows a copy of the Drill Hole Plan Map produced using a Hewlett Packard Deskjet 932C printer. Also shown on the figure is the location of the key block as discussed in the previous section. Lines showing the locations of the vertical sections that will be created in the following section have also been superimposed.

To exit the 'Drill Hole Plan Map' display and return to the Drill Hole mode:

SELECT: the 'Return' command

14.8.4 *Creating a drill hole section map*

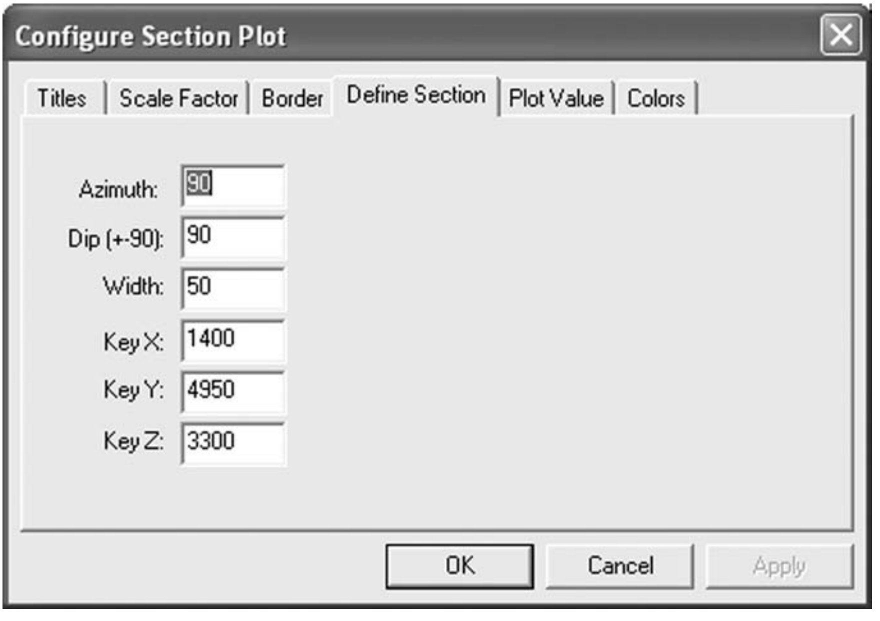
In this section, the steps necessary to create a drill hole section map are illustrated.
From the 'Drill Hole Mode':

SELECT: the 'Section Map' command

A section map will be displayed using the control variables that were stored in the 'Tutor.kon' project file. To change the 'Section Map' display control variables:

SELECT: the 'Configure' command

The 'Configure Plan Map' dialog box will then be displayed. The dialog box tabs and most of the dialog items are the same as those used with the Plan Map as described in the previous section. The particular dialog items that are used to control the display of a drill hole section map are described here. The 'Plot Value' tab is used to specify if (1) the raw drill hole file assay values, (2) regularized composite values, or (3) nothing is to be plotted along each drill hole. The 'Define Section' tab controls the location of the section in three-dimensional space.



This is done by specifying the section 'Azimuth', 'Dip', 'Width' 'Key X', 'Key Y', and 'Key Z' coordinates. The 'Section' key coordinates must not be confused with the 'Block Model' key coordinates which are used to define the block model geometry. These are not the same thing! The 'Section' key coordinate (S_x, S_y, S_z) specifies the (X, Y, Z) coordinate of the lower left corner in three-dimensional space of the section that is to be made.

The section 'Azimuth' defines the bearing of the section that will pass through the section 'Key' coordinate. The section 'Width' then specifies the section corridor. The section thickness is twice the section width. Drill holes which fall within one section width on either side of the true section plane will be displayed. The section 'Dip' is used to control only the direction of the labeling of the vertical axis. Only vertical sections may be plotted with CSMine, but the Z-axis can be plotted with the z-coordinate increasing in the upward direction ('Dip' +90) or with the z-coordinate increasing in the downward direction ('Dip' −90). In open pit mining the Z-axis is usually an elevation and thus positive up. In underground mining the Z-axis is usually a depth below the surface and thus positive down.

Figure 14.9 shows the drill hole section plot for the initial 'Configure Section Plot' dialog box values. The location of the section is also shown on the drill hole plan map, (Figure 14.8). Study these two figures and try to locate the Section key coordinates on the figures.

A section which runs North-South will be created next. The 'Define Section' tab with the new values is shown below. Specifically, the 'Azimuth' must be changed to 0, the section 'Width' to 50.0, and the section 'Key' coordinates to (1900, 4400, 3400). Also the 'Scale Factor' should be changed from 200 to 150 using the 'Scale Factor' tab, the 'First Labeled Y Tic' changed to 4500 using the 'Border' tab, and the 'Comment' should be changed to 'Section East 1900' using the 'Titles' tab. After making these changes, press the 'OK' button, and the section plot shown in Figure 14.10 will be displayed.

To exit the 'Drill Hole Section Map' display and return to the Drill Hole mode:

SELECT: the 'Return' command

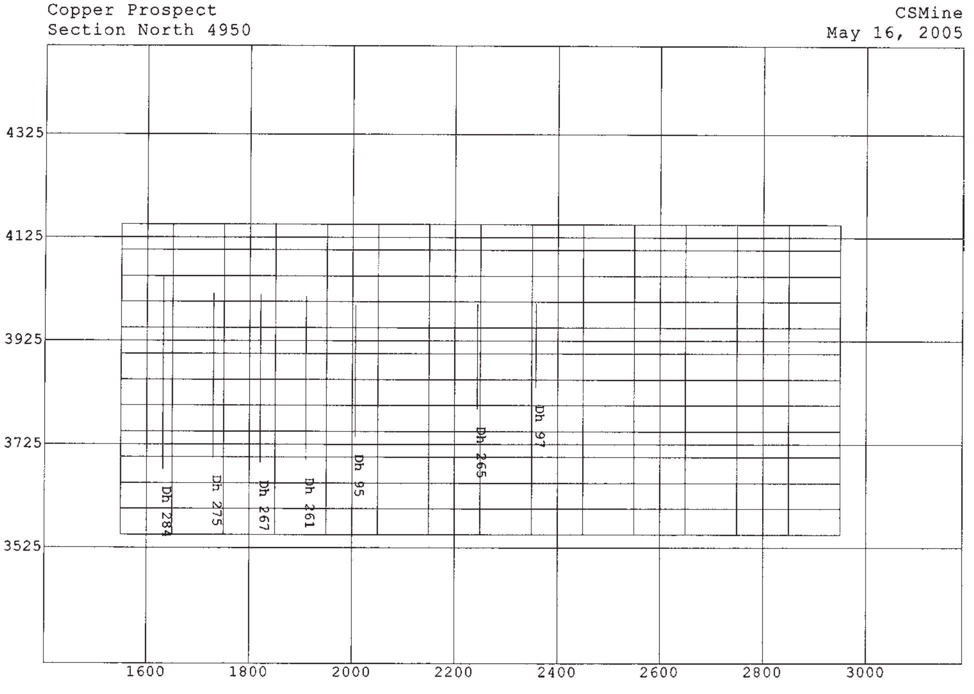
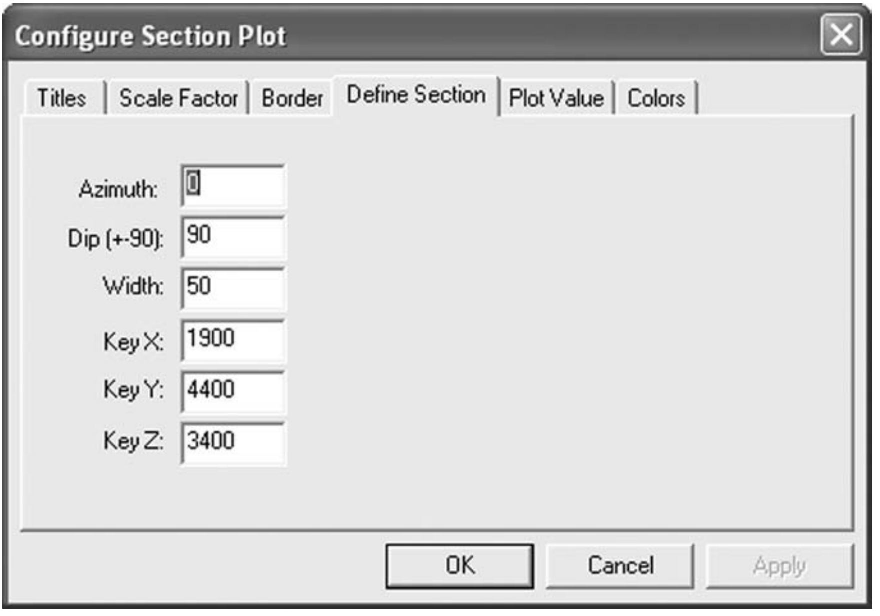


Figure 14.9. East-West drill hole section map.



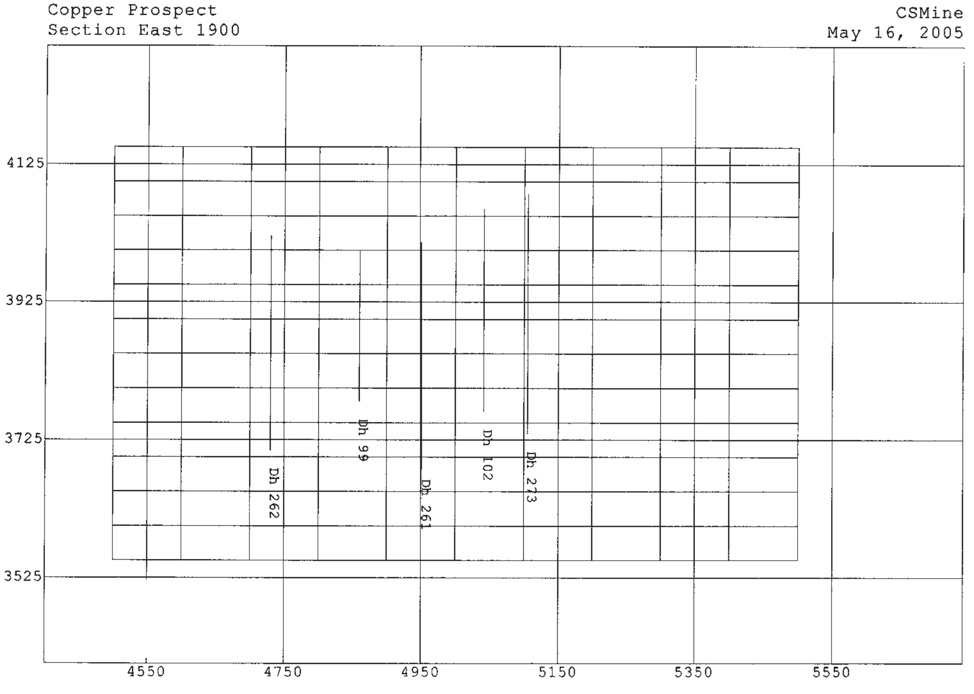


Figure 14.10. North-South drill hole section map.

14.9 THE COMPOSITE MODE

Before a block model can be created, the raw drill hole assay data must be regularized into composites of equal length. The Composite mode provides the program commands for dealing with regularized composites. The basic functions provided are:

1) calculate composites from the drill hole data,
2) plot drill hole section maps showing composite values,
3) store and load composite values from disk files.

14.9.1 *Calculating composites*

In order to be able to calculate composites, a drill hole file must have been read into the program's memory via the Drill Hole mode. If this has not been done, refer to tutorial Section 14.8.1 on reading a drill hole file and read the file 'ArizCu.dhf' at this time.

First CSMine must be put into the Composite mode. This is done using the 'Mode' command from the main menu. From the 'Drill Hole Mode' menu:

SELECT: the 'Mode' command

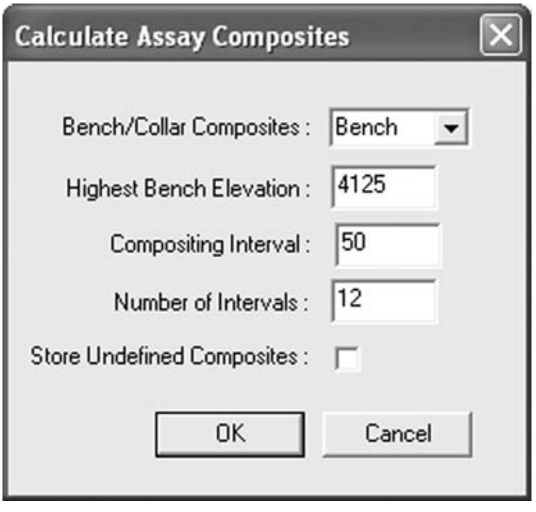
From the drop down menu:

SELECT: the 'Composite' command

The main data screen will then be cleared and the text 'Composite Mode' will be written across the top of the screen. Next the 'Calculate' command will be used to bring up the data entry screen for calculating the composites.

SELECT: the 'Calculate' command

The 'Calculate Assay Composites' dialog box will then be displayed.



Two types of compositing are available, bench or collar. Bench compositing has been selected which means that the drill holes are assumed to be vertical. Composites are then calculated such that the center coordinates of each interval fall at the same elevations as the center coordinates of the blocks for the block model. For a description of how collar composites are used with angled drill holes in the block model see Section 15.4.3.2.

If a composite is undefined, i.e. there are no assay values available in the corresponding section of the drill hole, CSMine assigns a value of −1 to the composite. The option to 'Store Undefined Composites' has been turned off, as there is no reason to do so with the tutorial example. The 'Highest Bench Elevation' is set to 4125.00, which is the same elevation as the key Z block coordinate value defined earlier via the Block Model mode. The 'Composite Interval' is 50 feet, which is the same as the block height selected. The 'Number of Intervals' has been set to 12, which is the same as the number of blocks in the Z direction. Thus for each drill hole, the raw assay data will be regularized to form composites 50 feet in length whose centers fall at the same elevation as the centers of the blocks in the specified block model.

To begin the calculations:

SELECT: the 'OK' button

The Composite Mode data screen showing the calculated composite values will be displayed when the calculations have been completed.

14.9.2 *Storing and loading composite files*

Writing a composite file is done in almost the same manner as reading a drill hole file as outlined in Section 14.8.1. From the 'Composite Mode' menu:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Save As' command

The standard Windows file dialog box will appear, from which the user may select an existing composite file with the extension '.cmp', or enter the name of a new file to write the composites to.

The reading of an existing composite file is done in a similar manner by first selecting the 'File' command, and then selecting the 'Open' command from the drop down menu, after which an existing composite file can be selected using the standard Windows file dialog box.

To return to the Drill Hole mode:

SELECT: the 'Mode' command

From the drop down menu:

SELECT: the 'Drill Hole' command

14.9.3 *Drill hole section plots with composites*

The procedure for creating a drill hole map with composite values is the same as described earlier in the section covering the Drill Hole mode. The only difference is that now composite values may be plotted along the drill holes. From the 'Drill Hole Mode' menu:

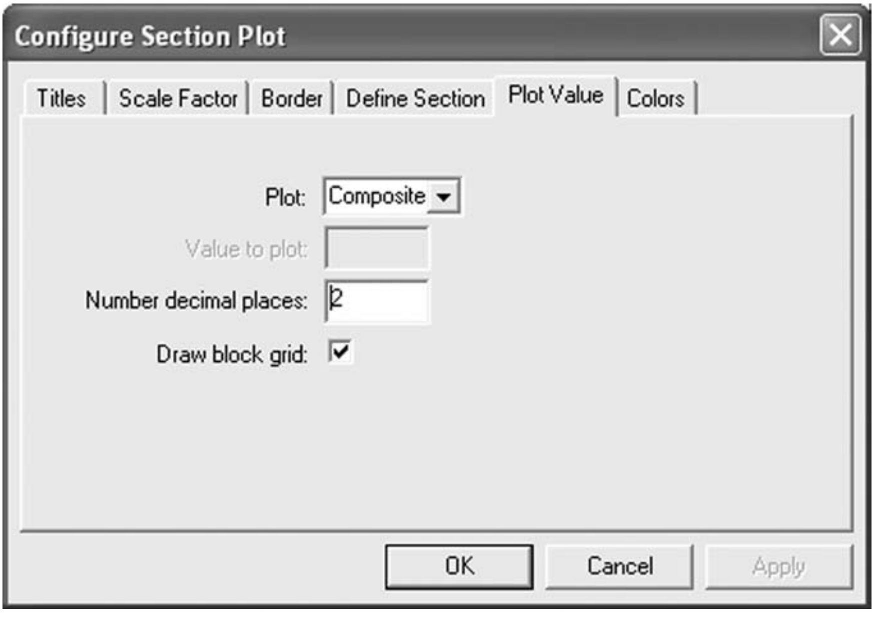
SELECT: the 'Section Map' command

A section map will be displayed using the current control variable settings. To reset the values to those stored in the tutorial project file, read the project file 'Tutor.kon' as described in Section 14.7.

To change the 'Section Map' display control variables:

SELECT: the 'Configure' command

The 'Configure Section Map' dialog box will then be displayed. The dialog box tabs and most of the dialog items are the same as those used with the Plan Map and Section Map as described in Section 14.8.3 and 14.8.4. The 'Plot Value' tab is shown.



In order to plot the composite values on the section, change the value in the 'Plot' field from 'Nothing' to 'Composite' using the drop down menu. Push the 'OK' button to update the section plot and close the dialog box. Figure 14.11 shows the section North 4950 with composite values displayed.

To exit the 'Drill Hole Section Map' display and return to the Drill Hole mode:

SELECT: the 'Return' command

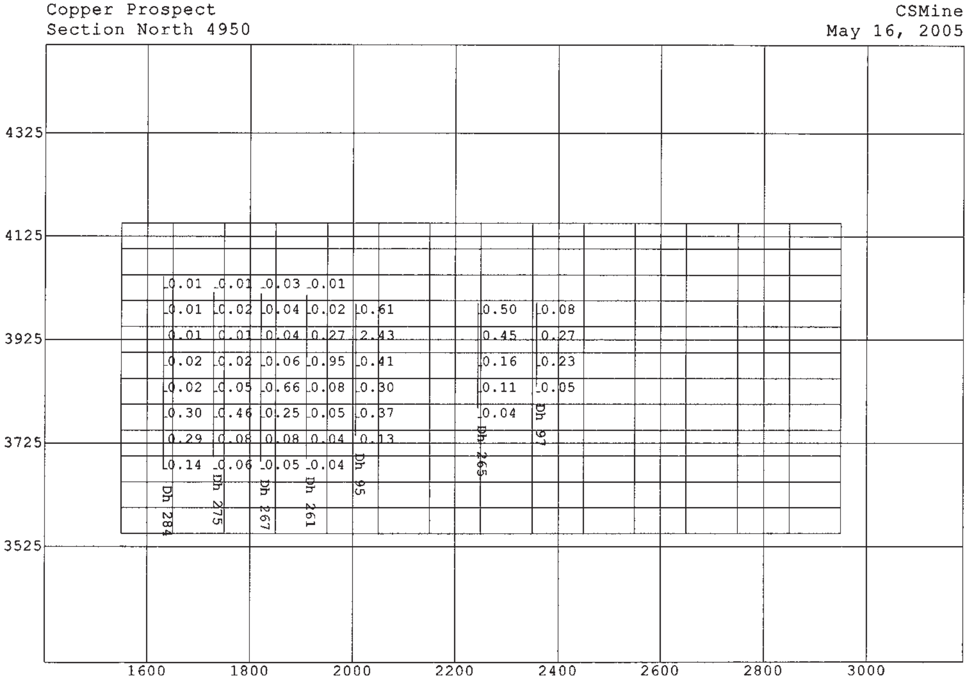


Figure 14.11. Section showing composite values.

14.10 THE BLOCK MODE

The Block mode is used to create the block model of grades for the deposit and to display the results in bench and section plots. Additional commands allow for assigning economic values to the blocks and for calculating the final economic pit limits.

Before block grades can be calculated, composite values must be present. Composites can be read from a disk file or calculated from a drill hole file. If composites are not currently present in the program's memory, then refer to the previous section and create the necessary composites before proceeding.

14.10.1 *Calculating block grades*

First, CSMine must be put into the Block mode. This is done using the 'Mode' command. From the 'Drill Hole Mode' or 'Composite Mode' menu:

SELECT: the 'Mode' command

From the drop down menu:

SELECT: the 'Block' command

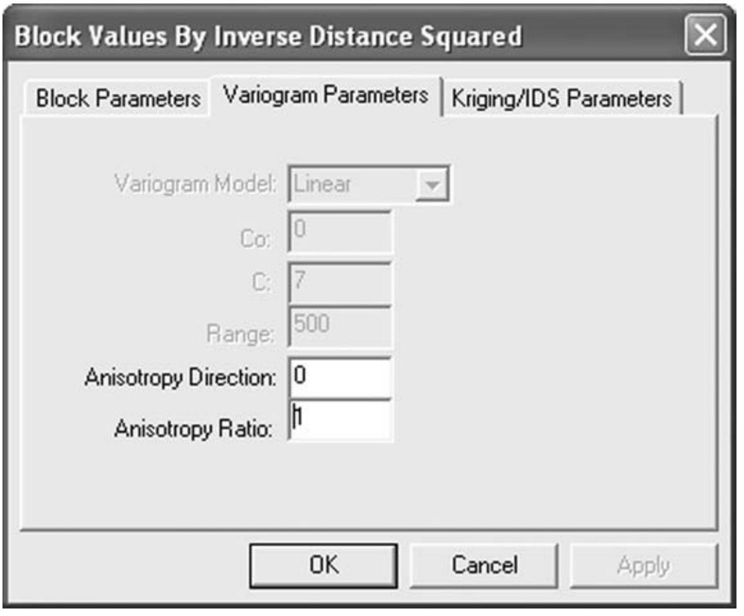
The main data screen will then be cleared and the text 'Block Mode' will be written across the top of the screen. Next the 'Calculate' command will be used to bring up the dialog box for calculating the block grades.

SELECT: the 'Calculate' command

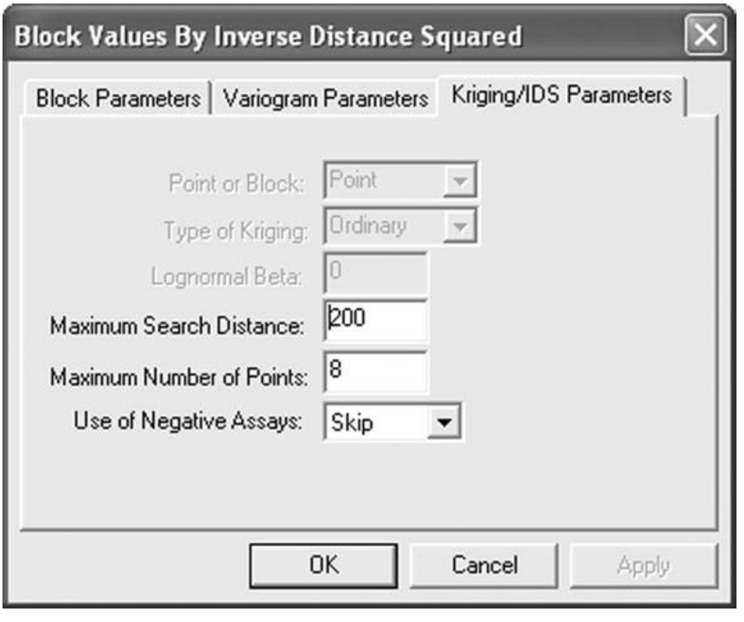
From the drop down menu:

SELECT: the 'Values by IDS' command

The 'Block Values By Inverse Distance Squared' dialog box will then be displayed.



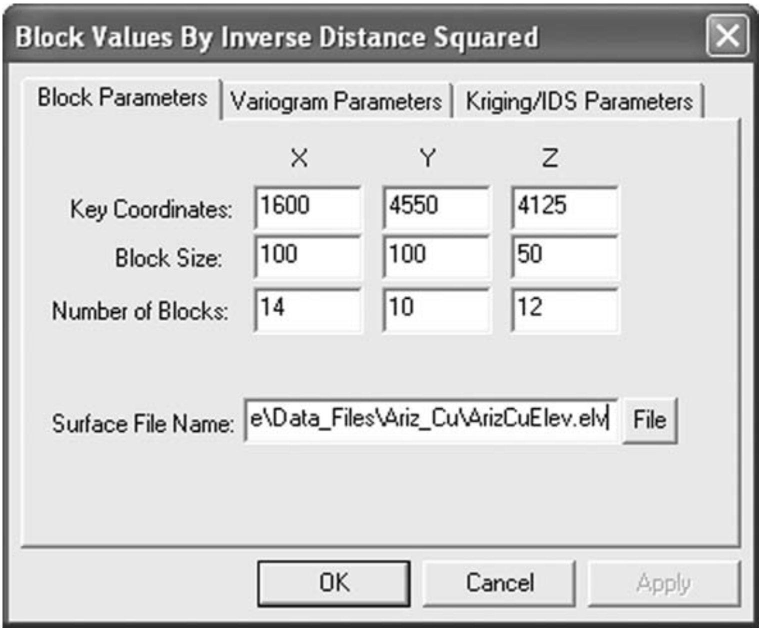
The same input screen is also used when kriging is selected for assigning the block grades. The menu items that do not apply to Inverse Distance Squared block grade estimation are



displayed with a dim color, while those menu items that do apply are displayed with a bright screen color. A complete discussion of the Inverse Distance Squared estimation technique is given in Chapter 15.

Under the 'Variogram Parameters' heading, the 'Anisotropy Direction' and 'Anisotropy Ratio' can be set to control the shape of the search ellipse used to locate the composites that influence a block's grade. In this case an 'Anisotropy Ratio' of 1 has been entered to indicate a circular search window.

Under the 'Kriging/IDS Parameters' heading, the 'Max Search Distance' defines the radius of the search circle. In this case, all composites that fall within 200 feet of the center of a block can then be used to estimate the grade of that block. The 'Maximum Number of Points' value of 8 means that only the eight closest composites to the block's center will be used in the calculation. Finally the 'Use of Negative Assays' has been set to 'Skip', instructing the program to skip negative assays, as −1 indicates that the assay is undefined.



Under the 'Block Parameters' heading, the 'Key Coordinates' (K_x, K_y, K_z) define the (x, y, z) coordinates of the center of the key block as was shown in Figure 14.7. Please refer to this figure. The 'Block Size' values (B_x, B_y, B_z) specify the size of the blocks in the X, Y, Z directions and the 'Number of Blocks' (N_x, N_y, N_z) defines the total number of blocks for which a grade is to be assigned in each direction. The 'Surface Topography Definition File' is the line for entering the name of the file that contains the surface topography information. Press the 'File' button to bring up the standard Windows file dialog that can be used for selecting the surface topography file name. A contour map of the original surface topography with the selected block grid overlain was shown previously in Figure 14.4. Please refer to this figure. The surface topography file contains the elevation of the surface at the center of each block in plan view shown in Figure 14.4. Shown in Figure 14.12 are the surface elevations for

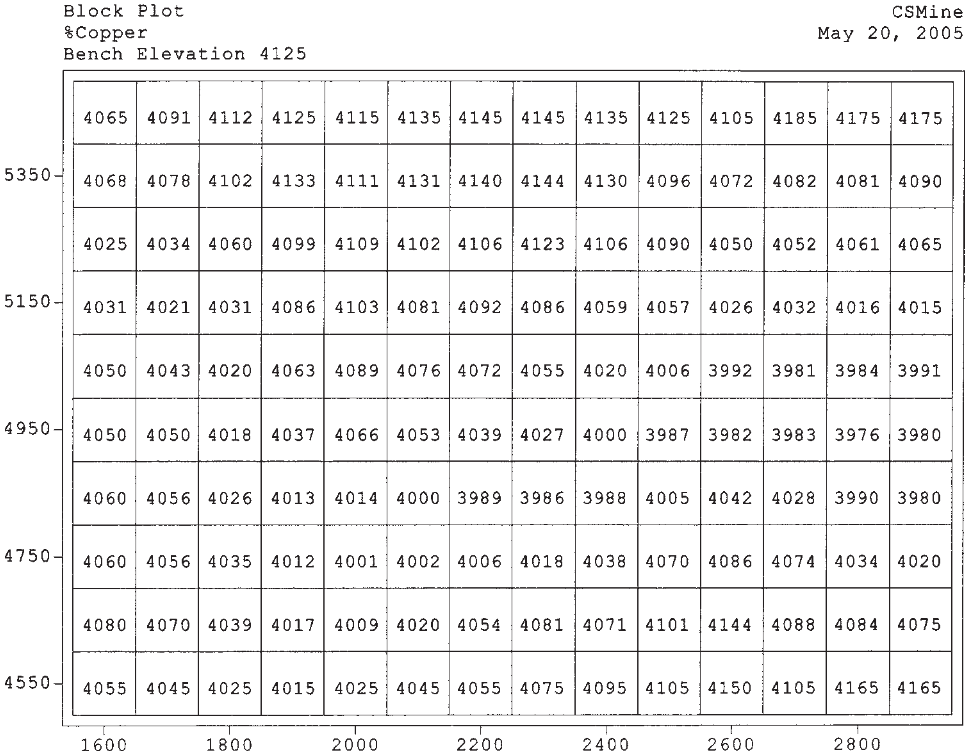


Figure 14.12. Block plot of surface topography file 'ArizCuElev.elv'.

each block contained in the file 'ArizCuElev.elv'. Figure 14.13 shows a contour map of the surface topography data that can be compared to the original map shown in Figure 14.4.

To begin calculating the block grades:

SELECT: the 'OK' button

The total number of blocks for grade assignment followed by the number of the block currently being assigned will be displayed at the top of the screen. If for some reason, the file 'ArizCuElev.elv' was not copied from the distribution disk to the working disk then following the error message will appear:

Error: Can't open surface topography file.

If this message is displayed, then copy the file 'ArizCuElev.elv' from the distribution disk to the working disk and repeat the steps outlined above. It is not necessary to have a surface elevation file when generating the grades for the blocks. However, if one indicates that there is a file, in this case 'ArizCuElev.elv', and it is not available the error message appears. If the surface elevation file has not yet been created, and this is often the case when first evaluating different block sizes and block model locations, simply leave the 'Surface File Name' line blank. When you have decided upon the block/block model geometry, then create this file and enter the name so that the best analysis is made. You can try this with the Arizona Copper data set.

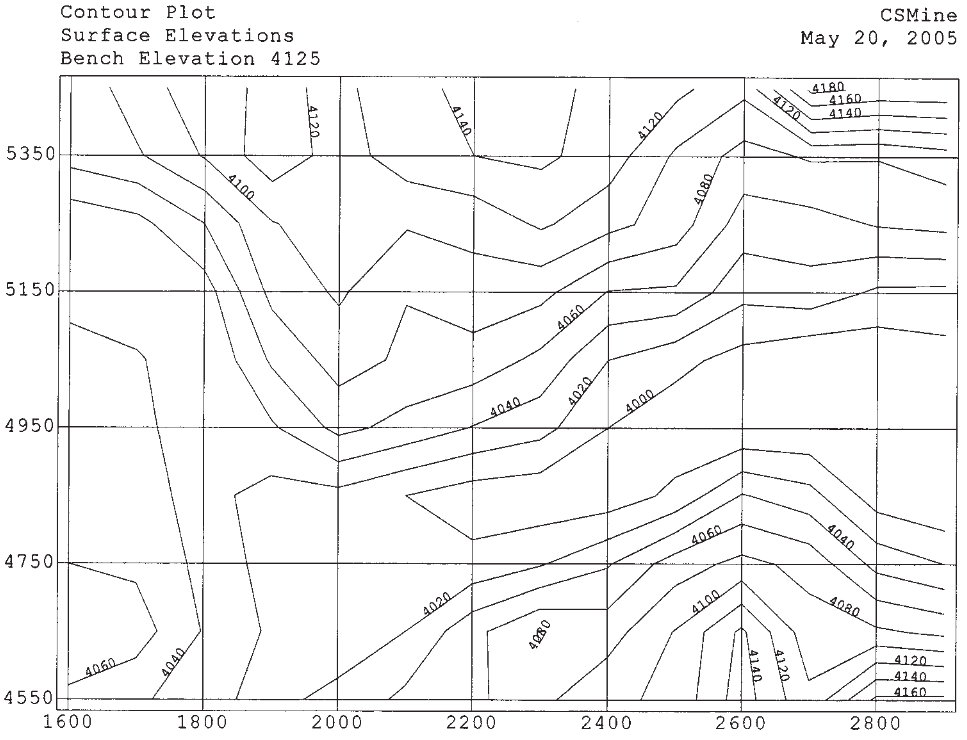


Figure 14.13. Contour map of surface topography file 'ArizCuElev.elv'.

After completing the calculations, the 'Block Values by Inverse Distance Squared' dialog box will be cleared and the block values displayed in the data window. The main data screen will display the *i*, *j*, *k* block index for each block (see Figure 14.7) followed by the block value and the number of composites used to estimate each block value. Blocks that are above the surface, i.e. air blocks, are given a value of −2.0. Blocks for which no composites could be found within the search ellipse are undefined and are given a value of −1.0.

14.10.2 *Creating block value plots*

In this section, the commands necessary for creating plots showing the block values in plan or section are covered. From the 'Block Mode' menu:

SELECT: the 'Plot' command

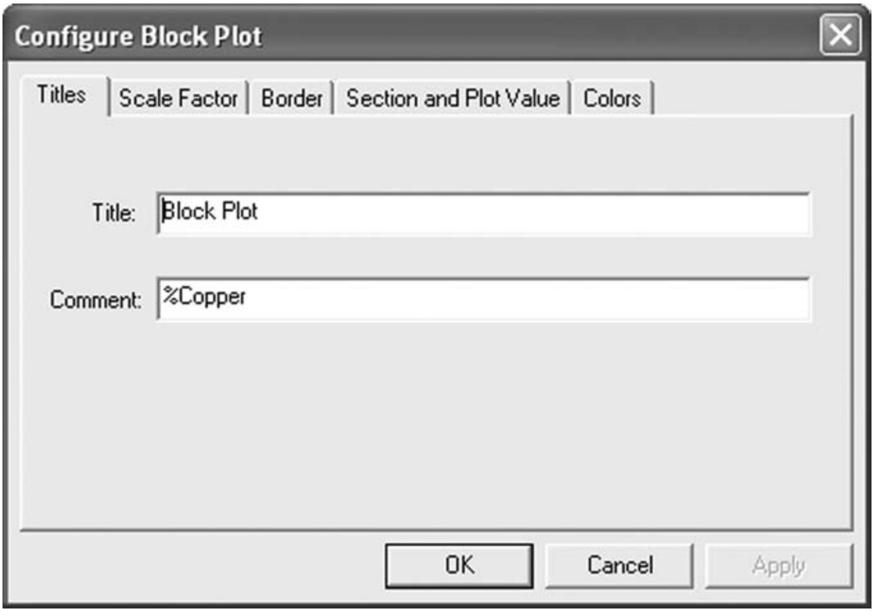
From the drop down menu

SELECT: the 'Block Plot' command

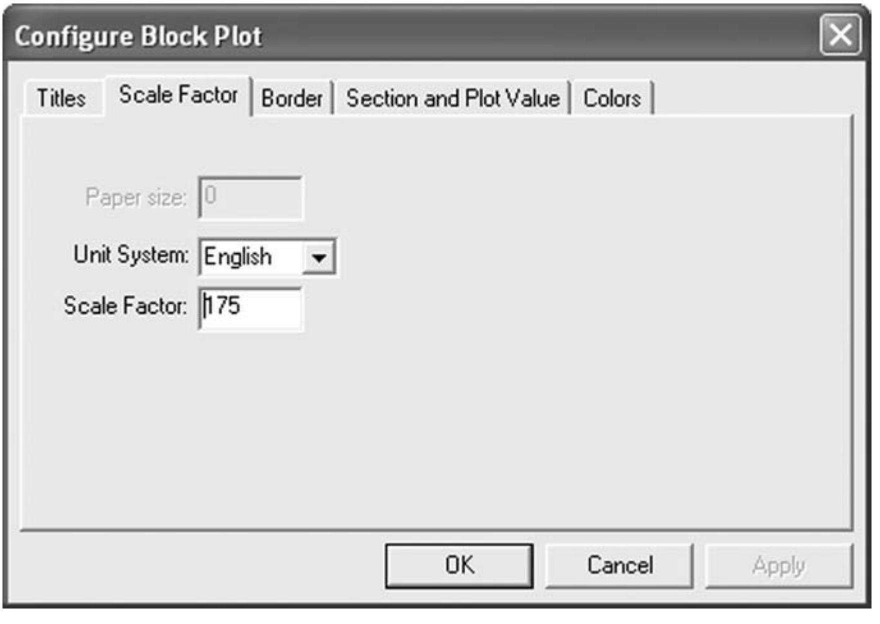
A block plot will be displayed using the control variables that were stored in the 'Tutor.kon' project file. To change the 'Block Plot' display control variables:

SELECT: the 'Configure' command

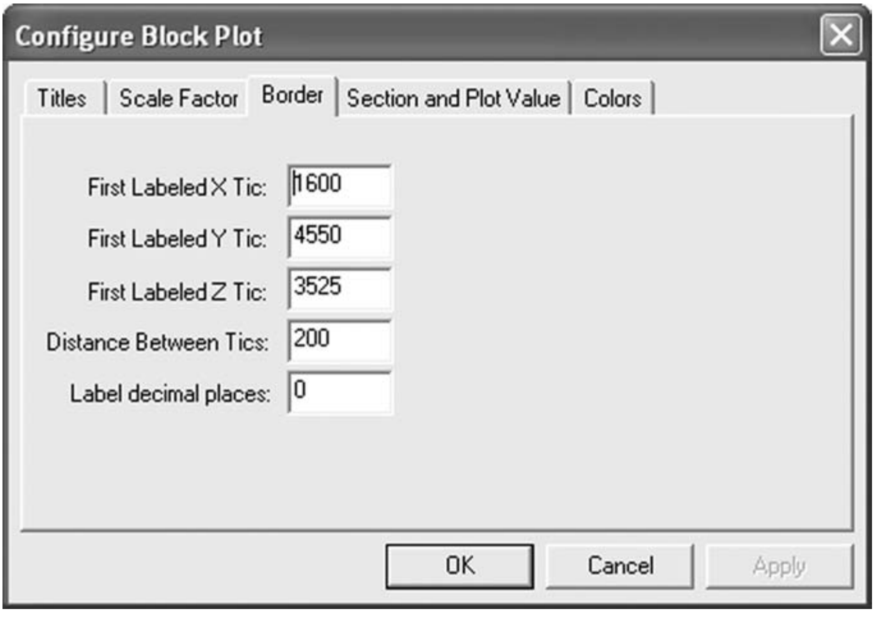
The 'Configure Block Plot' dialog box will then be displayed as shown.



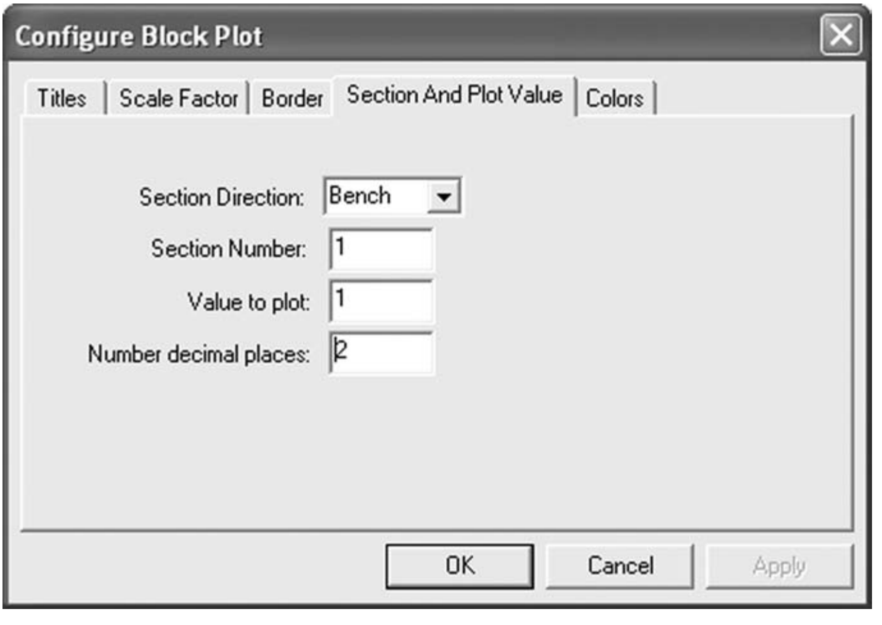
The 'Configure Block Plot' dialog box is set up similarly to the dialog boxes for the drill hole and section maps. The 'Titles' tab is for entering the map title and comment. The 'Scale Factor' tab is used for entering the scale factor and for indicating whether the data being plotted are defined using metric or English units.



The ‘Border’ tab is used for entering values to control the border labeling.



The ‘Section and Plot Value’ tab contains the options that control which bench or section to plot, which data value to plot, and how to display the block values. The ‘Section Direction’ is used to select Bench, North, or East sections for plotting. The ‘Section Number’ determines



which bench or section to plot. The value here corresponds to the *i*, *j*, *k* coordinate system illustrated previously in Figure 14.7. Bench 1 corresponds to the highest bench, which has a midpoint elevation of 4125. Since most of the blocks on this bench are above the topographic surface, a more interesting bench will be selected for plotting. Change the 'Section Number' to 4. Bench 4 has an elevation of 3975.
To view the plot for bench 4:

SELECT: the 'OK' button

The block plot shown in Figure 14.14 will be displayed.

An alternative method for selecting the bench or section to be displayed is by selecting the 'Next' command to change the display to the next bench or section, or by selecting the 'Previous' command to change the display to the previous bench or section. These commands are useful once the display control values have been set using the 'Configure Block Plot' dialog box and they eliminate the need to open the dialog box each time a different bench or section is to be displayed.

In addition to bench plots, views through vertical sections may also be plotted. North-South running sections are specified by giving their location along the X (or East) axis. East-West running sections are specified by giving their location along the Y (or North)

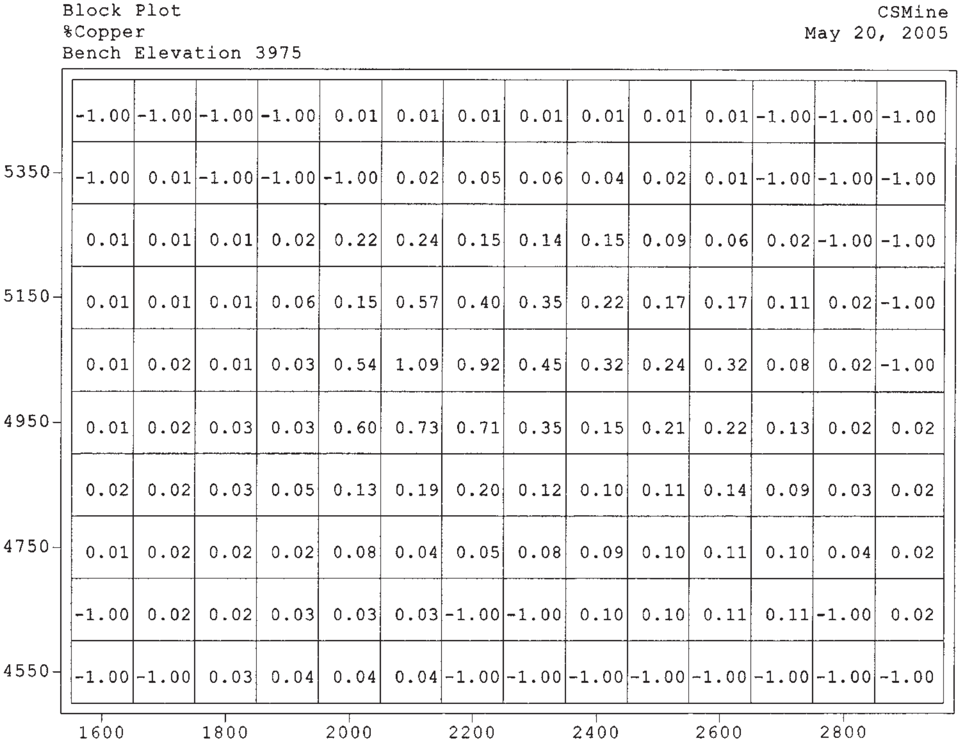


Figure 14.14. Block plot with block values written inside blocks.

axis. Although this sounds a little confusing, the reader will quickly get used to the notation. For example, to plot the East- West section with a constant northing coordinate of 4850 (see Figure 14.7):

SELECT: the 'Configure' command
SELECT: the 'Section and Plot Value' tab
SELECT: the 'Section Direction' field

From the drop down menu:

SELECT: 'North'

Finally, change the 'Section Number' to 4. This will configure the program to plot section North 4, which is a vertical section through the block model with a constant northing coordinate of 4850 (see Figure 14.7). To view the plot:

SELECT: the 'OK' button

The section shown in Figure 14.15 will then be displayed on the screen. As usual, anytime a graphics screen is displayed, a hard copy plot may be produced by selecting the 'File' command, from which the 'Print' command can then be selected from the drop down menu.

At this time you are encouraged to experiment with the other screen options that were not covered. Try changing the 'Number Decimal Places' line, which is used to control the format of the values written inside the blocks.

To return to the Block Mode main data window:

SELECT: the 'Return' command

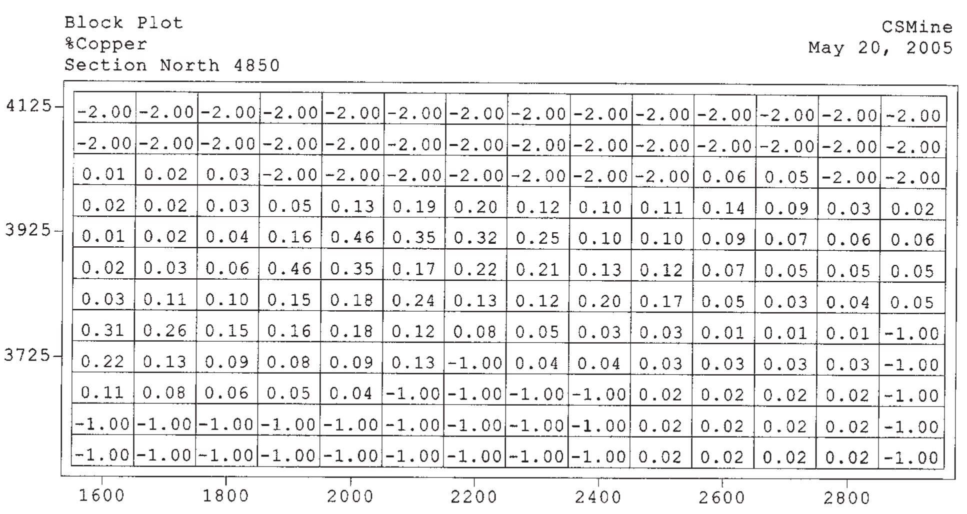


Figure 14.15. Vertical section North 4.

14.10.3 *Creating contour maps*

Another useful method for presenting block model data is with a contour map. In this section, creation of contour maps with CSMine is illustrated. From the 'Block Mode' menu:

SELECT: the 'Plot' command

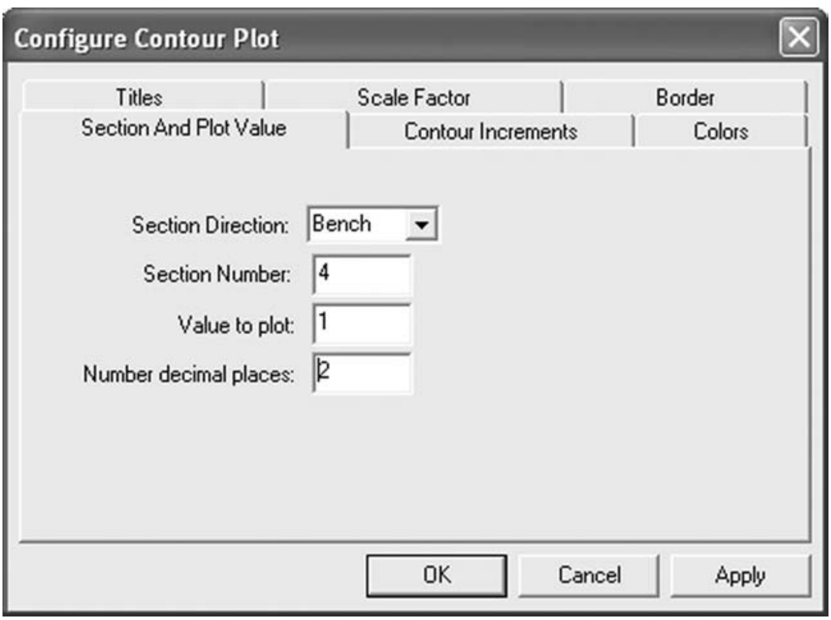
From the drop down menu:

SELECT: the 'Contour Plot' command

A contour plot will be displayed using the control variables that were stored in the 'Tutor.kon' project file. To change the 'Contour Plot' display control variables:

SELECT: the 'Configure' command

The 'Configure Contour Plot' dialog box will then be displayed. The dialog box layout is similar to the other Plotting menu screens, with the options specific to contour map plotting found under the 'Contour Increments' tab. As with the block plots, contour maps may be plotted through any bench or section. As was done with the previous block plot example, Bench 4 will first be selected for contouring. Under the 'Section and Plot Value' tab, insure that the 'Section Direction' indicates 'Bench', the 'Section Number' is set to 4, the 'Value to plot' is set to 1, and the 'Number of decimal places' is set to 2.

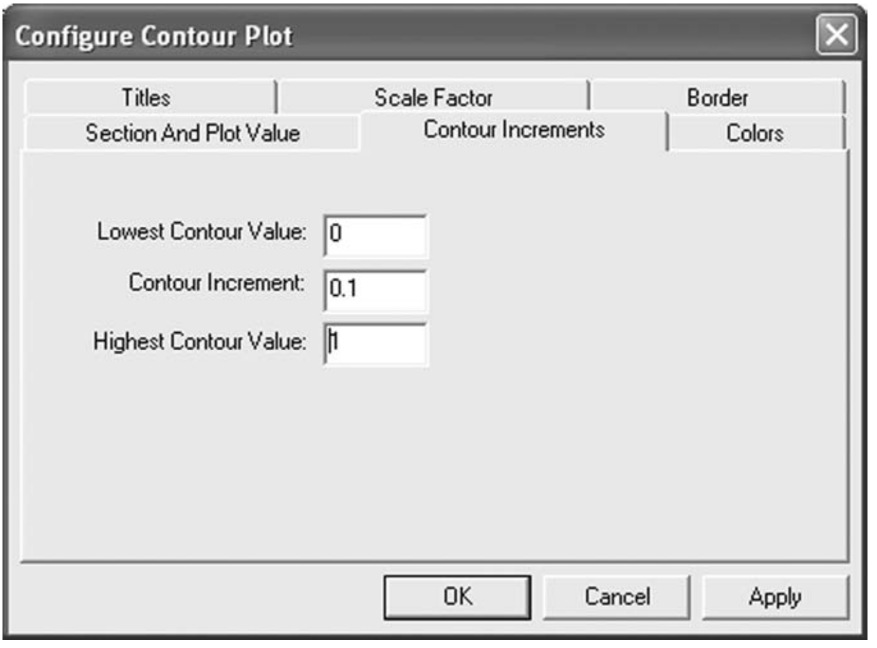


Using the 'Contour Increments' tab, set the 'Lowest Contour Value' to 0, the 'Contour Increment' to 0.1, and the 'Highest Contour Value' to 1.

To display the contour map as configured:

SELECT: the 'OK' button

The contour map shown in Figure 14.16 will then appear on the screen.



Contour maps may also be created through North or East vertical sections.

SELECT: the ‘Configure’ command
SELECT: the ‘Section and Plot Value’ tab
SELECT: the ‘Section Direction’ field

From the drop down menu:

SELECT: ‘North’

This will configure the program to plot section North 4, which is a vertical section through the block model with a constant northing coordinate of 4850. The same section was examined previously in the block plot section, see Figure 14.7 for reference. To view the plot:

SELECT: the ‘OK’ button

The screen should then appear as shown in Figure 14.17.

The screen colors may be set using the ‘Colors’ tab in the ‘Configure Contour Plot’ dialog box. A hard copy plot may be produced by selecting the ‘File’ command, from which the ‘Print’ command can then be selected from the drop down menu. The ‘Next’ and ‘Previous’ commands can be used to step the display through the available sections or benches for the current configuration.

At this time you are encouraged to experiment with the contour plotting options by changing the menu parameters to see what effect they have on the displaying of the contour maps. To exit the Contour Map screen:

SELECT: the ‘Return’ command

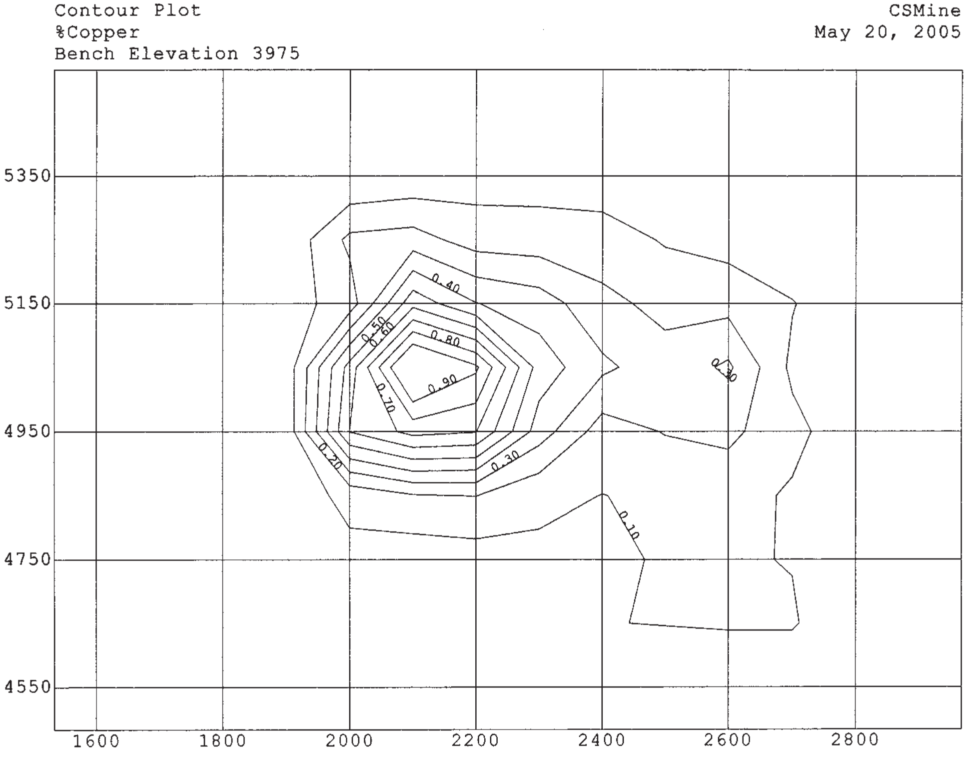


Figure 14.16. Contour map through Bench 4.

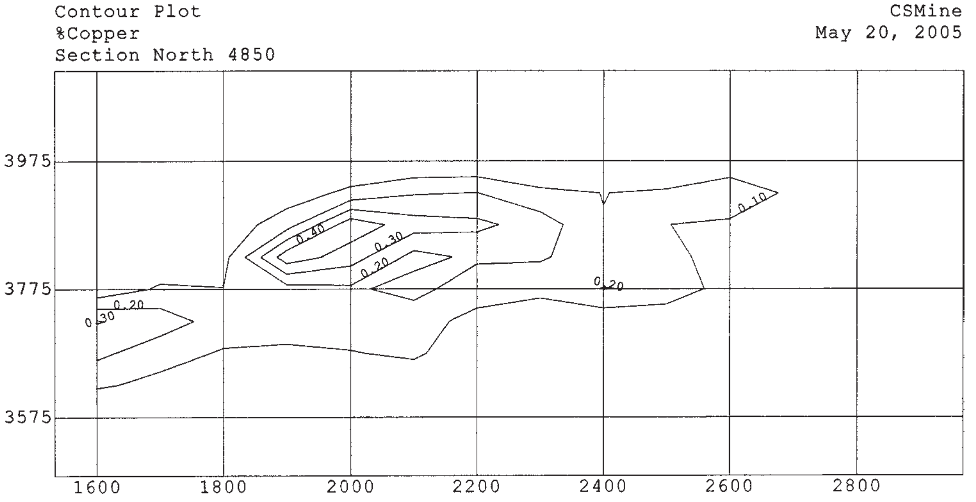


Figure 14.17. Contour map through section North 4.

14.10.4 *Assigning economic values to the blocks*

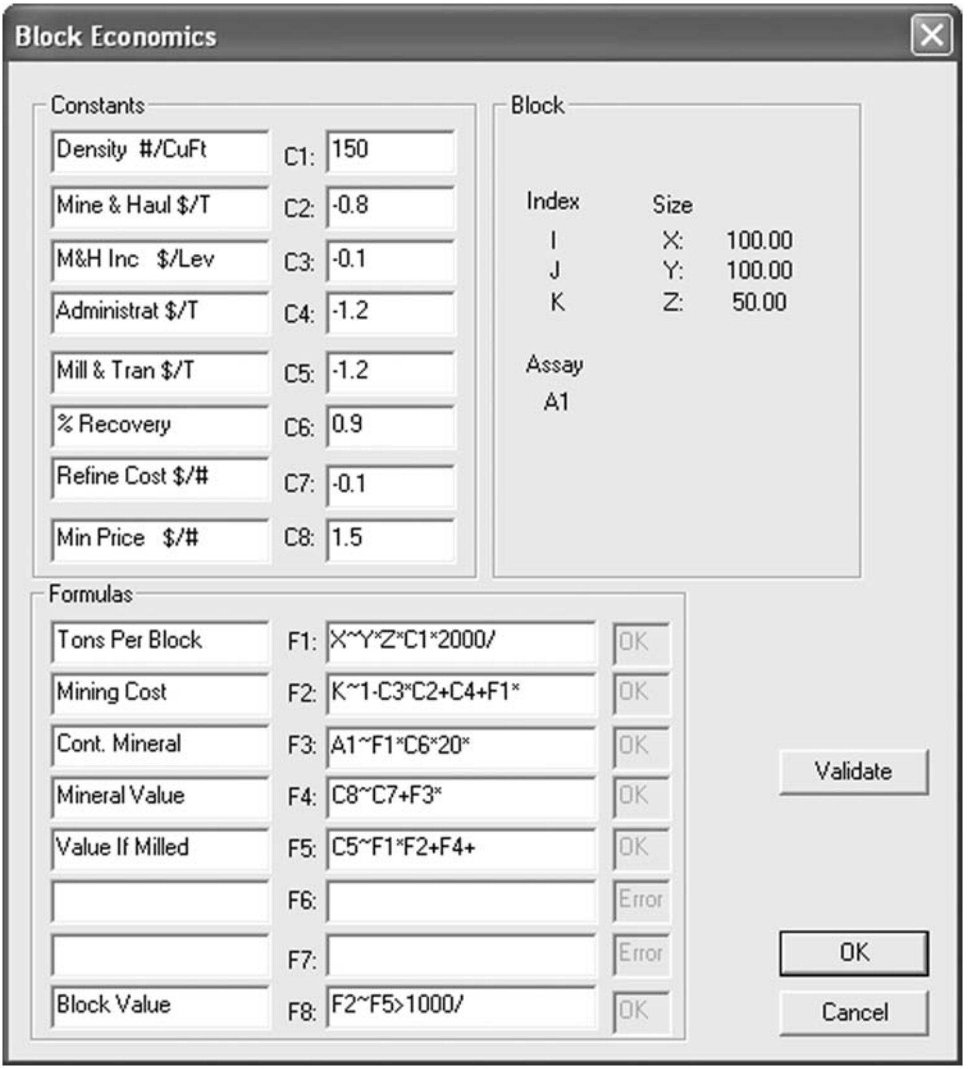
The next step in the modeling process is to use the block grades and a block's position in the deposit to assign an economic value to each block. The economic block values can then be used to plan the final economic pit limits. From the 'Block Mode' menu:

SELECT: the 'Calculate' command

From the drop down menu:

SELECT: the 'Economic Block Values' command

The 'Block Economics' dialog box will then be displayed.



While this dialog box may at first look quite formidable, assigning economic values to the blocks is actually quite simple. In the lower part of the screen (right side), up to eight formulae (Fl, . . . , F8) can be entered which specify how the calculations are to be performed. The lower part (left side) contains short names for the formulae identifying what calculations the formulae perform. The upper part (center) of the screen provides space for entering constants that can be used in the formulae, and the upper part (left side) contains labels for the constants. The upper part (right side) contains identifiers to use for the block size (X, Y, Z) and a block's *i*, *j*, *k* coordinate position.

The method used to specify formulae is called Reverse Polish Notation (RPN) and should be familiar to users of Hewlett Packard calculators. RPN notation allows entering any formula unambiguously without the use of parenthesis. As an example, formula Fl is used to calculate the tons per block and reads

X ∼ Y * Z * C1 * 2000/

The symbol '∼' is used to indicate hitting the <ENTER> key on a hand held calculator. In standard mathematical notation, this formula would be written as

Tons = (X * Y * Z * C1)/2000

Formulae have been entered for calculating:

F1: The tons per block;

F2: The mining cost per block. This includes a basic mining and transportation (C2) an increase in transportation cost with increasing depth of the pit (C3), and administration costs charged per block mined (C4);

F3: The amount of mineral contained in the block which includes mill recovery (C6);

F4: The mineral value contained in the block which includes the mineral price minus refining costs (C7);

F5: The value of the block if milled;

F8: The final value to assign to the block. This is the greater of (a) the mining cost or (b) the mining cost plus revenue if the block is mined and milled as ore. It is expressed in thousands of dollars.

This is a rather simple method for evaluating the economics of each block. More complex methods can be used by entering the required formulae.

A complete discussion of how to change the formulae as well as a more detailed discussion of the formulae shown here is given in Chapter 15. For now, the economic values will be calculated using the formulae provided.

To begin calculating economic block values

SELECT: the 'OK' button

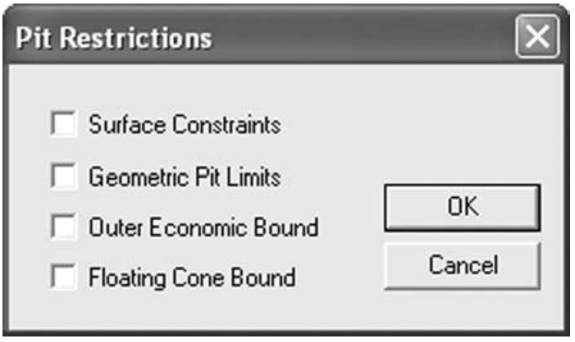
The 'Block Economics' dialog box will be closed, and when the calculations are complete, the main data window redisplayed.

14.10.5 *The Restrictions command*

The 'Restrictions' command is used to reduce the block model from the large three dimensional rectangle shown in Figure 14.7 to the set of blocks which meet various mining constraints. From the 'Block Mode' menu:

SELECT: the 'Restrictions' command

The 'Pit Restrictions' dialog box will then be displayed as shown.



As can be seen, there are four different options for restricting the block model. Each of these options is discussed in the order they appear. Each constraint will be turned on in order and the results will be viewed in a vertical block plot.

Surface constraints. A large number of blocks were assigned a grade of −2.0, which indicates that these blocks are above the surface. Turning on the 'Surface Constraints' will restrict the block model such that these blocks will not be displayed in the data window nor will they be shown in the various block plots. To turn the 'Surface Constraints' on, click the mouse in the small box next to the constraint name. A small check mark will appear indicating that this constraint is now turned on. To apply the 'Surface Constraint' restriction:

SELECT: the 'OK' button

The main data window display will be updated to show only those blocks that lie below the surface topography.

The restricted block model can now also be viewed using the Block Plot option by executing the following series of commands:

SELECT: the 'Plot' command

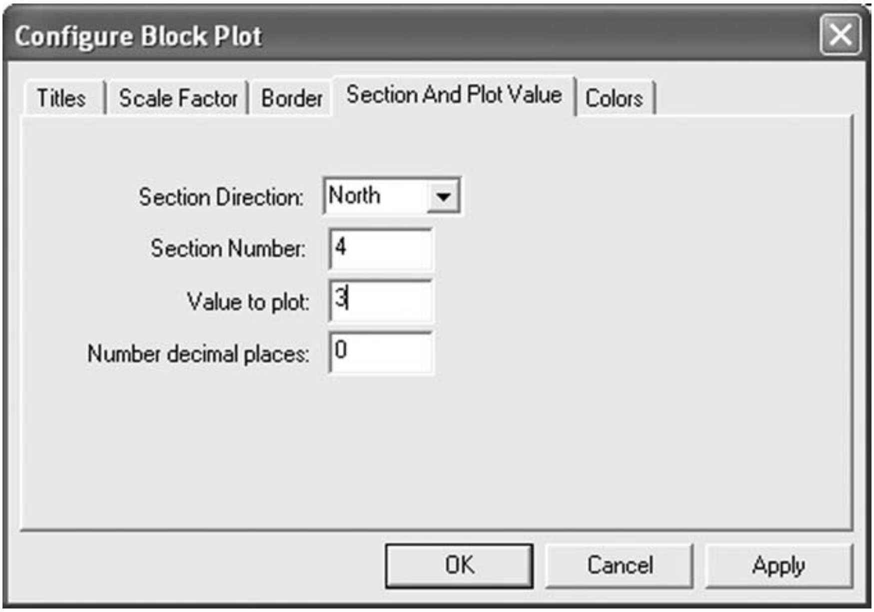
From the drop down menu:

SELECT: the 'Block Plot' command

The block plot will be displayed using the current control variables. To configure the plot to display section North 4850:

SELECT: the 'Configure' command

The 'Configure Block Plot' dialog box as explained in Section 14.10.2 will be displayed. Use the 'Section and Plot Value' tab and change the 'Section Direction' to 'North', and the 'Section Number' to 4, the 'Value to plot' to 3, and the 'Number decimal places' to 0.



To view the plot as configured:

SELECT: the 'OK' button

The block plot shown in Figure 14.18 will then be displayed. As can be seen, the blocks above the surface are left blank.

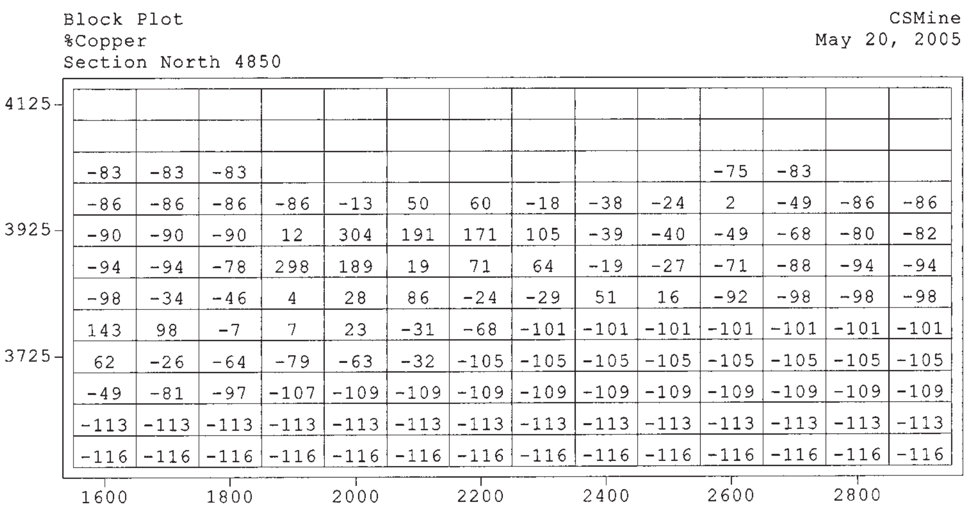
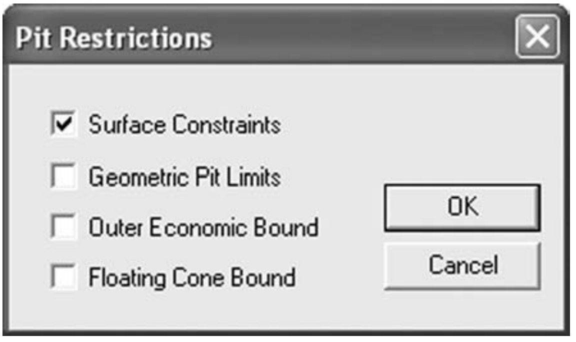


Figure 14.18. Section North 4 plot with surface constraints on.

After viewing the plot, return to the 'Pit Restrictions' window with the following series of commands:

SELECT: the 'Return' command {to exit the Graphics screen}
SELECT: the 'Restrictions' command

The 'Pit Restrictions' dialog box should now appear on the screen.



The geometric pit limits. The 'Geometric Pit Limits' option restricts the pit to those blocks that are physically possible to mine within the block model given the current pit slopes. The 'Slopes' command is used to enter the pit slopes and is discussed in the next section. The default pit slopes are set to 1:2 in all four directions. For the block size of 100 ft × 100 ft × 50 ft, this results in 45 pit slopes. To turn the 'Geometric Pit Limits' constraints on, click the mouse in the small box next to the constraint name. A small check mark will appear indicating that this constraint is now turned on. To apply the 'Geometric Pit Limits' constraint restriction:

SELECT: the 'OK' button

The main data window display will be updated to show only those blocks that lie below the surface topography and within the geometric pit limits.

The restricted block model can now also be viewed using the Block Plot option by executing the following series of commands:

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Block Plot' command

The block plot shown in Figure 14.19 should appear on the screen. As can be seen, a large number of blocks are now left blank. The blocks shown with values written inside are the only blocks on this section that can be mined if the required pit slopes are to be maintained.

After viewing the plot, return to the 'Pit Restrictions' window with the following series of commands:

SELECT: the 'Return' command {to exit the Graphics screen}
SELECT: the 'Restrictions' command

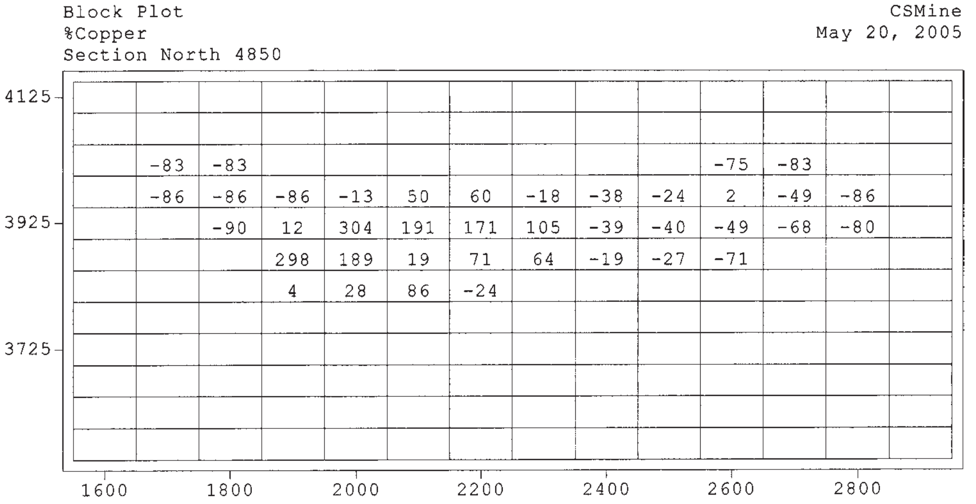
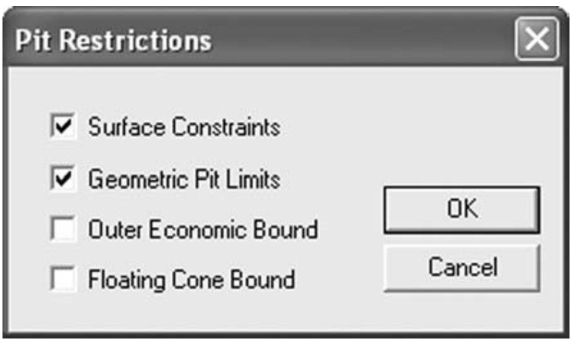


Figure 14.19. Section North 4 with surface and geometric pit limits constraints turned on.

The ‘Pit Restrictions’ dialog box should now appear on the screen.



The outer economic bound. The ‘Outer Economic Bound’ is simply the pit that would result from the union of all cones meeting the specified pit slopes which apex in a block with a positive economic value. This pit is useful for planning purposes since any block that falls outside these pit limits would never be mined under the specified economic conditions. To turn the ‘Outer Economic Bound’ constraints on, click the mouse in the small box next to the constraint name. A small check mark will appear indicating that this constraint is now turned on. To apply the ‘Outer Economic Bound’ constraint restriction:

SELECT: the ‘OK’ button

The main data window display will be updated to show only those blocks that lie below the surface topography and within the outer economic bound.

The restricted block model can now also be viewed using the Block Plot option by executing the following series of commands:

SELECT: the ‘Plot’ command

From the drop down menu:

SELECT: the ‘Block Plot’ command

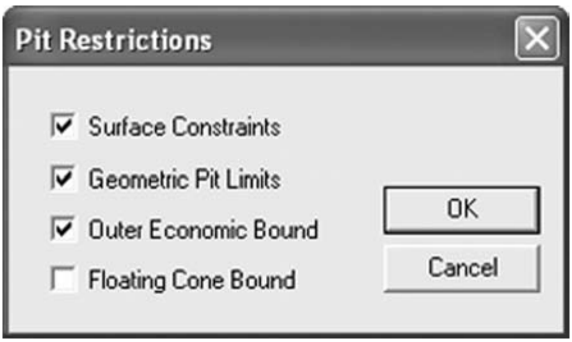
The block plot shown in Figure 14.20 should appear on the screen. The blocks shown in Figure 14.20 are the same as those shown in Figure 14.19. The difference is the thick line that is drawn around the positive blocks. The blocks that fall inside the thick line make up the outer economic bounds. If each North section were to be plotted, positive valued blocks would always fall within the pit defined by the thick lines.

After viewing the plot, return to the ‘Pit Restrictions’ window with the following series of commands:

SELECT: the ‘Return’ command {to exit the Graphics screen}

SELECT: the ‘Restrictions’ command

The ‘Pit Restrictions’ dialog box should now appear on the screen.



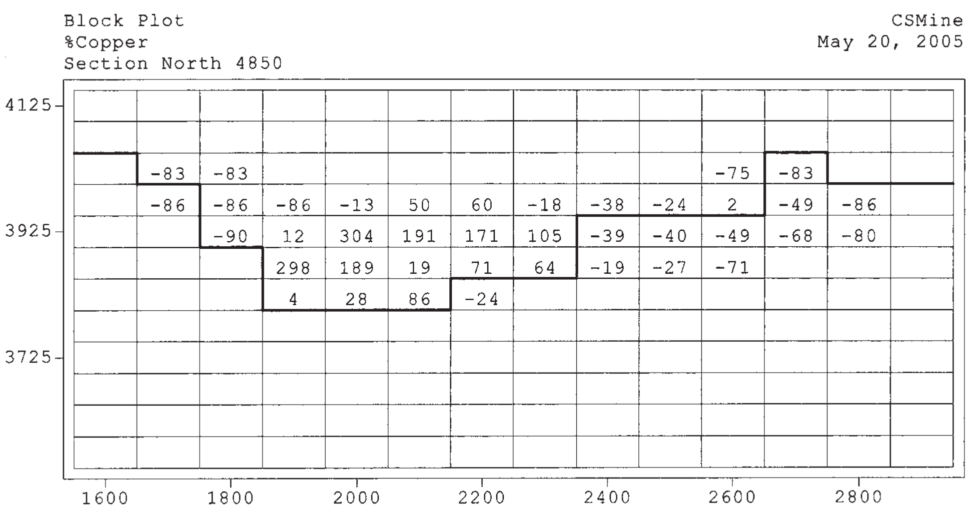


Figure 14.20. Section North 4 with ‘Outer Economic Bound’ on.

The floating cone bound. The final restrictions option is used to turn on the 'Floating Cone Bound'. A simple Floating Cone algorithm is used which is described in detail in Chapter 15. To turn the 'Floating Cone Bound' constraints on, click the mouse in the small box next to the constraint name. A small check mark will appear indicating that this constraint is now turned on. To apply the 'Floating Cone Bound' constraint restriction:

SELECT: the 'OK' button

The main data window display will be updated to show only those blocks that lie below the surface topography and within the floating cone bound.

The restricted block model can now also be viewed using the Block Plot option by executing the following series of commands:

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Block Plot' command

The block plot shown in Figure 14.21 should appear on the screen. This plot takes on the same appearance as Figure 14.20, but in this case the thick lines represent the optimal economic pit limits as defined by the Floating Cone algorithm. Note that this algorithm cannot guarantee the producing of true optimal pit limits. It does however produce a true inner bound. The true optimal economic pit limits thus lie somewhere between those from the outer economic bound shown in Figure 14.20 and the floating cone bound shown in Figure 14.21. More sophisticated algorithms such as the Lerchs-Grossmann algorithm can be used to find the true optimal economic pit limits. However a program which implements this algorithm

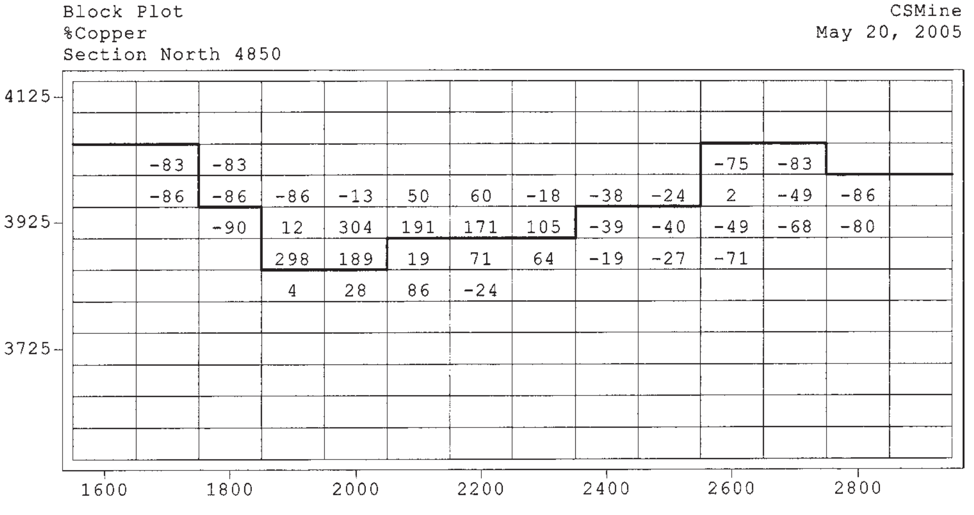


Figure 14.21. Section North 4 with floating cone bound on.

would take a considerable amount of computer memory and time to run, and was thus deemed not appropriate for an introductory program like CSMine.

One may wonder why some of the positive blocks shown in Figure 14.21 are not mined with the Floating Cone algorithm. For example, the block with a value of 64 has no blocks over it, and thus from viewing only Figure 14.21, it would appear that this block should be mined. However if the three-dimensional effect of the pit is taken into account, then the positive value of 64 for this block does not support the costs encountered from the negative valued blocks that must be mined in order to mine this block. This can be verified by the block plot through section East 8 shown in Figure 14.22. It can be seen that negative valued blocks totaling −133 must be mined in order to mine the positive 64 block, which would not be economic.

After viewing the block plot, return to the Block Mode main data window by:

SELECT: the 'Return' command

14.10.6 *Pit plots*

The pit plot is the final type of plot available for presenting the results of the various restricted block models discussed in the previous section. First the 'Restrictions' command will be used to turn on all the pit constraints. From the 'Block Mode' menu:

SELECT: the 'Restrictions' command

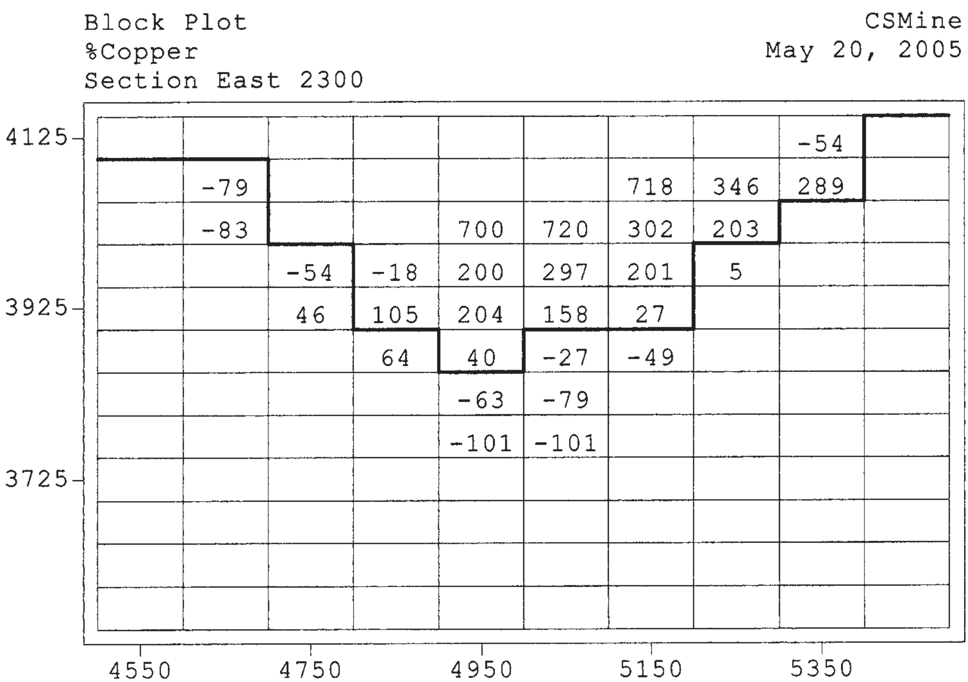
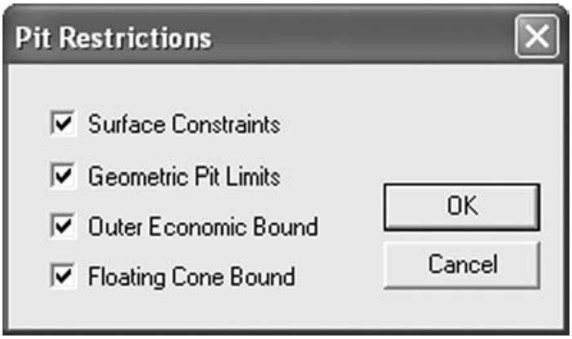


Figure 14.22. Block plot through section East 8.

Turn on all the restrictions. The 'Pit Restrictions' window should appear as shown.



SELECT: the 'OK' button

To draw the pit plot with the surface constraints on:

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Pit Plot' command

The pit plot shown in Figure 14.23 will then be displayed. This plot shows a bird's eye view of the block model. If each block had its bench number written on the top of the

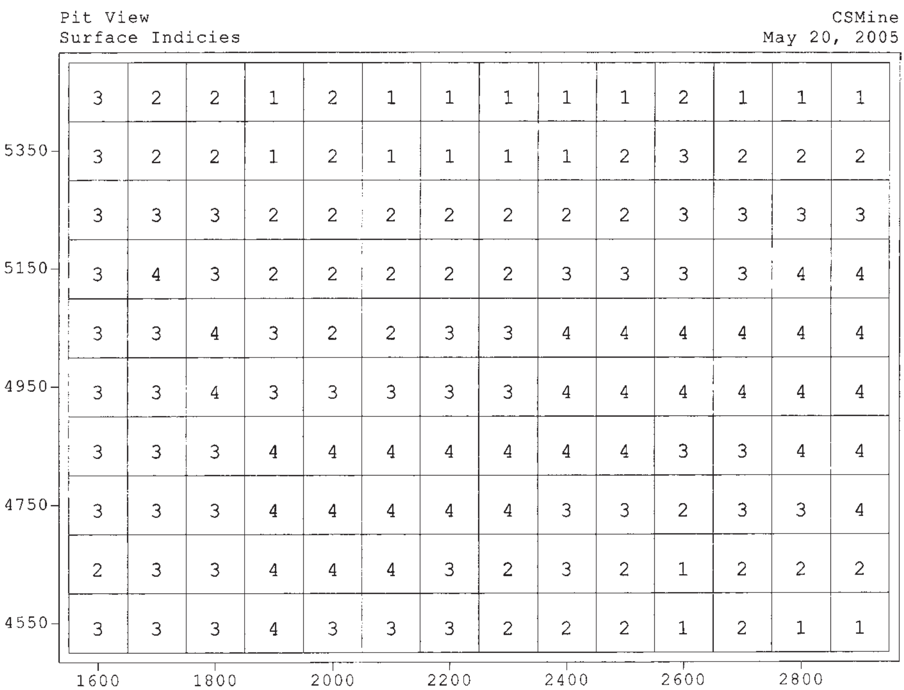


Figure 14.23. Pit plot showing surface constraints.

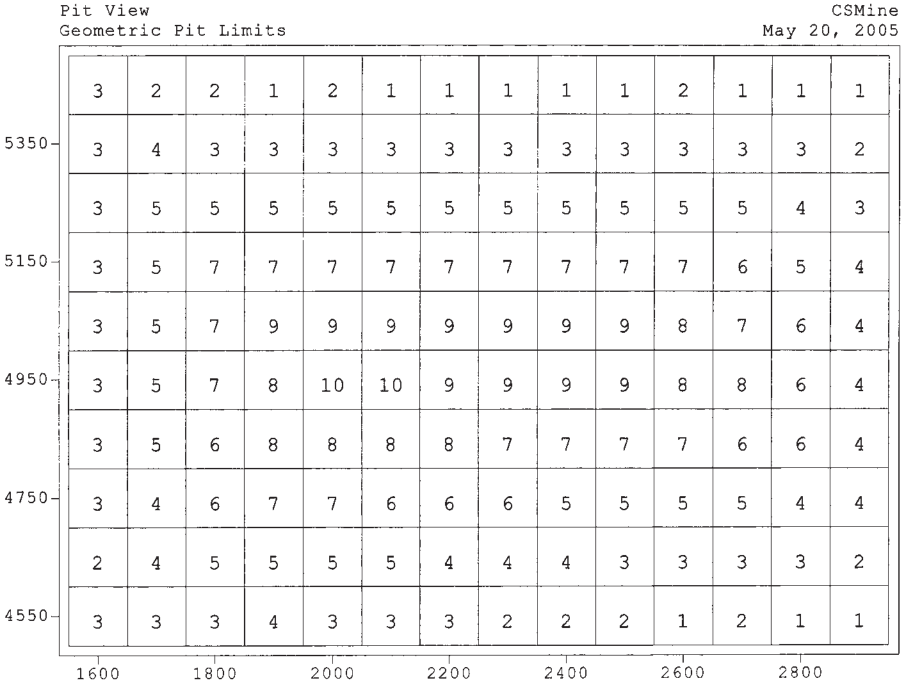


Figure 14.24. Pit plot showing geometric pit limits.

block, then looking straight down over the pit, one would see the block numbers shown in Figure 14.23.

Pit plots for the four pit restrictions can be displayed by selecting the ‘Surface’, ‘Geometric’, ‘Outer_Economic’, and ‘Floating_Cone’ commands from the ‘Pit Plot’ window command line. These plots are shown in Figure 14.25, Figure 14.26, and Figure 14.27 respectively. This method of presenting the block models can be of help in the planning process. These plots can also be used to verify the results of the various restricted block models.

After viewing the last of the pit plots, return to the Block Mode main data window by:

SELECT: the ‘Return’ command

14.10.7 *The Slopes command*

The ‘Slopes’ command is used to change the pit slopes from the default values. The designation ‘North Face Slope’ denotes the slope on the north side of the pit, the ‘South Face Slope’ denotes the slope on the south side of the pit, etc. From the Block Mode main data window:

SELECT: the ‘Slopes’ command

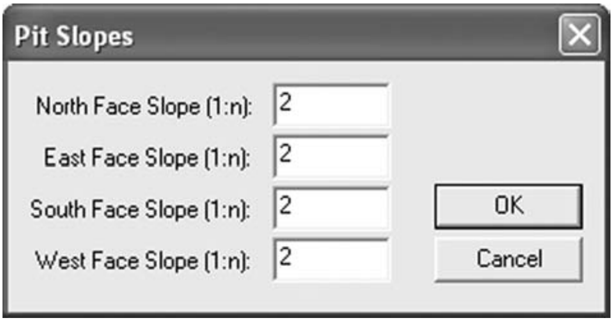
Pit View
Outer Economic Bound

CSMine
May 20, 2005

	3	2	2	1	2	1	1	1	1	1	2	1	1	1
5350	3	3	3	3	3	3	3	3	3	3	3	2	2	2
	3	4	5	5	5	5	5	5	5	3	3	3	3	3
5150	3	5	6	7	6	6	6	6	5	5	5	3	4	4
	3	5	7	8	8	7	6	6	6	6	5	4	4	4
4950	3	5	6	7	9	8	7	7	7	7	5	4	4	4
	3	4	6	8	8	8	7	7	5	5	5	3	4	4
4750	3	4	6	6	7	6	6	6	5	3	3	3	3	4
	2	4	4	5	5	5	4	4	4	3	3	3	2	2
4550	3	3	3	4	3	3	3	2	2	2	1	2	1	1
	1600		1800		2000		2200		2400		2600		2800	

Figure 14.25. Pit plot showing outer economic bound.

The 'Pit Slopes' dialog box will then appear.



Change the 'North Face Slope' value from 2 to 3. With the given block size of 100 ft × 100 ft × 50 ft, this would give a slope along the north face of the pit of 56°.

The effect of increasing the pit slopes is that more positive valued blocks can be mined and less negative valued blocks need to be removed to get at the ore, resulting in greater

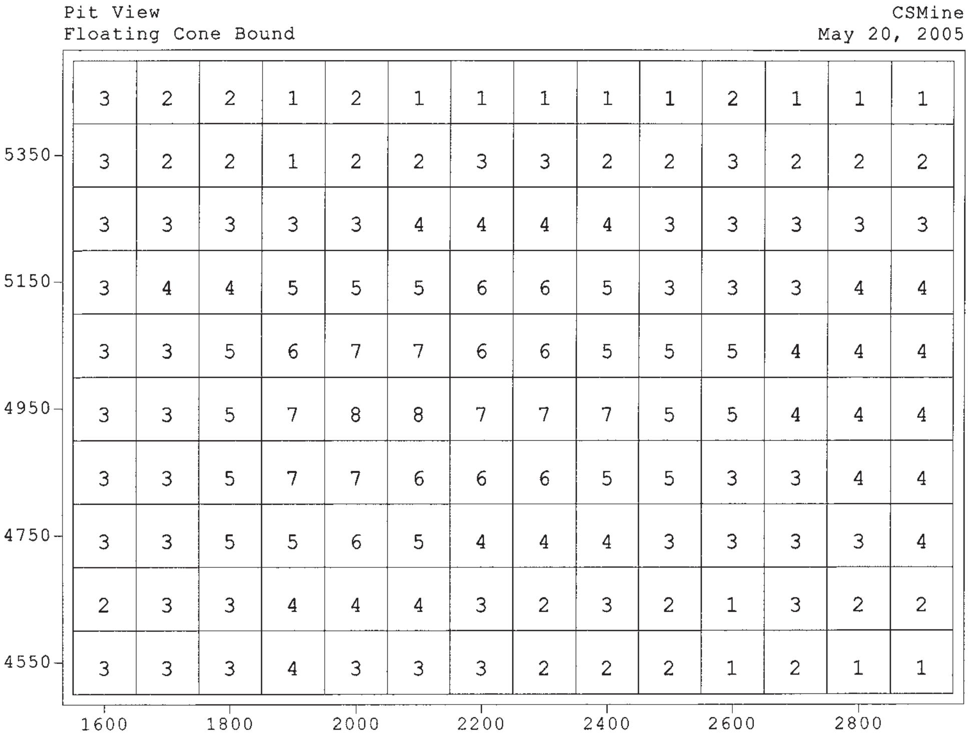


Figure 14.26. Pit plot showing floating cone bound.

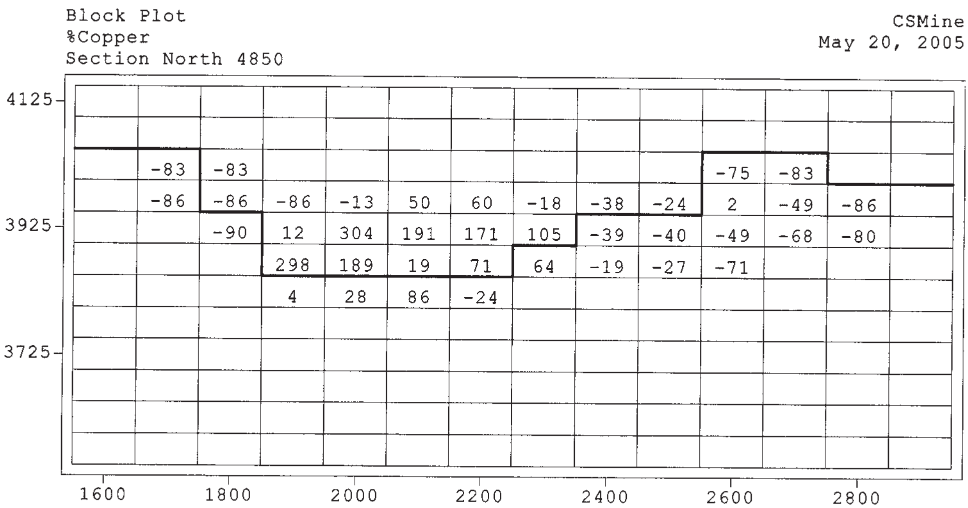


Figure 14.27. Section North 4 with 'Floating cone Bound' on and 'North Face Pit Slope' set to 1:3.

overall economy. The effects of changing the 'North Face Slope' can be viewed in the block plot of section North 4 shown earlier by issuing the following commands:

SELECT: the 'OK' button {to apply changes and close the dialog box}

If the pit restriction options are not all turned on, then open the 'Restrict' command dialog box and turn on the 'Floating Cone Bound' before continuing.

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Block Plot' command

The block plot shown in Figure 14.27 will then be displayed on the screen. If a different section or bench is displayed, use the 'Configure' command to open the 'Configure Block Plot' dialog box and set the values to plot section North 4. Compare this figure with Figure 14.22 and observe that two more positive valued blocks with values 19 and 71 are mined. Try the same exercise changing "n" for the 'East Face Slope' to 3.

After viewing the section plot, return to the Block Mode main data window by:

SELECT: the 'Return' command

14.10.8 *The Save and Print commands*

The 'Save' or 'Save As' command can be used to write either composites or a block model to a disk file, which can later be read into the program with the 'Open' command. The 'Save' command will use the existing file name if one has been defined either in the project file or through a previous use of the 'Save As' command. With the 'Save' command, the standard Windows file dialog box is normally not invoked. With the 'Save As' command, the Windows file dialog box is opened to allow the user to specify a new or existing file to write the data to. The disk file format used with the 'Save' or 'Save As' command is an ASCII format that is compatible with spreadsheet type programs such as Excel.

If the block model has been restricted using the 'Restrictions' command, the complete set of blocks is still written when either the 'Save' or 'Save As' commands are used. Alternatively, to save a restricted block model, the 'Save As Restricted' command may be used. Using this command, only the blocks that meet the current restrictions will be written. Block models written using this command cannot be read back into CSMine with the 'Open' command. They can however be imported into a spreadsheet program such as Excel.

From the 'Block Mode' menu:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Save' command

Or

SELECT: the 'Save As' command

Or

SELECT: the ‘Save As Restricted’ command

Printing is done through the standard windows print dialog through which any local or network printer currently available can be selected. To print:

SELECT: the ‘File’ command

From the drop down menu:

SELECT: the ‘Print’ command

The ‘Print Preview’ commands may be used to preview the pages that will be printed, and the ‘Print Setup’ command can be used to configure the printer and select various printer options. Both of these commands use the standard Windows dialogs.

14.11 CONCLUSION

This concludes the CSMine tutorial. First time users should now be familiar with the basic design and function of the CSMine program. While most of the program commands were covered, some were not. A complete discussion of all of the CSMine program commands is given in Chapter 15.

14.12 SUGGESTED EXERCISES

Several simple exercises are suggested below to give new users to CSMine some additional practice with using the program.

1) Use the ‘File’ – ‘Save As’ command to create a data file of the block grade values for the unrestricted block model. Import these values into a spreadsheet program and plot a grade/tonnage curve for the deposit.

2) Assume three planning periods. Change the economic values and run the Floating Cone algorithm to assist in developing a mining sequence plan for the three periods.

3) Use kriging as the estimation technique instead of Inverse Distance Squared. Use the variogram calculation and modeling routines in CSMine to calculate the required variogram. The example data set used in the tutorial has a highly skewed lognormal distribution and thus lognormal geostatistics must be used. Sections 15.11 and 15.12 of the CSMine User’s Guide include a complete example using the ‘ArizCu.dhf’ data set. The required log transformation of the data, calculation of the required β factor, and the final variogram modeling of the log-transformed data is discussed.

4) Change the block size from 100 ft × 100 ft × 50 ft to 50 ft × 50 ft × 25 ft and repeat the tutorial from the beginning. You will have to make a new surface topography file for the increased number of blocks required with the smaller block size. Figure14.4 and Figure14.13 can be used to obtain the required surface elevations.

CHAPTER 15

CSMine user's guide

CSMine was developed as an easy to learn and use microcomputer-based program for teaching the principles of computerized mine planning. The main emphasis of the program is on open pit mine planning, but CSMine is general enough so that many of its functions can be used for underground mine planning as well. The program is designed to take raw drill hole data through the block modeling process to the generation of the final economic pit limits. Emphasis within the program has been placed on data manipulation and graphical presentation. Major features of the program include:

1) Graphical displays of drill hole data:
   a) Drill hole plan maps.
   b) Drill hole section or profile maps.

2) Compositing of raw drill hole data to regularly spaced samples for processing by the block model.

3) Block modeling by the Inverse Distance Squared method, ordinary kriging, lognormal kriging, and universal kriging.

4) Graphical presentation of the block model data:
   a) Block plots through any bench or section.
   b) Contour plots through any bench or section.

5) Assigning of economic values to the blocks and their graphical presentation as in item 4.

6) Final pit limit generation:
   a) Geometric pit limits defined by the surface topography and pit slope constraints.
   b) Economic pit limits defined by a three-dimensional Floating Cone algorithm.

The program is divided into three modules to deal with the three types of data. These are:

1) The Drill Hole mode – used to read in and display the raw drill hole data.

2) The Composite mode – used to regularize the raw drill hole data into composites of equal length.

3) The Block mode – used to create and display the block model, to assign economic values to the blocks, and to generate the final economic pit limits.

15.1 BASICS

15.1.1 *File types*

The types of data files, their extension, and maximum size are listed below.

Description	Extension	Size
Project file	.KON	
Drill hole file	.DHF	1000 drill holes, 1000 assays per hole
Composite file	.CMP	8000 composites
Block file	.BLK	32000 blocks
Elevation file	.ELV	8000 blocks
Variogram file	.VGM	6 variograms

15.1.2 *The project file*

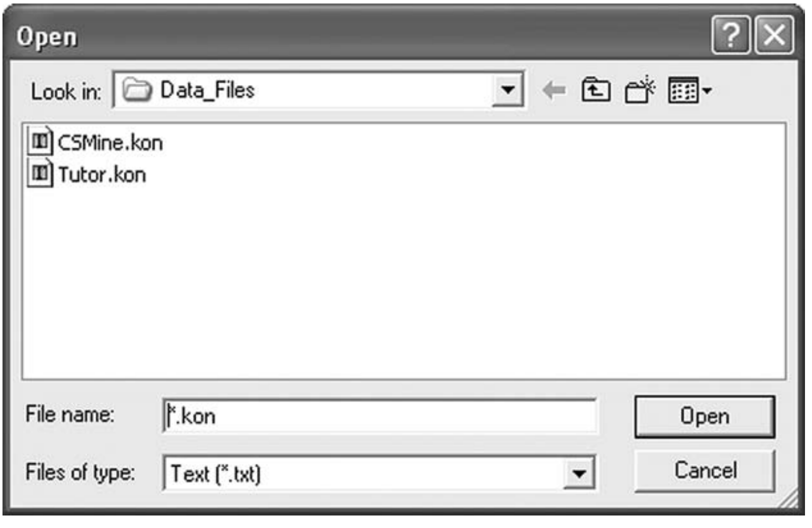
A project file contains all the parameters that can be entered into the various program dialog boxes, such as the block model size and graphics factors. A project file can be saved at the end of a session and read at the beginning of the next session. This eliminates the need to re-enter parameters each time the program is run. Any number of project files can be created. To read a project file, from menu line located at the top of the program window for the Drill Hole, Composite, and Block modes:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Load Project File' command

The standard Windows file dialog box will then appear from which the name of an existing project file to read can be selected.



When a project file is read, the current program menu values are replaced by the values contained in the project file. No data files are read, however. The drill hole, composite, and/or block files associated with the project file must also be read.

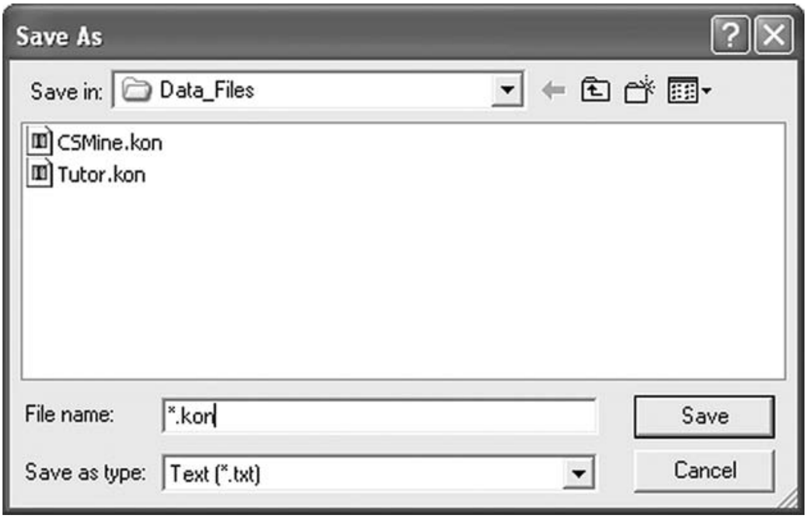
To save a project file, from menu line located at the top of the program window for the Drill Hole, Composite, and Block modes:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Save Project File' command

If a project file name is currently defined through the use of a previous 'Load Project File' or 'Save Project File As' command, then the project file will be written to the defined file. If no project file name is currently defined, the standard Windows file dialog will appear from which the name of an existing project file can be selected or a the name of a new file entered.



To save a project file with a new name, from the menu line located at the top of the program window for the Drill Hole, Composite, and Block modes:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Save Project File As' command

The standard Windows file dialog will appear as shown above from which the name of a new file entered.

15.1.3 *Changing modes*

The CSMine program is organized around three program modes: the Drill Hole mode discussed in Section 15.2, the Composite mode discussed in Section 15.3, and the Block mode discussed in Section 15.4. Each mode has an associated main data window. The current mode determines what type of data are displayed in the data window as well as what type of data can be manipulated by the various program commands.

The current mode is always displayed at the top of the window. The command line located at the top of the data window for each of the three program modes contains the 'Mode' command that is used to switch between the three program modes.

To change program modes, from the 'Drill Hole Mode', 'Composite Mode' or 'Block Mode' menu:

SELECT: the 'Mode' command

From the drop down menu:

SELECT: the 'Drill Hole', 'Composite', or 'Block' command

The associated data window will be updated to display any data currently stored in memory for the mode selected.

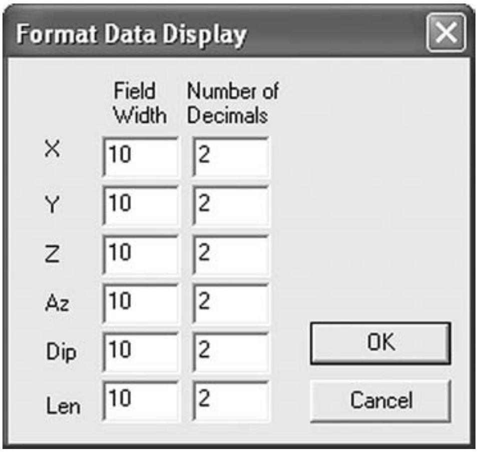
15.1.4 *Formatting the data screen*

The format of the data screen can be set for each of the three groups of data. This means the total column width and number of decimal places may be set for each data column.

From the 'Drill Hole Mode', 'Composite Mode' or 'Block Mode' menu:

SELECT: the 'Format' command

The 'Format Data Display' dialog box will then appear.



The elements shown in the 'Format Data Display' dialog box will vary depending on the current program mode. The 'Field width' specifies the total column width for the selected data item. The maximum allowable field width is 15 digits. The 'Number of Decimals' specifies the number of digits to the right of the decimal place. The maximum number of decimals is 13. When the format has been set for the desired columns:

SELECT: the 'OK' command

to close the dialog box and update the screen with the new format settings, or

SELECT: the ‘Cancel’ command

to close the dialog box and keep the previously selected format settings.

15.1.5 *Sorting data*

The data can be sorted in ascending order for any data column.

From the ‘Drill Hole Mode’, ‘Composite Mode’ or ‘Block Mode’ menu:

SELECT: the ‘Sort’ command

A drop down menu will then appear from which the data value on which the sort is to be performed can be selected. The choices available will depend on the current program mode:

Drill hole mode: # ID, X, Y, Z, Azimuth, Dip, Length
Composite mode: # ID, X, Y, Z, Assay 1
Block mode: ijk, Value 1

Selecting the field ‘#’ as the value to sort on removes the sort, and the records will be displayed in the order they were read from the associated data file.

15.1.6 *Printing data*

Printing of data or graphics displays can be done from any data or plot window using the standard Windows print interface.

From a main data or plot window:

SELECT: the ‘File’ command

From the drop down menu:

SELECT: the ‘Print Setup’ command

to open the standard Windows dialog box for configuring the printer,

SELECT: the ‘Print Preview’ command

to view the printout as it will appear on the printed paper, and

SELECT: the ‘Print’ command

to open the standard Windows dialog box for controlling and initiating the printing. This dialog box can also be used to select the pages to be printed. Printouts in CSMine are currently limited to standard $8.5'' \times 11''$ (American) or A4 (Metric) sized paper.

15.1.7 *Coordinate system description*

A right-handed coordinate system is used throughout the program (Figure 15.1).

The Y-axis lies along the North-South direction with y increasing to the North. The X-axis lies along the East-West direction with x increasing to the East. The Z-axis lies along the elevation direction with z increasing upward and decreasing downward.

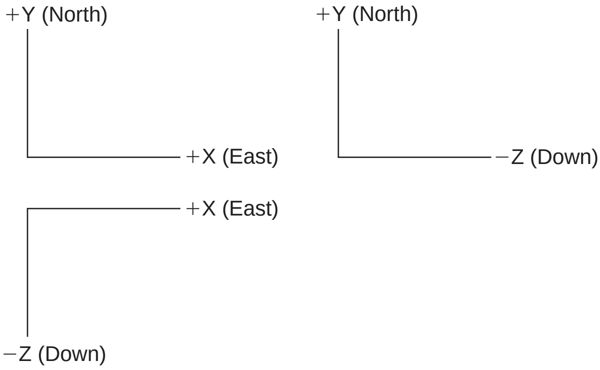
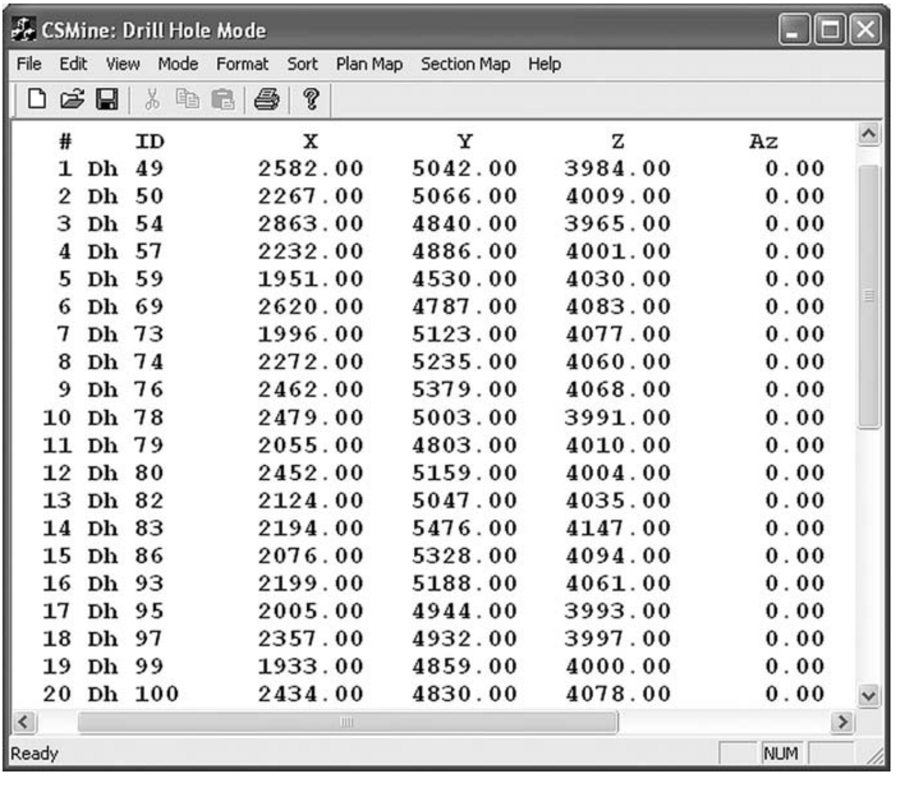


Figure 15.1. The right-handed coordinate system used in CSMine.

15.2 DRILL HOLE MODE

The main data window for the Drill Hole mode with drill hole collar information from the 'ArizCu.dhf' data set is shown below. A description of the drill hole file and the commands associated with the Drill Hole mode are discussed in the sections that follow.



#	ID	X	Y	Z	Az
1	Dh 49	2582.00	5042.00	3984.00	0.00
2	Dh 50	2267.00	5066.00	4009.00	0.00
3	Dh 54	2863.00	4840.00	3965.00	0.00
4	Dh 57	2232.00	4886.00	4001.00	0.00
5	Dh 59	1951.00	4530.00	4030.00	0.00
6	Dh 69	2620.00	4787.00	4083.00	0.00
7	Dh 73	1996.00	5123.00	4077.00	0.00
8	Dh 74	2272.00	5235.00	4060.00	0.00
9	Dh 76	2462.00	5379.00	4068.00	0.00
10	Dh 78	2479.00	5003.00	3991.00	0.00
11	Dh 79	2055.00	4803.00	4010.00	0.00
12	Dh 80	2452.00	5159.00	4004.00	0.00
13	Dh 82	2124.00	5047.00	4035.00	0.00
14	Dh 83	2194.00	5476.00	4147.00	0.00
15	Dh 86	2076.00	5328.00	4094.00	0.00
16	Dh 93	2199.00	5188.00	4061.00	0.00
17	Dh 95	2005.00	4944.00	3993.00	0.00
18	Dh 97	2357.00	4932.00	3997.00	0.00
19	Dh 99	1933.00	4859.00	4000.00	0.00
20	Dh 100	2434.00	4830.00	4078.00	0.00

15.2.1 *Drill hole data file description*

A drill hole file has three basic components:

the file header	(record type 0),
the drill hole collar records	(record type 1), and
the drill hole assay records	(record type 2).

A drill hole file begins with the header records. The remaining lines of the file contain a series of drill hole records. The data for each drill hole consists of two basic parts, the collar record and the assay records.

Files may be created with any ASCII word processor or text editor.

15.2.1.1 *Drill hole file header*

The drill hole file must begin with a header. The first header line gives the number of assay values and has the following format:

Record type:	Always 0.
Blank:	Just a space between record type and number of assays.
Number of assays:	1 since the current version of CSMine can accept only one assay value.

The next line in the header gives the name of the assay value in the file and has the following format:

Record type:	Always 0.
Blank:	Just a space between record type and assay name.
Assay name:	An eight-character name.

As an example, the two file header lines for the 'ArizCu.dhf' drill hole file provided with the distribution disk are:

```
0  1
0  % Copper
```

15.2.1.2 *Drill hole collar record*

Each drill hole is defined first by a collar record. The collar record must contain the following:

Record type:	Always 1.
Blank:	Just a space between record type and hole ID.
Hole ID:	An 8 character hole identifier.
X_c, Y_c, Z_c:	The coordinates of the drill hole collar.
Azimuth:	Azimuth of the drill hole in degrees measured clockwise from North.
Dip:	The dip of the drill hole measured from the horizontal plane; a dip of +90 indicates a vertical up hole and a dip of −90 indicates a vertical down hole.
Length:	The length of the drill hole from the collar to the end of the hole.

There are no rigid formatting requirements for the position of values within a record or for the number of decimal places a value contains except that the Hole ID must start in column 3 and it must end in column 10. The program will not be able to read files that are not set up like this.

A drill hole file may contain up to 1000 collar records.

15.2.1.3 *Drill hole assay records*

Following each drill hole collar record are up to 1000 assay records. Each assay record contains the following:

Record type:	Always 3.
Blank:	Just a space between record type and hole ID.
Hole ID:	The 8 character hole ID.
From length:	The distance from the collar to the start of the assay.
To length:	The distance from the collar to the end of the assay.
Values:	One assay value.

The same rules for formatting the collar records apply to the assay records.

One assay value must be entered for each interval along the entire length of the drill hole. The assay records must be entered sequentially, i.e. the 'From length' from the previous record must be the same as the 'To length' of the current record. Any negative number can be entered where assay values are missing or undefined. Although the program logic simply ignores negative assays, they must be entered so that the program can properly read the file.

15.2.1.4 *Example drill hole data file*

The following example is a part of the drill hole file 'ArizCu.dhf' provided on the distribution disk.

```
0  1
0  %   Copper
1  Dh  49   2582.00  5042.00  3984.00  0.00  −90.0  145.00
3  Dh  49   0.00  8.00  −0.99
:
:
3  Dh  49   8.00  10.00  0.12
3  Dh  49   136.00  141.00  0.02
3  Dh  49   141.00  145.00  0.09
1  Dh  50   2267.00  5066.00  4009.00  90.00  −60.0  484.00
3  Dh  50   0.00  10.00  −0.99
3  Dh  50   10.00  17.00  0.18
3  Dh  50   17.00  22.00  0.25
:
:
3  Dh  50   474.00  479.00  0.02
3  Dh  50   479.00  484.00  0.02
```

15.2.2 *Reading a drill hole file*

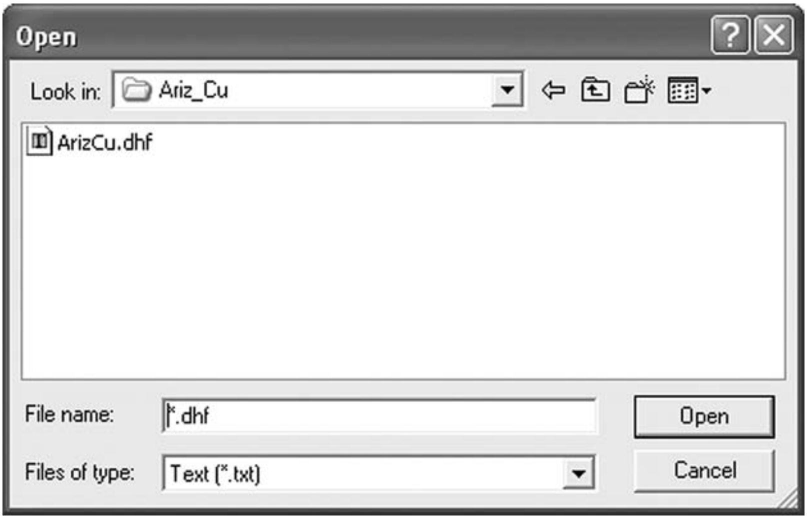
The program must be in the Drill Hole mode before a drill hole file can be read. If it is not, then change modes as described in Section 15.1.3. From the main Data window

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Open' command

The standard Windows file dialog box will then appear from which the name of an existing drill hole file to read can be selected.



If any problems were encountered in reading the file, an error message will be displayed. If the file was successfully read, the drill hole collar records will be displayed in the data window.

15.2.3 *Plotting a drill hole plan map*

After a data file has been read, a drill hole plan map can be created. The program must be in the Drill Hole mode. If it is not, then change modes as described in Section 15.1.3.

From the 'Drill Hole Mode' menu:

SELECT: the 'Plan Map' command

A drill hole plan map will then be displayed using the current settings for the various control variables as shown above.

15.2.3.1 *The Configure command*

To configure the plan map display:

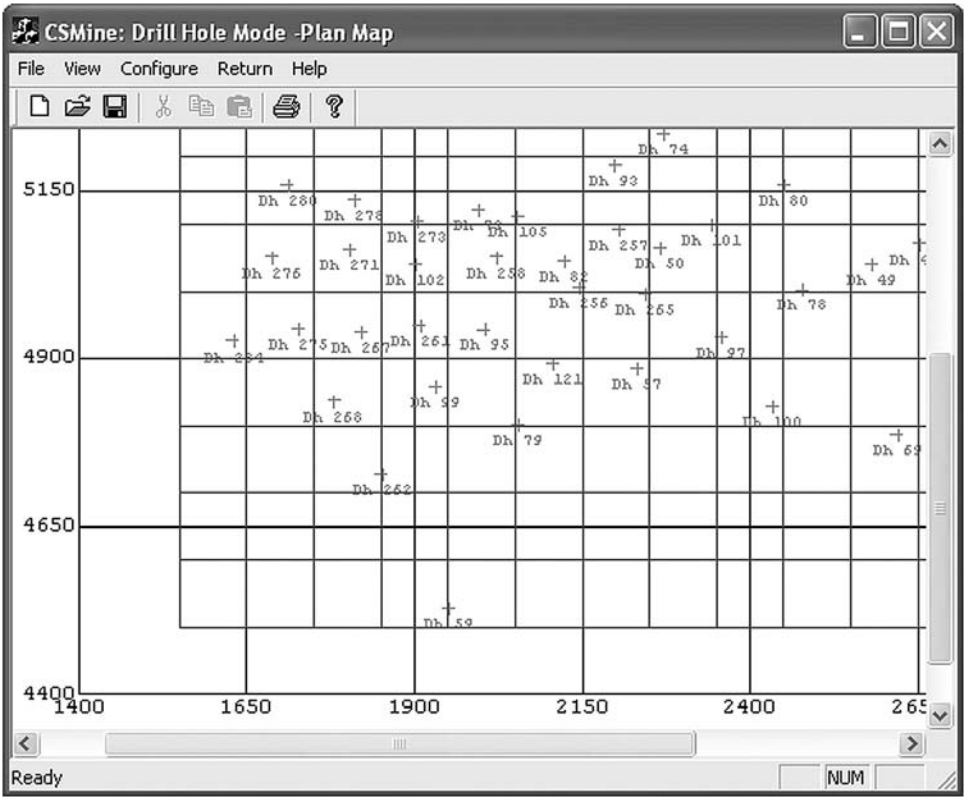
SELECT: the 'Configure' command

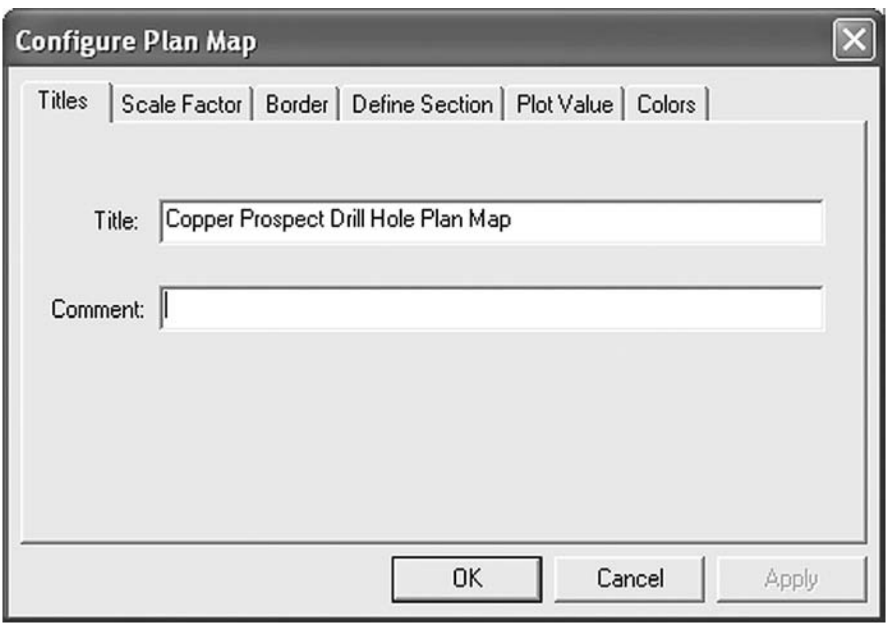
The 'Configure Plan Map' dialog box will then be displayed.

The dialog box consists of six tabs. The 'Titles' tab is used for entering the map title and comment.

Map Title: A character label, from 0 to 64 characters in length, displayed at the top of the map.

Comment: A character label, from 0 to 64 characters in length, displayed beneath the map title.

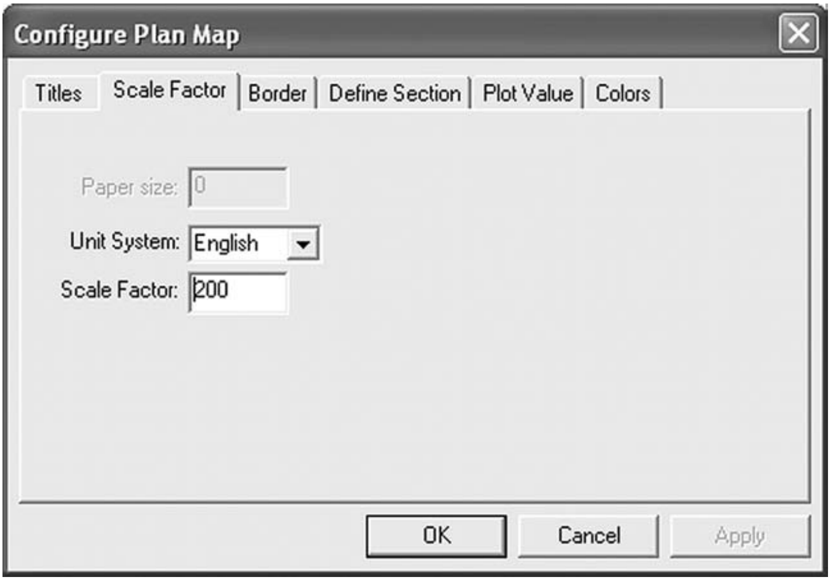

CSMine: Drill Hole Mode -Plan Map
File View Configure Return Help
5150
4900
4650
4400
1400
1650
1900
2150
2400
Dh 74
Dh 93
Dh 80
Dh 105
Dh 257
Dh 101
Dh 276
Dh 271
Dh 50
Dh 49
Dh 78
Dh 95
Dh 121
Dh 268
Dh 79
Dh 97
Ready
NUM


Configure Plan Map
Titles
Scale Factor
Border
Define Section
Plot Value
Colors
Title:
Copper Prospect Drill Hole Plan Map
Comment:
OK
Cancel
Apply

The 'Scale Factor' tab is used for entering the map unit system and scale factor.

Unit System: Plotting may be done in either English or metric units. Changing the 'Unit System' will usually require changing the 'Map Scale Factor' as well.

Map Scale Factor: This is the actual scaling factor used when transforming the data to the plotter paper. Initially this value is set so that the plot will just fit on the paper. The value should be changed to a more appropriate number for the unit system being used. For the English unit system the scale factor represents data feet per map inch. For the metric system this number represents map meters per data meter. For example, a metric scale of 1:800 means that 100 data meters will be plotted on 12.5 cm of paper.



The 'Border' tab is used for entering the locations of the first labeled X and Y axis tics, the distance between labeled tics, and the number of decimal places to use with the labels.

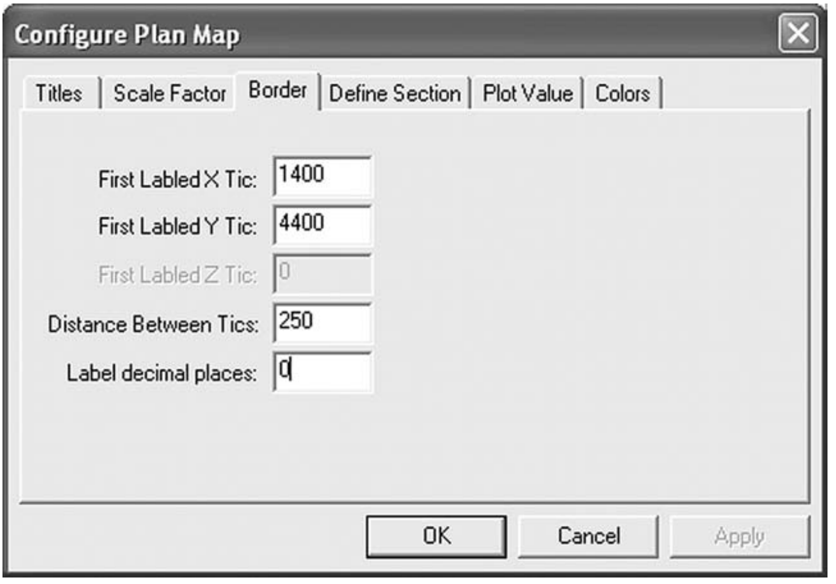
First Labeled X Tic: This is the point on the map where labeling of the X-axis is to begin.

First Labeled Y Tic: This is the point on the map where labeling of the Y-axis is to begin.

First Labeled Z Tic: This is the point on the map where labeling of the Z-axis is to begin. For the drill hole plan map, this value is not used. It is used when plotting vertical section maps.

Distance Between Tics: This is the distance in data units between labeled tics.

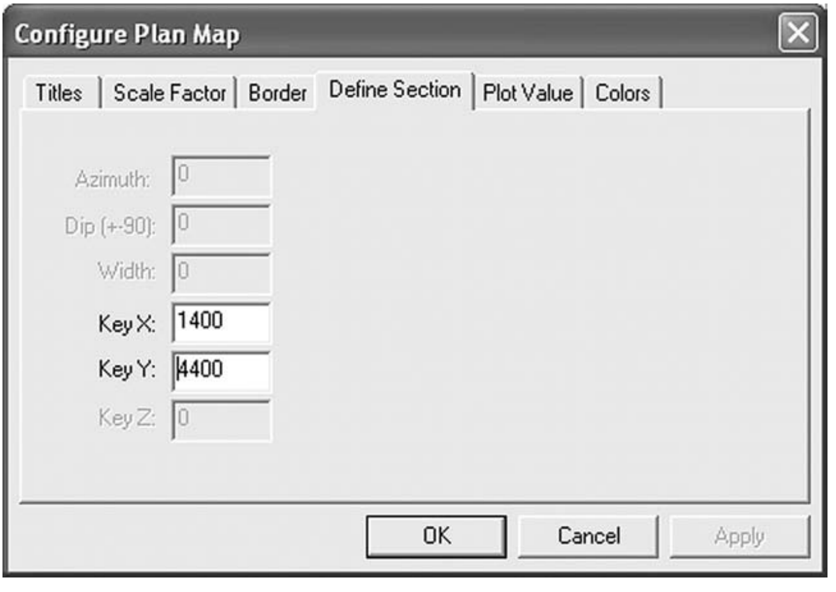
Label decimal places: This is the number of decimal places to use with the border axis labels.



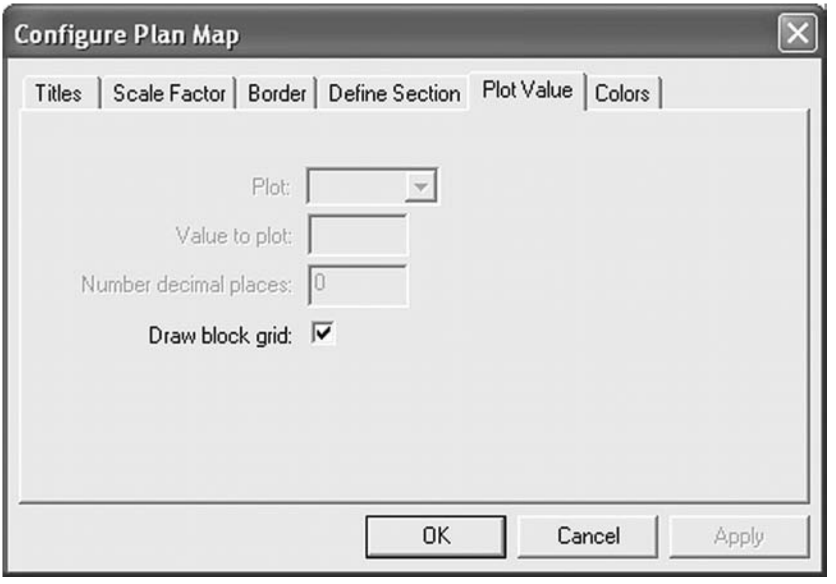
The 'Define Section' tab is used for defining the X and Y coordinates at the lower left corner of the plan map.

Key X: This is the minimum X coordinate from the data to plot. Initially this value is set to the minimum X coordinate for all of the drill holes. It may be changed to better center the map on the paper.

Key Y: The same as above except for the Y value.

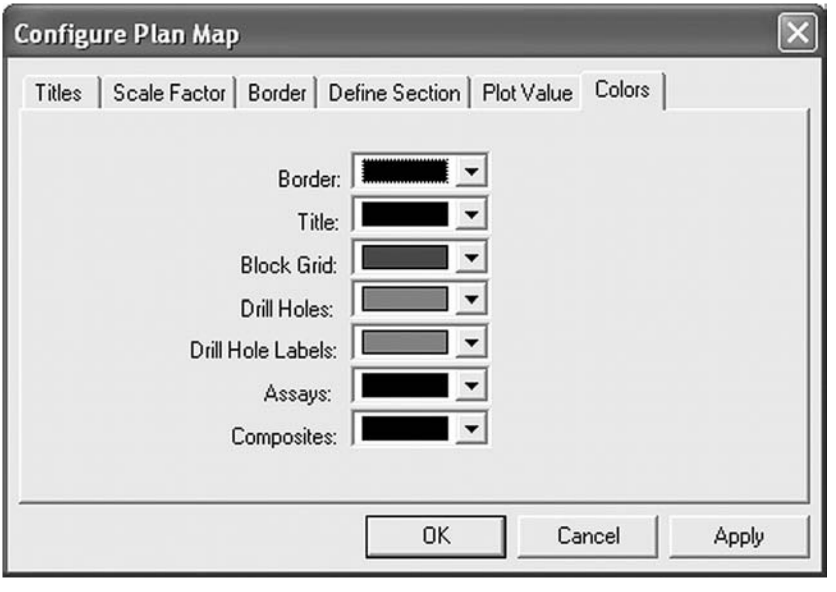


The 'Plot Value' tab is used to indicate whether the block model grid is to be drawn on the plan map.



Draw Block Grid: The check box is used to indicate whether the block model grid defined using the parameters in the Inverse Distance Squared dialog box (that can be reached from the Block mode) should be plotted over the drill holes.

The 'Colors' tab is used to indicate the color to use with the various items displayed graphically.



To update the Plan Map close the dialog box:

SELECT: the 'OK' button

The 'Configure Plan Map' dialog box will be closed and the 'Plan Map' redisplayed using the current control variable settings.

To update the 'Plan Map' without closing the dialog box:

SELECT: the 'Apply' button

The 'Plan Map' will be redisplayed using the current control variable settings and the 'Configure Plan Map' dialog box will remain open. This option will only become available if a dialog box item has been modified.

To close the dialog box without updating the 'Plan Map':

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

15.2.3.2 *The Print command*

To print the 'Plan Map' as currently configured and displayed:

SELECT: the 'File' command

From the drop down menu:

SELECT: the 'Print' command

The standard Windows print dialog will then appear from which the printer can be selected, configured, and the print initiated.

15.2.3.3 *The Return command*

To exit the 'Plan Map' window and return to the 'Drill Hole Mode' data window:

SELECT: the 'Return' command

15.2.4 *Plotting a drill hole section map*

After a data file has been read, drill hole section maps can be created. The program must be in the Drill Hole mode. If it is not, then change modes as described in Section 15.1.3.

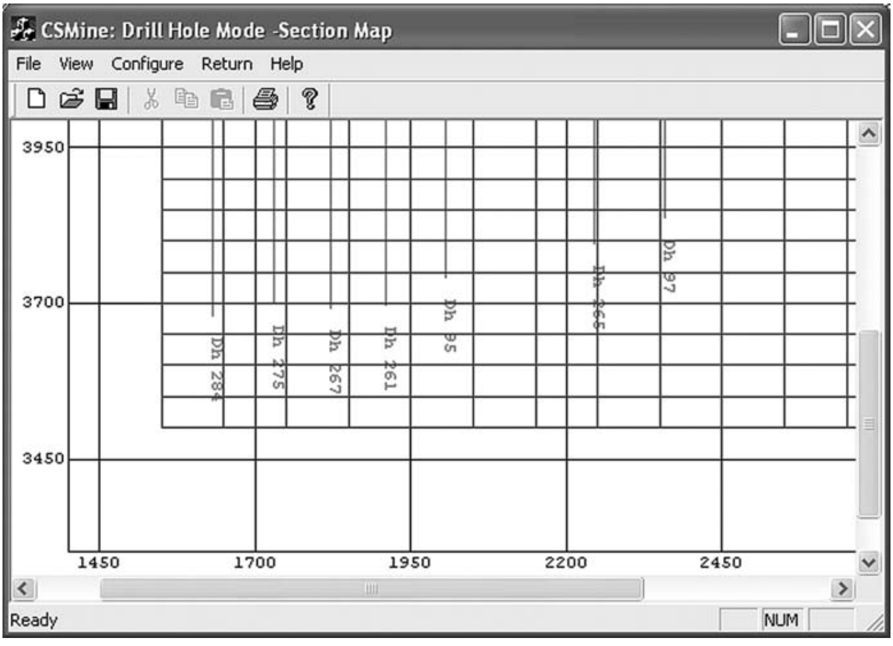
From the 'Drill Hole Mode' menu

SELECT: the 'Plan Map' command

A drill hole section map will then be displayed using the current settings for the various control variables.

15.2.4.1 *How CSMine creates a section*

Vertical drill hole sections can be taken at any orientation through the deposit. The section is defined by section key coordinates (S_x, S_y, S_z) and an azimuth. The dip of the section can be either plus (+90°) for upright sections or minus (−90°) for inverted sections. The width of the section controls which of the holes will be projected onto the section.



Since specifying parameters for sections can be somewhat confusing, a drill hole plan map should be plotted first and used for reference. Shown in Figure 15.2 is a drill hole plan map for the 'ArizCu.dhf' example data set.

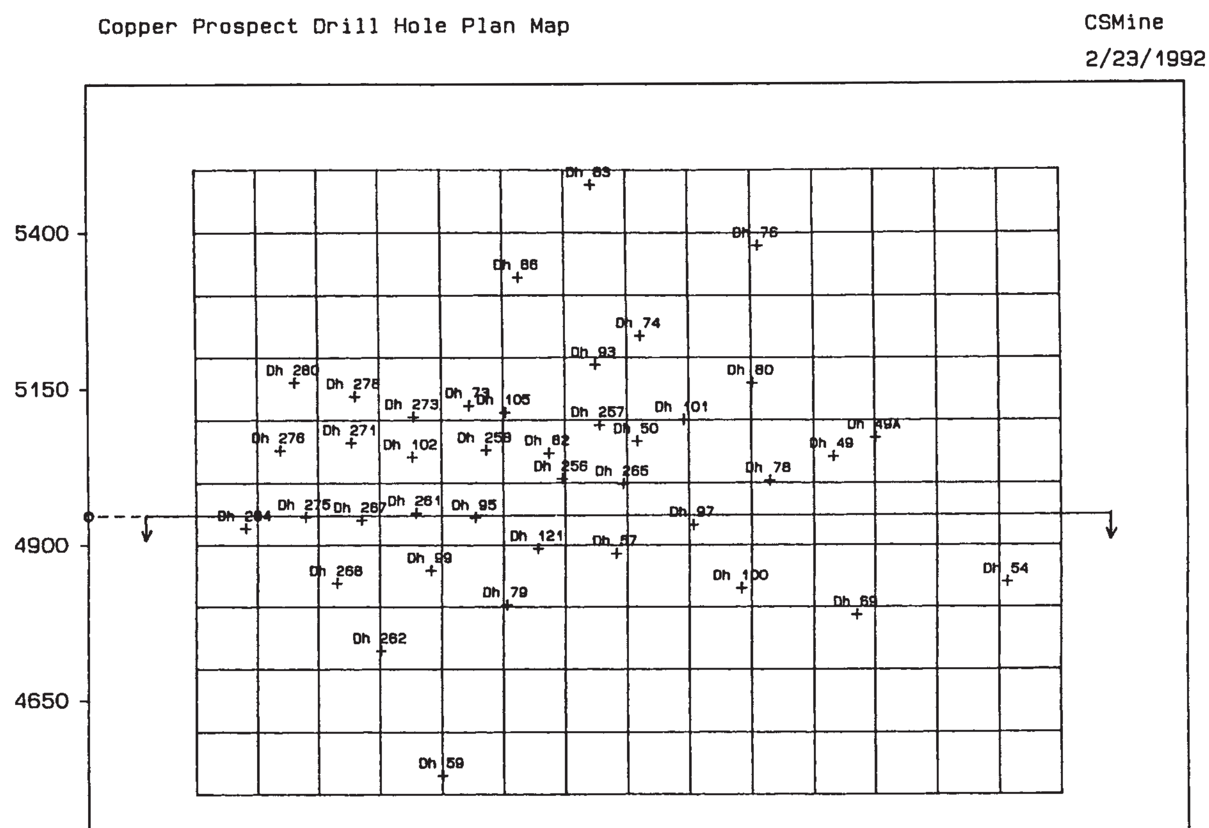
Assume a section is to be plotted along the line shown in Figure 15.2. First the azimuth of the line is needed. This is the angle in decimal degrees measured clockwise from North. In this example the section azimuth is 90°.

Next the section key coordinates (S_x, S_y, S_z) are required. These are not the same as the key coordinates (K_x, K_y, K_z) defined for the Block model.

The section plane will extend from the key coordinates to the edge of the graphics screen or plotter paper along the plane defined by the azimuth and the section dip. The actual area covered depends on the scaling factor. The X and Y key coordinates (S_x, S_y) can easily be read from the drill hole plan map. The key Z coordinate (S_z) should be set to a value lower than the bottom of the deepest hole that will be plotted on the section for upright sections (dip +90), and to a value higher than the collar of the highest hole for inverted sections (dip −90). The key coordinates for the section shown in plan view in Figure 15.2 are

S_x = The anchor X coordinate for the section
S_y = The anchor Y coordinate for the section
S_z = The anchor Z coordinate for the section

The program calculates the perpendicular distance from each drill hole collar to the section plane. If this distance is less than the section width, the entire hole is projected onto the section and plotted. This gives an effective section thickness that is twice the value specified. No additional testing is done to plot or truncate holes that partially pass through the section.



Copper Prospect Drill Hole Plan Map
CSMine
2/23/1992
5400
5150
4900
4650
Dh 83
Dh 76
Dh 86
Dh 74
Dh 93
Dh 280
Dh 80
Dh 278
Dh 73
Dh 105
Dh 273
Dh 257
Dh 101
Dh 50
Dh 49A
Dh 276
Dh 271
Dh 102
Dh 259
Dh 82
Dh 49
Dh 256
Dh 265
Dh 78
Dh 264
Dh 275
Dh 267
Dh 261
Dh 95
Dh 97
Dh 121
Dh 87
Dh 268
Dh 99
Dh 100
Dh 54
Dh 79
Dh 69
Dh 262
Dh 59

Drill holes are projected onto the section by the following mathematics:

1) Translate the point (P) to the plot by the section key coordinates:

$$T_x = P_x - S_x$$
$$T_y = P_y - S_y$$
$$T_z = P_z - S_z$$

2) Multiply by the rotation factors

$$T_x = T_x * \cos(p) + T_y * \sin(p)$$
$$T_y = -T_x * \sin(p) + T_y * \cos(P)$$
$$T_z = T_z * \sin(\mathrm{Dip})$$

where

$$p = 90° - \mathrm{Azimuth}$$

T_y is the perpendicular distance from the hole to the section. The values of T_x and T_z are scaled and plotted.

Section azimuths from 0 to 360 positive or negative degrees may be used. An azimuth greater than 180 degrees can be thought of as reversing the section. The coordinates along the X-axis will decrease to the right instead of increase. One must be sure that the key coordinates are properly specified. In the example shown in Figure 15.2, a section azimuth of 210 degrees with the key coordinates given where the arrow now is would reverse the section.

Labeling of the section plot axis is straight forward if the section lies along a North-South or East-West axis. The plotted X-axis and Y labels will correspond to the X and Y plan map axis labels respectively. Labeling of non-orthogonal sections is handled differently, however, since points on the section are in their own relative coordinate system. The X-axis is labeled starting with a value of 0 at the key X coordinate.

15.2.4.2 *The Configure command*

To configure the section map display:

SELECT: the ‘Configure’ command

The ‘Configure Section Plot’ dialog box will then be displayed. The dialog box consists of six tabs. The ‘Titles’, ‘Scale Factor’, ‘Border’, and ‘Colors’ tabs are the same as those used for configuring a Plan Map as described in Section 15.2.3.1.

Azimuth: The azimuth of the section is entered in decimal degrees measured clockwise from North.

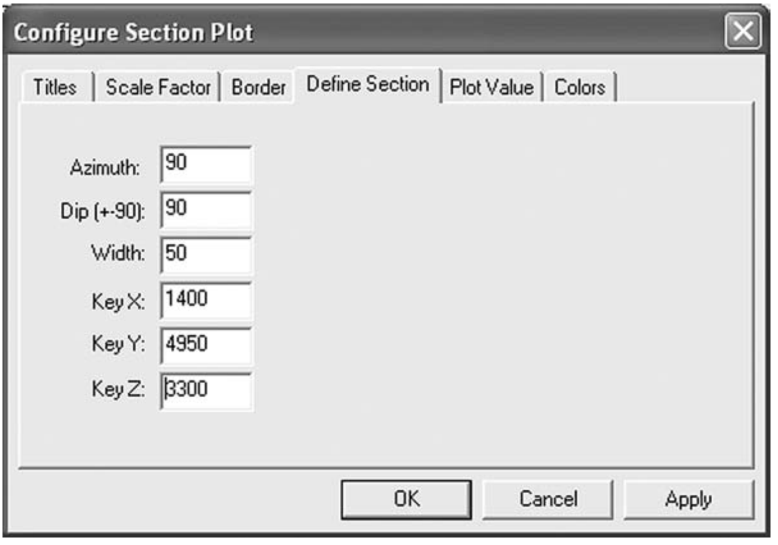
Dip: A dip of +90 indicates an upright vertical section (Z values increasing upward) and a dip of −90 indicates an inverted vertical section (Z values increasing downward).

Width: The width of the section controls which drill holes will be projected onto the section. For each hole in the drill hole file, the distance from the collar coordinates to the section line (defined by the key coordinates and azimuth) is calculated. If this distance is less than the section width, the entire hole is projected onto the section.

Key X: The anchor X coordinate for the section.

Key Y: The anchor Y coordinate for the section.

Key Z: The anchor Z coordinate for the section.



See the description given in Section 15.2.4.1 for a detailed description of the 'Key X', 'Key Y', and 'Key Z' coordinates.

The 'Plot Value tab' is used to indicate which value to plot, and to specify the data formatting.

Plot: Assay or composite values may be plotted beside the drill holes. Select 'Nothing', 'Assays', or 'Composites' from the drop down menu. When plotting assays, the assay values are read from the disk file specified when the drill hole file was read. Be sure this file is present if the 'Plot Assays' option is selected. Composite values are stored in memory and do not need to be read from an associated file when the 'Plot Composites' option is selected.

Value to Plot: Since the current version of CSMine allows for only one assay, this parameter is currently not being used.

Number decimal places: The number of digits to plot to the right of the decimal point when plotting assay or composite values. For example, 2 would result in the value 10 being plotted as 10.00.

Draw Block Grid: The check box is used to indicate whether the block model grid using the parameters defined in the 'Inverse Distance Squared' dialog box should be plotted over the drill holes.

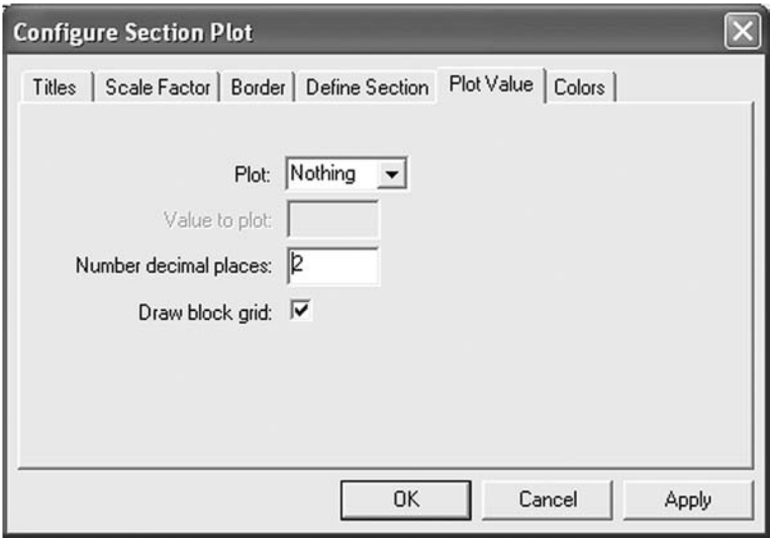
To update the 'Section Plot' close the dialog box:

SELECT: the 'OK' button

The 'Configure Section Plot' dialog box will be closed and the Section Map redisplayed using the current control variable settings.

To update the 'Section Plot' without closing the dialog box:

SELECT: the 'Apply' button



The 'Section Plot' will be redisplayed using the current control variable settings and the 'Configure Section Plot' dialog box will remain open. This option will only become available if a dialog box item has been modified.

To close the dialog box without updating the 'Section Plot':

SELECT: the 'Cancel' button

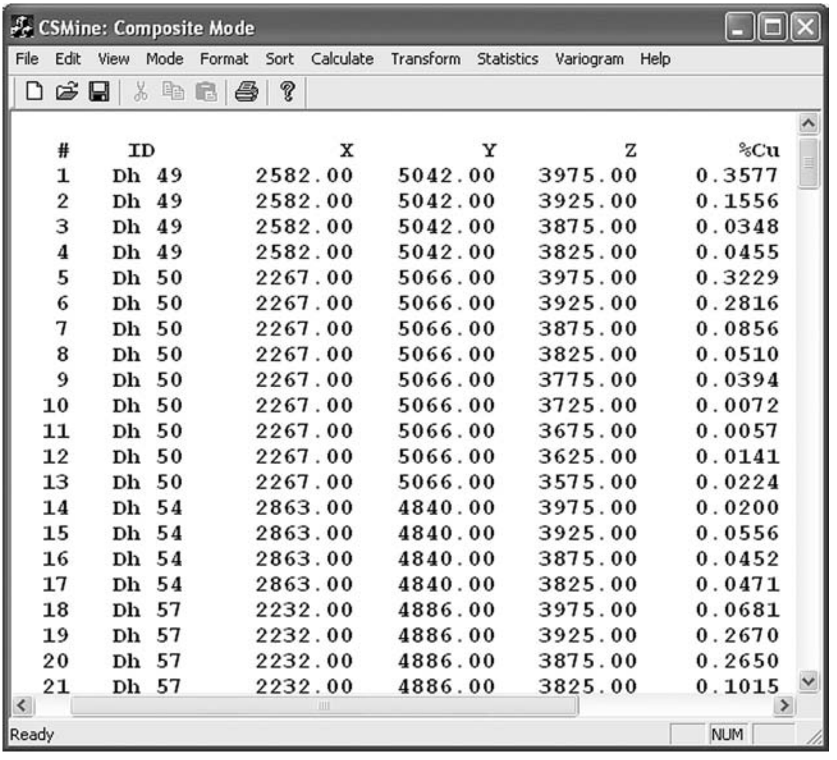
The dialog box will be closed and any changes made to dialog box values will be discarded.

15.3 COMPOSITE MODE

The main Data window for the Composite mode with the composites calculated from the 'ArizCu.dhf' data set is shown below. A description of the drill hole compositing, the composite file format, and the commands associated with the Composite mode are discussed in the sections that follow.

Before the block model can be run, the original drill hole data must be composited. Compositing involves calculating samples at regular intervals down the length of a drill hole from the original drill hole assays. Compositing can be thought of as re-cutting the drill hole core into equal length pieces. This is required so that samples will have the same weight or influence when used to calculate block values.

Two types of compositing are available: bench and collar. With bench compositing, the length of the composites are equal to the bench height and calculated such that the centers of the composites have the same Z coordinates as the centers of the benches in the block model. With collar compositing, the composites are calculated at regular intervals starting from each drill hole collar. Bench compositing is most appropriate with vertical holes. Collar compositing must be used with angled holes such as with holes drilled in a fan from an underground exploration drift.



#	ID	X	Y	Z	%Cu
1	Dh 49	2582.00	5042.00	3975.00	0.3577
2	Dh 49	2582.00	5042.00	3925.00	0.1556
3	Dh 49	2582.00	5042.00	3875.00	0.0348
4	Dh 49	2582.00	5042.00	3825.00	0.0455
5	Dh 50	2267.00	5066.00	3975.00	0.3229
6	Dh 50	2267.00	5066.00	3925.00	0.2816
7	Dh 50	2267.00	5066.00	3875.00	0.0856
8	Dh 50	2267.00	5066.00	3825.00	0.0510
9	Dh 50	2267.00	5066.00	3775.00	0.0394
10	Dh 50	2267.00	5066.00	3725.00	0.0072
11	Dh 50	2267.00	5066.00	3675.00	0.0057
12	Dh 50	2267.00	5066.00	3625.00	0.0141
13	Dh 50	2267.00	5066.00	3575.00	0.0224
14	Dh 54	2863.00	4840.00	3975.00	0.0200
15	Dh 54	2863.00	4840.00	3925.00	0.0556
16	Dh 54	2863.00	4840.00	3875.00	0.0452
17	Dh 54	2863.00	4840.00	3825.00	0.0471
18	Dh 57	2232.00	4886.00	3975.00	0.0681
19	Dh 57	2232.00	4886.00	3925.00	0.2670
20	Dh 57	2232.00	4886.00	3875.00	0.2650
21	Dh 57	2232.00	4886.00	3825.00	0.1015

Figure 15.3 shows an example of bench compositing with a section through the 'ArizCu.dhf' data set. Figure 15.4 shows an example of collar compositing with fan drilling data.

Drill hole section maps can be plotted with the composite values displayed beside the drill holes. This option is available from the Drill Hole mode. The generation of section maps is discussed in Section 15.2.4.

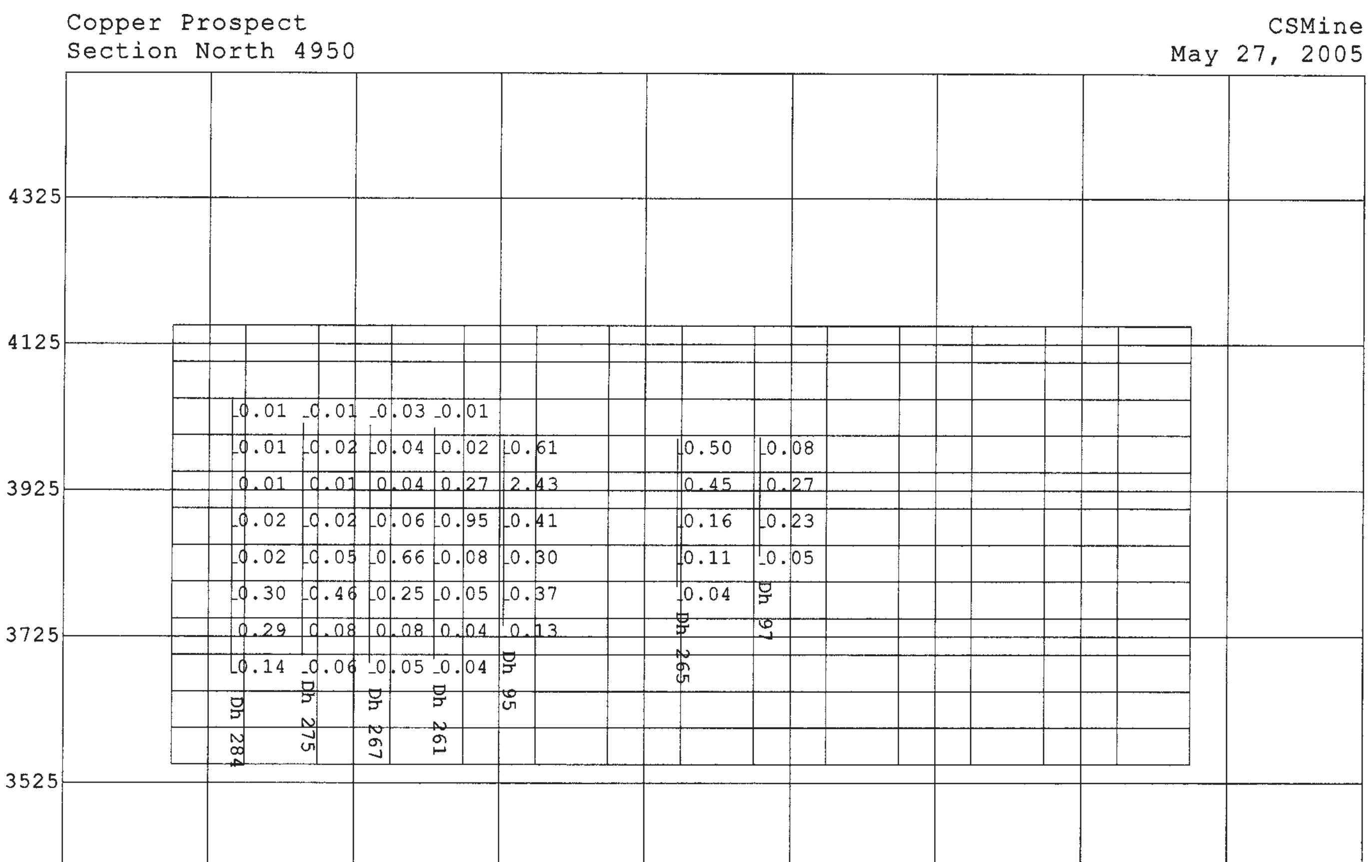
The maximum number of composites that can be stored in memory is 8000.

15.3.1 *How composites are calculated*

Figure 15.4 illustrates the concept of compositing with a vertical hole. Composites are calculated using a weighted average of the n assays that fall within the composite interval by the formula:

$$C = \frac{\sum_{i=1}^{n} A_i L_i}{\sum_{i=1}^{n} L_i}$$

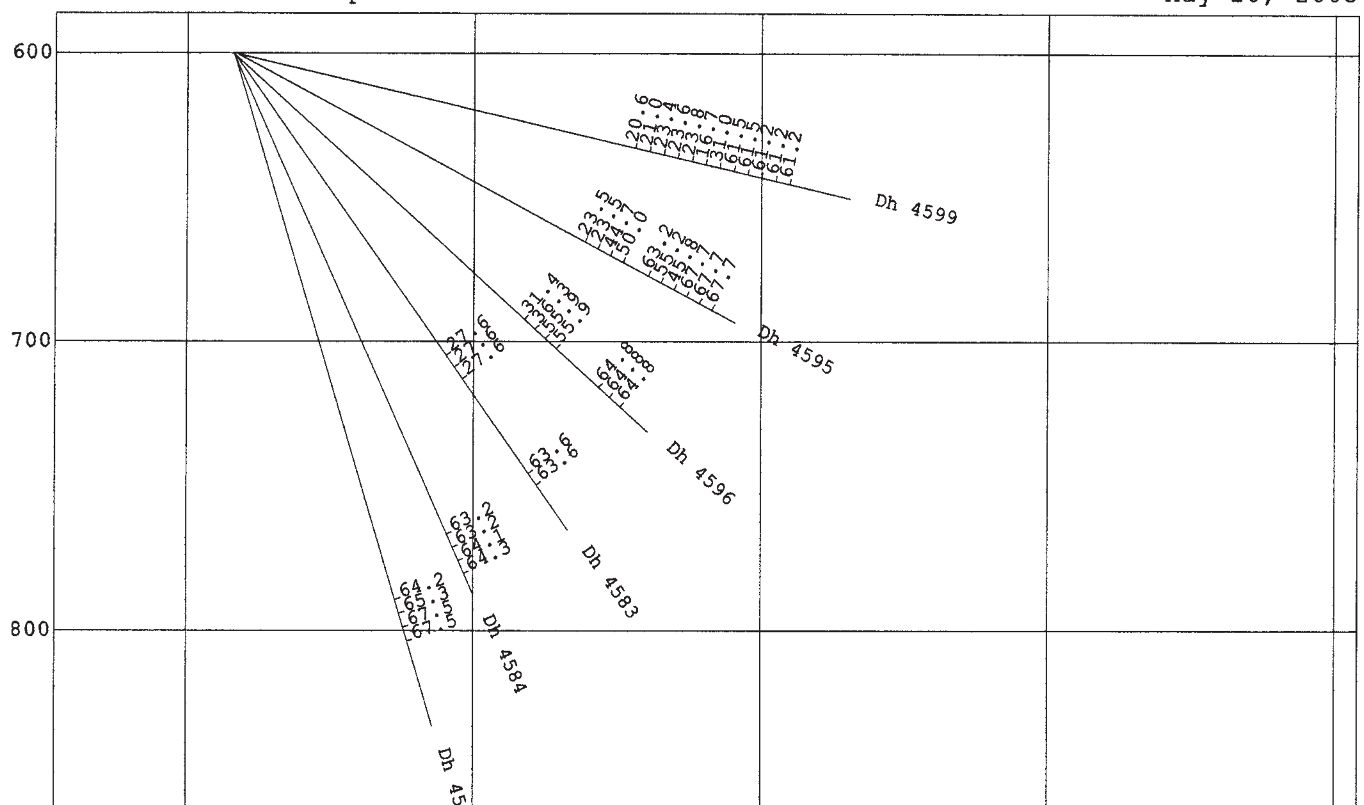
where C is the composite value, A_i is the value of assay i, and L_i is the length of assay i.



Copper Prospect
Section North 4950
CSMine
May 27, 2005
4325
4125
3925
3725
3525
0.01 0.01 0.03 0.01
0.01 0.02 0.04 0.02 0.61 0.50 0.08
0.01 0.01 0.04 0.27 2.43 0.45 0.27
0.02 0.02 0.06 0.95 0.41 0.16 0.23
0.02 0.05 0.66 0.08 0.30 0.11 0.05
0.30 0.46 0.25 0.05 0.37 0.04
0.29 0.08 0.08 0.04 0.13
0.14 0.06 0.05 0.04
Dh 284
Dh 275
Dh 267
Dh 261
Dh 95
Dh 265
Dh 97

CSMine
May 28, 2005

5 meter collar composites



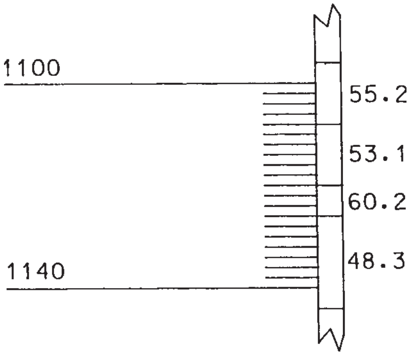


Figure 15.5. The calculation of bench composites.

The value of the 40-foot high composite from elevation 1100 to 1140 in Figure 15.5 is calculated as follows:

$$C = \frac{55.2 * 8 + 53.1 * 12 + 60.2 * 6 + 48.3 * 14}{40} = 52.9$$

If undefined assays are encountered, they are ignored. For example, if the last assay in this example were undefined, the value of the 40 foot composite would be calculated as follows:

$$C = \frac{55.2 * 8 + 53.1 * 12 + 60.2 * 6}{26} = 55.4$$

It is possible that an entire composite will be undefined. In this case the composite is discarded and not stored in memory.

The coordinates of the composite centers (X, Y, Z) are calculated by the following formulas:

$$\mathrm{X} = \mathrm{X_c} + \mathrm{Length} * \cos\,(90^\circ - \mathrm{Azimuth}) * \cos\,(\mathrm{Dip})$$
$$\mathrm{Y} = \mathrm{Y_c} + \mathrm{Length} * \cos\,(90^\circ - \mathrm{Azimuth}) * \sin\,(\mathrm{Dip})$$
$$\mathrm{Z} = \mathrm{Z_c} + \mathrm{Length} * \sin\,(\mathrm{Dip})$$

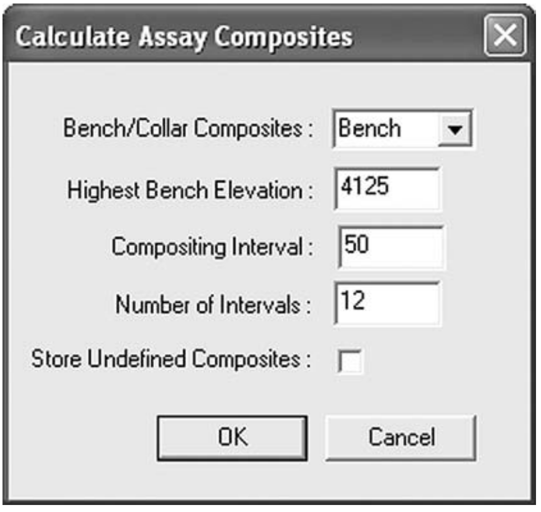
where X_c, Y_c, Z_c are the coordinates of the drill hole collar, 'Azimuth' is the azimuth of the drill hole in degrees, 'Dip' is the dip of the drill hole measured from the horizontal plane, and 'Length' is the distance from the drill hole collar to the center of the composite along the line of sight of the drill hole.

15.3.2 *Creating composites*

First, a drill hole file must be read and the collar records stored in memory. Refer to Section 15.2.2 for a description of how to read a drill hole file. The drill hole file is read again automatically in order to retrieve the assay values when the composites are calculated. If the drill hole file being used resides on a floppy disk or CD, be sure the disk with the drill hole file is in the drive specified when the drill hole file was read before starting. Next the program must be switched to the Composite mode. Refer to Section 15.1.3 for a description of how to do this. From the 'Composite Mode' menu:

SELECT: the 'Calculate' command

The 'Calculate Assay Composites' dialog will then be displayed.



When the program is first loaded, the menu parameters are initialized to default values or to the values stored in the project file if a project file was read. The values for the 'ArizCu.dhf' data set are shown. These values may be altered at any time. The most recent values will always be displayed whenever the menu is entered during a program session.

Simply changing menu values does not alter the composite values currently stored in memory. The program must be explicitly told to recalculate composites by clicking the 'OK' button located at the bottom of the dialog box.

15.3.2.1 *Compositing menu value definitions*

Bench/Collar Composites: 'Bench' or 'Collar' composites may be selected from the drop down menu. Bench composites are calculated starting from the highest bench elevation, collar composites are calculated continuously starting from each hole collar.

Highest Bench Elevation: For bench compositing, this value along with the compositing interval is required for defining the coordinates of the composite centers. The 'Highest Bench Elevation' is defined as the center of the highest bench in the planned block model. It has the same value as the Z coordinate of the key block. For each drill hole the first composite calculated will be centered at this location. Composites are calculated at increments defined by the compositing interval until the specified number have been found or the bottom of the hole has been reached. With collar compositing this value is ignored.

Compositing Interval: The compositing interval determines the spacing between the centers of the composites. It is required for both collar and bench compositing. With bench compositing, this value should be the same as the block height of the planned block model.

Number of Intervals: The value defines the maximum number of composites which will be calculated per drill hole. Compositing continues until the number of intervals specified has been calculated or the bottom of the drill hole is reached.

Number of Assay Values: Always 1 since the current version of CSMine allows for only one assay value.

15.3.2.2 *Compositing dialog commands*

To calculate the composites:

SELECT: the 'OK' button

The program will then read the drill hole file and calculate the composites as specified. With floppy based systems, be sure the disk with the drill hole file is in the drive specified when the file was read from the Drill Hole menu. An error message will be displayed if it is not. Numbers will be displayed along the bottom of the screen indicating which composite the program is currently calculating. These numbers will be cleared when the program has completed the calculations. Compositing will continue until all holes have been processed or the maximum number of composites that can be stored is reached. The maximum number of composites that can be stored is 8000.

To close the dialog box without calculating new composite values:

SELECT: the 'Cancel' button

The 'Calculate Assay Composites' dialog box will be closed and any dialog box values that were changed will be discarded.

15.3.3 *Saving composite files*

The composites currently stored in memory may be written to disk for storage or processing by other programs. The program must be in the Composite mode. If it is not, then change modes as described in Section 15.1.3. From the 'Composite Mode' menu:

SELECT: the 'File' command

To save the composites to a previously defined file, from the drop down menu:

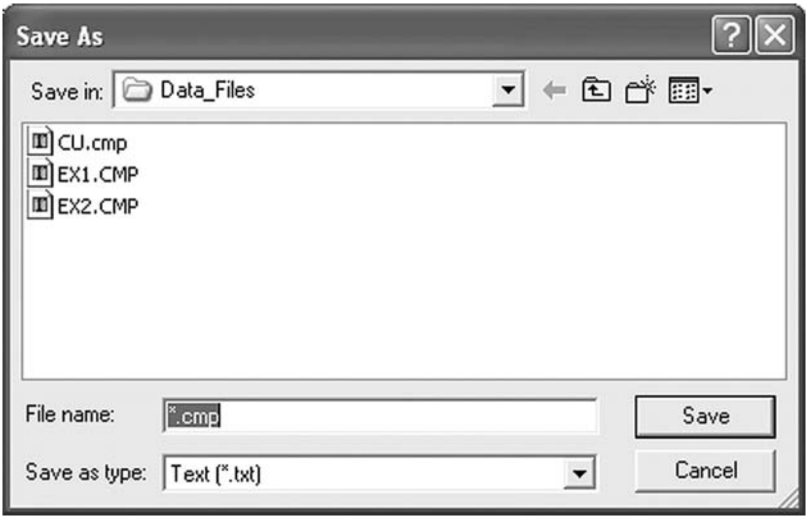
SELECT: the 'Save' command

If the composite file name has been previously defined through the earlier use of a 'Save' or 'Save As' command, or through a previously read project file, then the composite file will be saved to the defined file. If no composite file name is currently defined, the 'Save As' command will automatically be invoked.

To save the composites to a new file, from the drop down menu:

SELECT: the 'Save As' command

The standard Windows file dialog box will appear, from which the user may select an existing composite file with the extension '.cmp', or enter the name of a new file to write the composites to.



15.3.4 *Reading composite files*

A composite file that has been saved on disk can be read back into memory. The program must be in the Composite mode. If it is not, then change modes as described in Section 15.1.3. From the 'Composite Mode' menu:

SELECT: the 'File' command

From the drop down menu

SELECT: the 'Open' command

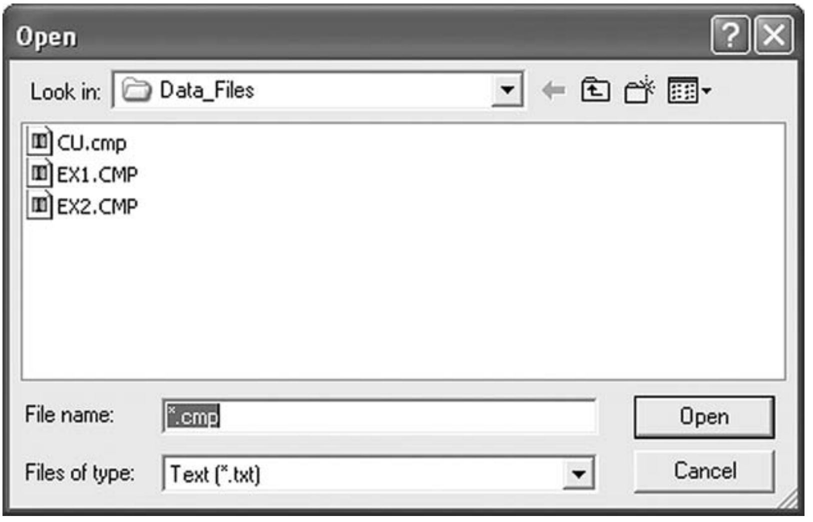
The standard Windows file dialog box will then appear from which the name of an existing composite file to read can be selected.

If there were any problems reading the file an error message will be displayed. If the file was successfully read, the composite records will be displayed in the data window.

Composites that have been read from disk in this manner can be used to create a block model. The program will also attempt to link the composites to drill holes when either a composite file or drill hole file is read. If successful, drill hole section plots will be able to display the composite values. If the linking step is unsuccessful and composites are not displayed on the drill hole section plots, the composites should be recalculated from the drill hole data currently stored in memory as described in Section 15.3.2.

15.3.5 *Composite file description*

The format of a composite file is described in this section. If only the CSMine program is used for composite manipulation, this information is not important. However, it may be



desired to study the composites with another program or even create a composite file with another program.

A composite file begins with the header lines. The first header line gives the dimension of the data set (2 for two-dimensional composite files, or 3 for three-dimensional files) and the number of assay values (always 1 for the current version of CSMine). The remaining header lines list the names of the assays.

Following the header lines are the composite records. A composite file contains up to 8000 composite records. Each record contains the following:

Hole ID: an 8 character hole ID to indicate which drill hole the composite came from.
X_m, Y_m, Z_m: the center coordinates of the composite.
Values: one assay values.

There are no rigid formatting requirements for the position of values within a record or for the number of decimal places a value contains except the following:

The hole ID must start in column 1 and it must end in column 8. CSMine is not able to read files that are not set up like this.

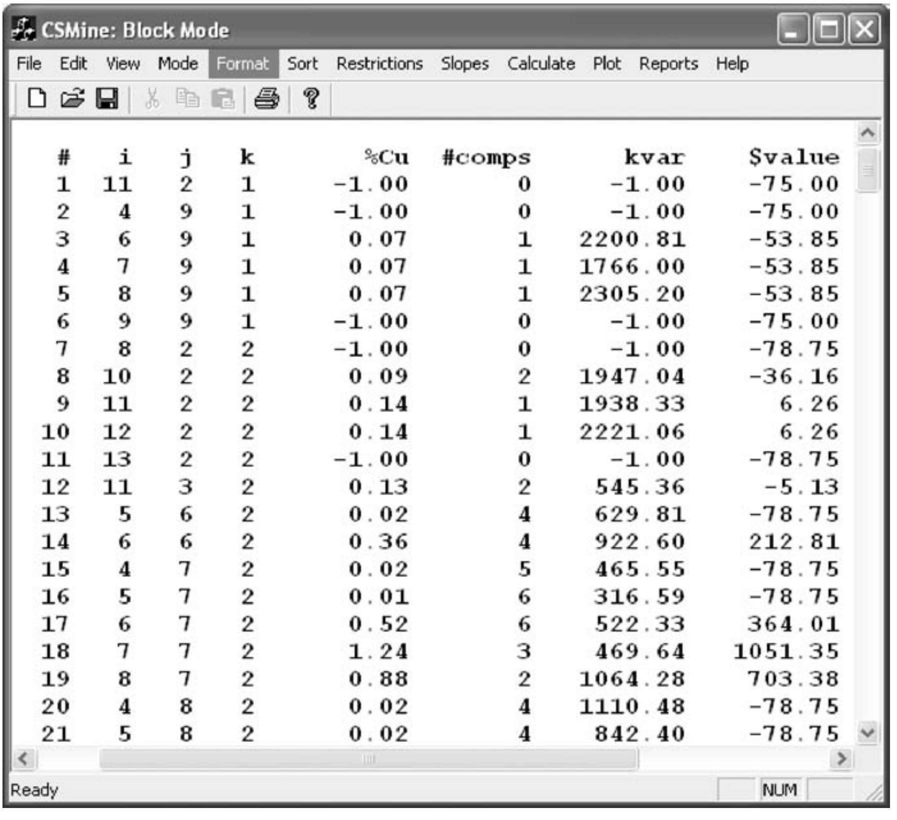
The following example shows a composite file with one assay value.

3	1				
%	Copper				
Dh	49	2582.00	5042.00	3975.00	0.3577
Dh	49	2582.00	5042.00	3925.00	0.1556
Dh	49	2582.00	5042.00	3875.00	0.0348
Dh	49	2582.00	5042.00	3825.00	0.0455
Dh	50	2267.00	5066.00	3975.00	0.3229
Dh	50	2267.00	5066.00	3925.00	0.2816
Dh	50	2267.00	5066.00	3875.00	0.0856
Dh	50	2267.00	5066.00	3825.00	0.0510

Dh	50	2267.00	5066.00	3775.00	0.0394
Dh	50	2267.00	5066.00	3725.00	0.0360
Dh	50	2267.00	5066.00	3675.00	0.0130
Dh	50	2267.00	5066.00	3625.00	0.0141
Dh	50	2267.00	5066.00	3575.00	0.0224
Dh	50	2267.00	5066.00	3525.00	0.0152
Dh	54	2863.00	4840.00	3975.00	0.0200
Dh	54	2863.00	4840.00	3925.00	0.0556
Dh	54	2863.00	4840.00	3875.00	0.0452
Dh	54	2863.00	4840.00	3825.00	0.0811

15.4 BLOCK MODEL MODE

The main data window for the Block mode with block values calculated from the 'ArizCu.dhf' data set is shown below. A description of block modeling, the block file format, and the commands associated with the Block mode are discussed in the sections that follow.



#	i	j	k	%Cu	#comps	kvar	$value
1	11	2	1	-1.00	0	-1.00	-75.00
2	4	9	1	-1.00	0	-1.00	-75.00
3	6	9	1	0.07	1	2200.81	-53.85
4	7	9	1	0.07	1	1766.00	-53.85
5	8	9	1	0.07	1	2305.20	-53.85
6	9	9	1	-1.00	0	-1.00	-75.00
7	8	2	2	-1.00	0	-1.00	-78.75
8	10	2	2	0.09	2	1947.04	-36.16
9	11	2	2	0.14	1	1938.33	6.26
10	12	2	2	0.14	1	2221.06	6.26
11	13	2	2	-1.00	0	-1.00	-78.75
12	11	3	2	0.13	2	545.36	-5.13
13	5	6	2	0.02	4	629.81	-78.75
14	6	6	2	0.36	4	922.60	212.81
15	4	7	2	0.02	5	465.55	-78.75
16	5	7	2	0.01	6	316.59	-78.75
17	6	7	2	0.52	6	522.33	364.01
18	7	7	2	1.24	3	469.64	1051.35
19	8	7	2	0.88	2	1064.28	703.38
20	4	8	2	0.02	4	1110.48	-78.75
21	5	8	2	0.02	4	842.40	-78.75

Block modeling is a technique used to interpolate assay values from irregularly distributed drill hole data to a regular two or three dimensional block grid. For each block in the grid, the surrounding composited drill hole assays are used to estimate the value for the entire block. Two types of interpolation techniques are available in the CSMine program: Inverse Distance Squared (IDS) and kriging.

The basic steps involved in creating a block model are:

1) Composite drill hole data to samples of equal length.
2) Define the block model grid size and location.
3) Determine the surface elevation for each block in the block model grid.
4) Assign each block in the model a grade.

The block model can then be used to estimate the grade and tons contained within a deposit. Economic values can be assigned to the blocks to aid in determining final pit limits and pit sequencing.

The remainder of this section describes how the block model is developed and used. A number of techniques that can be used to graphically display the block model results are discussed in Section 15.7 through Section 15.10.

15.4.1 *Defining the block model grid*

The block model is defined by three sets of parameters:

– the key block coordinates (K_x, K_y, K_z),
– the block size (B_x, B_y, B_z), and
– the number of blocks in the X, Y, and Z directions (N_x, N_y N_z).

Figure 15.6 shows a block model grid with the key block and coordinate system. The coordinates of each block are determined from the coordinates of the key block and the block size. The extent of the block model is controlled by specifying the number of blocks in the X, Y, and Z directions.

Blocks are displayed in the data window in *i*, *j*, *k* format. With this format

i denotes block count along the X axis,
j denotes block count along the Y axis, and
k denotes block count along the Z axis.

The *i*, *j*, *k* format is also used with block disk files.

The size of the block model is restricted to a maximum of 8000 blocks in plan ($N_x * N_y$), 255 blocks in the Z direction (N_z) and a maximum total number of blocks of 32000 ($N_x * N_y * N_z$).

15.4.2 *Surface topography*

The surface topography file is used to correctly evaluate blocks that lie at or near the surface. Blocks in the block model grid that lie above the surface are assigned a grade of −2 when block grades are assigned. The surface topography file is optional. However, the program cannot correctly determine which blocks lie above the surface without it.

The surface topography file contains the elevations at the center of each block in plan view throughout the block model. Figure 15.7 shows a contour map of the surface topography through the 'ArizCu.dhf' data set. Figure 15.8 shows a plan view of the block model with

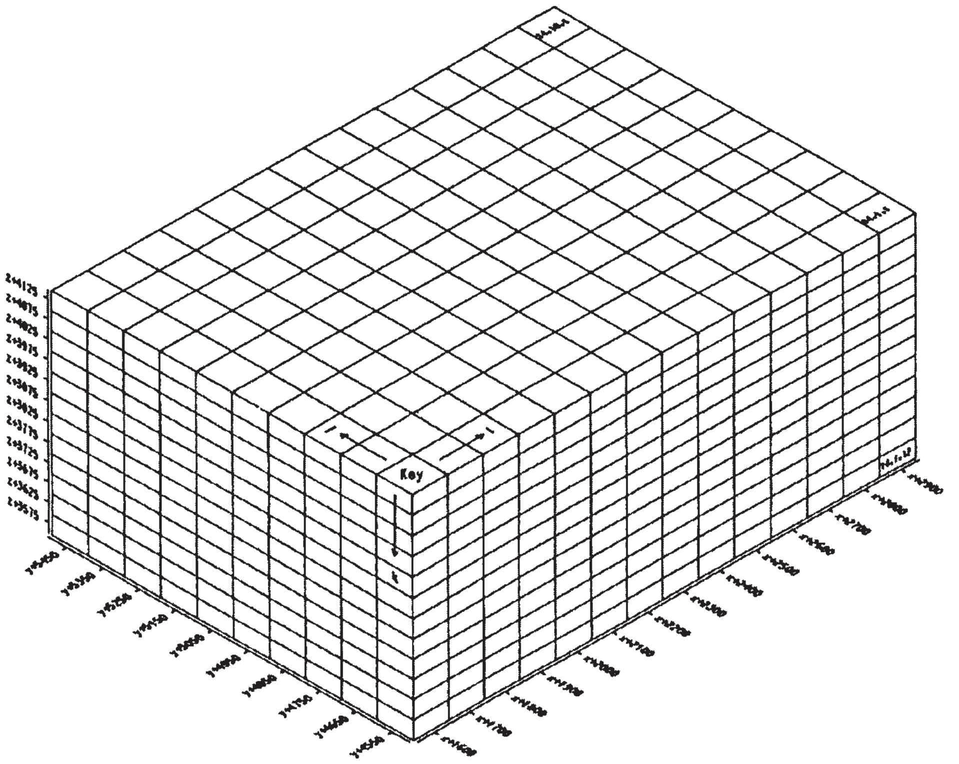


Figure 15.6. Block model grid definition.

surface elevations shown for the grid. It is this file that must be created and then read by the program.

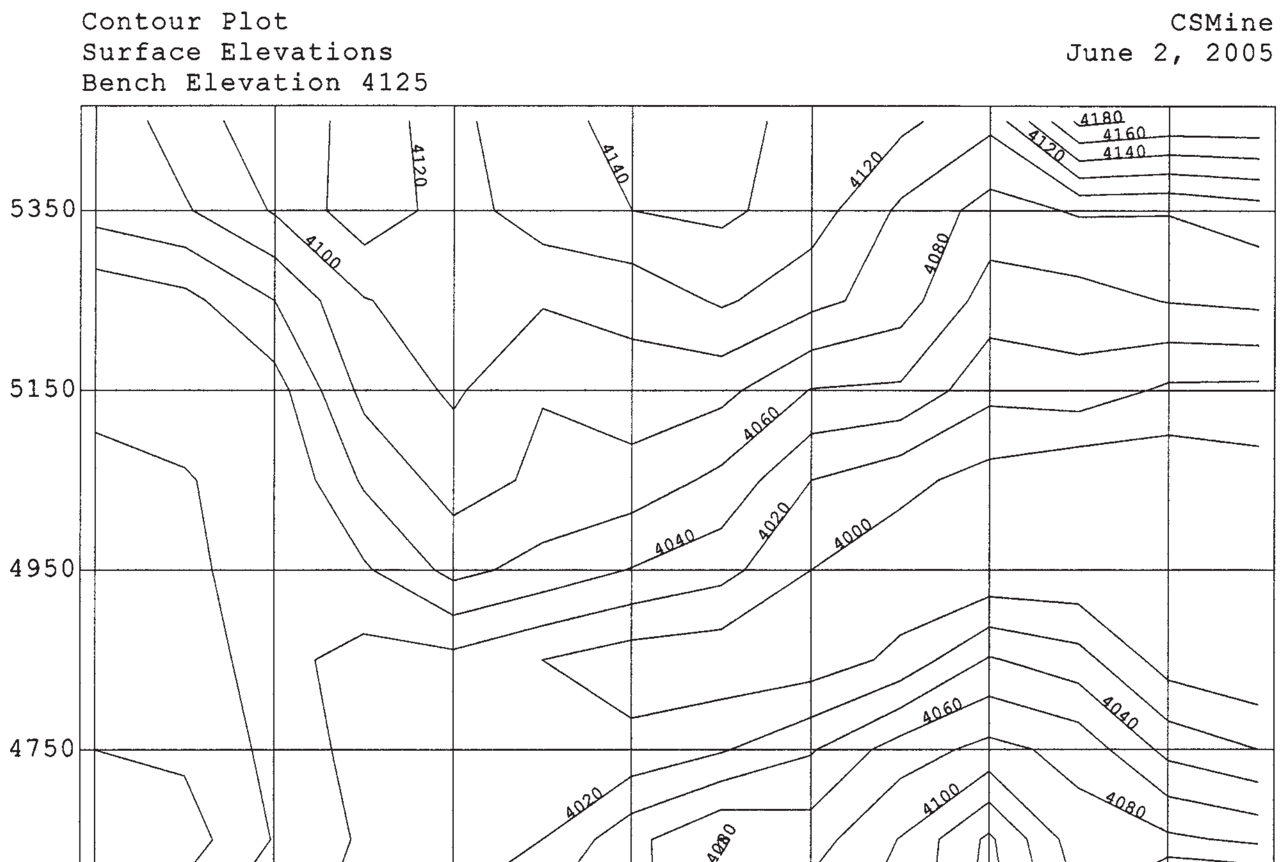
The name of the surface topography file is entered into the 'Block Values by Inverse Distance Squared/Kriging' dialog box. The file is read automatically when block grade calculations are performed. The surface topography file name can be left blank if no surface file is to be used.

15.4.2.1 *Surface topography file description*

The surface topography is entered in an *i*, *j*, *k* format identical to that used with block files as described in Section 15.4.7. The first line of the file may be left blank or it may contain the block file header. If the file contains the block header, it may also be read as an ordinary block file. This allows one to create contour maps and block plots of the topography as described in Section 15.8. The block file header is unnecessary if the data are to be used only for surface topography file definition. However, the first line of the file must be blank or contain the header!

Each record in a file contains the following:

i index: The integer index of the block count along the X axis.
j index: The integer index of the block count along the Y axis.


Contour Plot
Surface Elevations
Bench Elevation 4125
CSMine
June 2, 2005
5350
5150
4950
4750
4180
4160
4140
4120
4100
4080
4060
4040
4020
4000

Block Plot
Surface Elevations
Bench Elevation 4125

CSMine
June 2, 2005

	4065	4091	4112	4125	4115	4135	4145	4145	4135	4125	4105	4185	4175	4175
5350	4068	4078	4102	4133	4111	4131	4140	4144	4130	4096	4072	4082	4081	4090
	4025	4034	4060	4099	4109	4102	4106	4123	4106	4090	4050	4052	4061	4065
5150	4031	4021	4031	4086	4103	4081	4092	4086	4059	4057	4026	4032	4016	4015
	4050	4043	4020	4063	4089	4076	4072	4055	4020	4006	3992	3981	3984	3991
4950	4050	4050	4018	4037	4066	4053	4039	4027	4000	3987	3982	3983	3976	3980
	4060	4056	4026	4013	4014	4000	3989	3986	3988	4005	4042	4028	3990	3980
4750	4060	4056	4035	4012	4001	4002	4006	4018	4038	4070	4086	4074	4034	4020
	4080	4070	4039	4017	4009	4020	4054	4081	4071	4101	4144	4088	4084	4075
4550	4055	4045	4025	4015	4025	4045	4055	4075	4095	4105	4150	4105	4165	4165

k index:	The integer index of the block count along the Z axis. For surface topography files this index is always a 1.
Elevation:	The elevation of the center (in plan) of the block at this location.

One record for each block (in plan view) of the proposed block model must be present in the surface topography file. Thus if in plan the block model consists of 14 blocks (*i* index) having a width of 50 ft and 10 blocks (*j* index) having a length of 50 ft, the topography file must also consist of a file with an elevation for each of these blocks. The elevation block dimensions are the same as those used in to define the grid and the plan locations are the same.

15.4.2.2 *Example of surface topography file*

A portion of the surface topography file 'ArizElev.elv' provided with the distribution disk is shown below.

```
1600.0  4550.0  4125.0  100.0  100.0  50.0  14  10  1  1  0
   1 1 1  4055
   2 1 1  4045
   3 1 1  4025
   4 1 1  4015
   5 1 1  4025
   6 1 1  4045
   7 1 1  4055
   8 1 1  4075
   9 1 1  4095
  10 1 1  4105
  11 1 1  4150
  12 1 1  4105
  13 1 1  4165
  14 1 1  4165
   1 2 1  4080
   2 2 1  4070
```

15.4.3 *Assigning block values*

Two types of interpolation techniques are available with CSMine: Inverse Distance Squared (IDS) and kriging. Before discussing each of these methods, some basic concepts common to both methods are discussed.

15.4.3.1 *The search ellipse*

The search ellipse controls which samples are used for estimating a block's value. The search ellipse is defined by three parameters:

– the anisotropy direction (δ),
– the anisotropy ratio (r), and
– the maximum search distance.

The anisotropy direction defines the angle in degrees between the major axis of the search ellipse and the X-axis. The anisotropy ratio defines the ratio between the minor and major axes of the search ellipse (minor/major). An anisotropy ratio of 1 would indicate that the

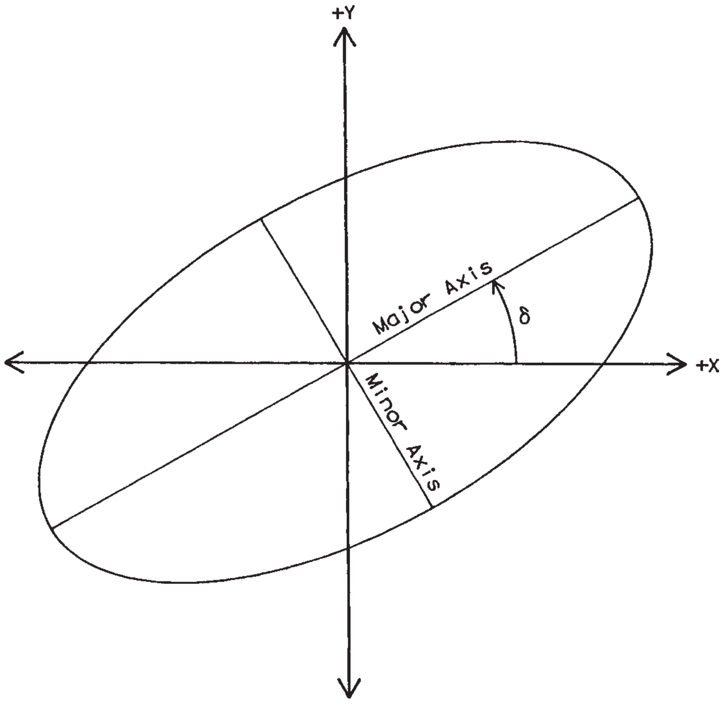


Figure 15.9. The search ellipse.

search ellipse is a circle. A ratio of 1/2 would indicate that the minor axis of the search ellipse is half as long as the major axis. In the case where the anisotropy ratio is 1 (a circle), the anisotropy direction has no effect. The maximum search distance defines the length of the major axis of the search ellipse. Figure 15.9 illustrates the three search ellipse parameters.

The distance (d) between the block center (X_b, Y_b, Z_b) and each sample (X_i, Y_i, Z_i) is adjusted for anisotropy effects by Equation 15.1:

$$d^2 = \left[(x_b - x_i)\cos\delta + (y_b - y_i)\sin\delta\right]^2 + r^2\left[(x_b - x_i)\sin\delta + (y_b - y_i)\cos\delta\right]^2 \quad (15.1)$$

15.4.3.2 *Block modeling in two and three dimensions*

Two dimensional block modeling is used with two-dimensional composite files and three dimensional block modeling is used with three-dimensional composite files.

For two dimensional block modeling, the number of blocks in the Z direction is 1 ($N_z = 1$), and the block size in the Z direction is not used ($B_z = 0$).

Three-dimensional block modeling with CSMine is performed as a series of two-dimensional models. If the exploration drill holes are vertical, then composites should be created so that the centers of the composites have the same Z coordinate as the block centers.

With angled exploration drill holes, collar compositing should be used, whereby composites are calculated at regular intervals along the length of the drill hole. This generally results

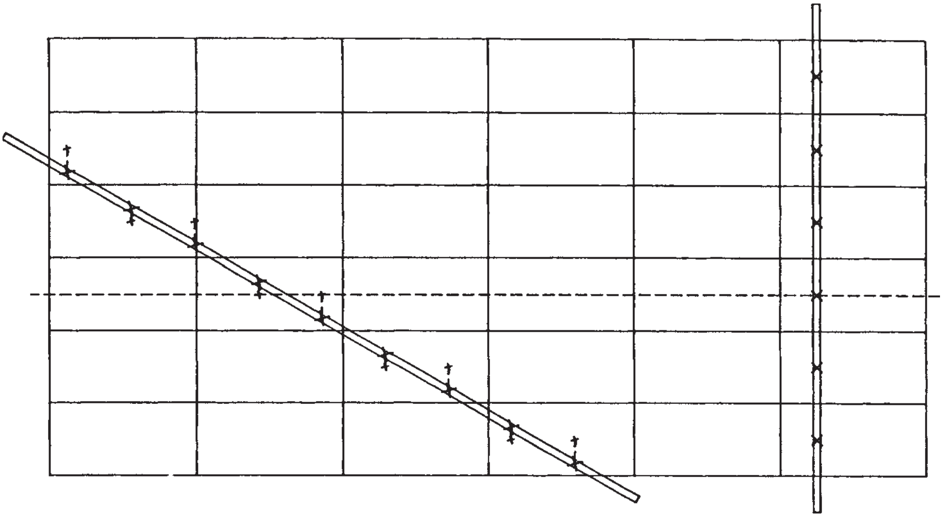


Figure 15.10. Three-dimensional block modeling with vertical and angled exploration drill holes.

in composites with Z center coordinates that do not necessarily line up with the Z coordinates of the block centers. In this case, the composites with Z center coordinates that fall within a given bench are projected onto a horizontal plane centered at the same elevation as the block centers for the bench. Two-dimensional block modeling is then performed for the bench with the projected composites. Figure 15.10 illustrates three dimensional block modeling with vertical and angled exploration drill holes.

15.4.3.3 *Block values by the Inverse Distance Squared method*

The Inverse Distance Squared method (IDS) is one of two methods that can be used to interpolate values for each block in the block model grid. The CSMine program is capable of performing both two- and three-dimensional IDS. Values are assigned to each block by weighting the samples that fall within a specified distance of the block center.

First the search ellipse must be defined. All samples that fall within the search ellipse are used to calculate the block grade. Samples that fall outside the search ellipse are ignored. The search ellipse can be defined by lengths of the major and minor axes the orientation of the ellipse and by the block height. The grade of each block is then determined by the following formula:

$$G = \frac{\sum_{i=1}^{n} G_i/D_i^2}{\sum_{i=1}^{n} 1/D_i^2} \tag{15.2}$$

where G is the block value, G_i is the value of sample i, and D_i is the distance from G_i to the block center adjusted for anisotropy effects.

Undefined blocks are handled in two ways. If a surface topography file has been read, the surface elevations are used to determine if a block lies above or below the surface. If more than one-half of a block lies above the surface no calculations are performed. Rather, the block is assigned a value of -2 to indicate the block is air. A value of -1 is assigned to blocks if no composite samples can be found within the search ellipse.

15.4.3.4 *Block values by kriging*

Kriging is geostatistical method for assigning block values. One of the several advantages kriging has over the Inverse Distance Squared method is that the estimation variance is calculated in addition to the block value. Thus, an indication of how reliable the block model values are is also obtained.

This section does not attempt to fully present the mathematics or underlying theory of kriging, but concentrates instead on describing how kriging is implemented in the CSMine program. For the theory behind geostatistics, the reader is referred to the published texts on the subject, such as those by Clark (1979) and David (1977).

The text by Clark (1979) is a short yet thorough and easily understood introduction to geostatistics while the second text by David (1977) provides a much more rigorous treatment of the subject.

Before kriging can be performed, a variogram model must first be obtained for the exploration drill hole composites. Variogram calculation and modeling is discussed in Section 15.12.

After variogram model fitting has been performed, the variogram parameters can be entered into the 'Block Values by Kriging' dialog box and kriging performed.

The types of kriging available in CSMine are:

Ordinary:	ordinary kriging,
Lognormal:	lognormal kriging,
Universal 1:	universal kriging with drift order 1, and
Universal 2:	universal kriging with drift order 2.

Ordinary kriging is the most common type of kriging used. For completeness, the derivation of the equations for ordinary point kriging as given in (David, 1977, p. 238) is reviewed.

The grade at point v having a true and unknown grade $Z(v)$ is to be estimated from n samples surrounding point v of known values $Z(X_i)$, $i = 1, \ldots, n$. The objective of point kriging is to find the weights w_i, $i = 1, \ldots, n$, that make the equation

$$Z^* = \sum_{i=1}^{n} w_i Z(x_i) \tag{15.3}$$

the 'best' estimator of $Z(v)$, where 'best' is defined as:

- the minimum variance linear estimator, and
- unbiased. This means that on average the value computed for $Z*$ should be equal to the true value of $Z(v)$ and not systematically higher or lower.

The estimation variance of $Z(v)$ by $Z*$ is given by

$$\sigma_e^2 = \sigma_v^2 - 2\sum_{i=1}^{n} w_i \sigma_{vx_i} + \sum_{i=1}^{n}\sum_{j=1}^{n} w_i w_j \sigma_{x_i x_j} \tag{15.4}$$

where σ_v^2 is the variance at a point v in the sample population, σ_{vx_i} the covariance between point v and sample x_i, and $\sigma_{x_i x_j}$ the covariance between samples x_i and x_j.

The unbiased condition means the expected value of Z^*, $E(Z^*)$, should equal the sample population mean m, which requires

$$E\left[\sum_{i=1}^{n} w_i Z(x_i)\right] = \sum_{i=1}^{n} w_i E\left[Z(x_i)\right] = m \tag{15.5}$$

Since $\quad E\left[Z(x_i)\right] = m,$ (15.6)

$$\sum_{i=1}^{n} w_i = 1$$

The problem of ordinary point kriging can thus be stated as:

$$\text{Minimize } \sigma_{\mathrm{e}}^2 \text{ subject to } \sum_{i=1}^{n} w_i = 1$$

The Lagrange principle states that to minimize a function $F(w)$ subject to the constraint that function $G(w) = 0$, the function $H(w, \lambda) = F(w) + \lambda G(w)$ should be minimized where λ is a new unknown Lagrange multiplier. Function $H(w, \lambda)$ can be minimized by setting the partial derivatives of H with respect to w and λ equal to zero and solving for the unknown w_{i}'s and λ. In terms of the variables here, the function to minimize is:

$$\begin{aligned} H &= \sigma_{\mathrm{e}}^2 + 2\lambda\left(\sum_{i=1}^{n} w_i - 1\right) \\ &= \sigma_v^2 - 2\sum_{\mathrm{i}=1}^{\mathrm{n}} w_{\mathrm{i}}\sigma_{\mathrm{vx_i}} + \sum_{i=1}^{n}\sum_{j=1}^{n} w_i w_j \sigma_{\mathrm{x_i x_j}} + 2\lambda\left(\sum_{i=1}^{n} w_i - 1\right) \end{aligned} \tag{15.7}$$

The required partial derivatives are

$$H_w = -2\sigma_{\mathrm{vx_i}} + 2\sum_{j=1}^{n} w_j \sigma_{\mathrm{x_i x_j}} + 2\lambda = 0$$

$$H_\lambda = 2\sum_{i=1}^{n} w_i - 2 = 0$$

This gives a linear system of $n+1$ equations with $n+1$ unknowns that may be rewritten as:

$$\sum_{j=1}^{n} w_j \sigma_{\mathrm{x_i x_j}} + \lambda = \sigma_{\mathrm{vx_i}}, i = 1, \ldots, n \tag{15.8}$$

$$\sum_{i=1}^{n} w_i = 1$$

In matrix form the required equations may be written as

$$[\Sigma] * [w] = [B]$$

i.e.

$$\begin{bmatrix} \sigma x_1 x_1 & \sigma x_1 x_2 & \ldots & \sigma x_1 x_n & 1 \\ \sigma x_2 x_1 & \sigma x_2 x_2 & \ldots & \sigma x_2 x_n & 1 \\ . & & & & \\ . & & & & \\ . & & & & \\ \sigma x_n x_1 & \sigma x_n x_2 & \ldots & \sigma x_n x_n & 1 \\ 1 & 1 & & 1 & 0 \end{bmatrix} * \begin{bmatrix} w_1 \\ w_2 \\ . \\ . \\ . \\ w_\mathrm{n} \\ \lambda \end{bmatrix} = \begin{bmatrix} \sigma \mathrm{vx}_1 \\ \sigma \mathrm{vx}_2 \\ . \\ . \\ . \\ \sigma \mathrm{vx}_\mathrm{n} \\ 1 \end{bmatrix}$$

All of the σ's can be obtained from the variogram and the unknown weights (w_i's) and the Lagrange multiplier (λ) can be solved for by $[w] = [\Sigma]^{-1}[B]$. Equation (15.3) can then be used to find $Z*$ and the variance of the estimate can be found by

$$\sigma_\mathrm{e}^2 = \sigma_v^2 - \sum_{\mathrm{i}=1}^{\mathrm{n}} \mathrm{w_i} \sigma_{\mathrm{vx_i}} + \lambda \tag{15.9}$$

With ordinary block kriging, the estimate and estimation variance for an entire block is calculated instead of values for a single point. This is usually done by numerical integration using a point definition grid. A 3 by 3 point definition grid as shown in Figure 15.11 is used in the CSMine program. The kriging equations for ordinary block kriging are basically the same as those for ordinary point kriging, with the exception that the variance of a point in the deposit σ_v^2 is replaced by the variance of a block in the deposit σ_b^2 and the covariance between point v and sample x_i, $\sigma_{\mathrm{vx_i}}$ is replaced by the covariance between the block b and sample x_i, $\sigma_{\mathrm{bx_i}}$,

$$\sigma_\mathrm{b}^2 = \frac{1}{\mathrm{n}^2} \sum_{\mathrm{i}=1}^{\mathrm{n}} \sum_{\mathrm{j}=1}^{\mathrm{n}} \sigma_{\mathrm{ij}} \tag{15.10}$$

$$\sigma_{\mathrm{bx_i}}^2 = \frac{1}{\mathrm{n}} \sum_{j=1}^{n} \sigma_{x_i x_j} \tag{15.11}$$

where n is the number of block grid points (9 in CSMine).

Although kriging does not require the composite data to follow any particular type of statistical distribution, lognormal kriging has been found to yield better results when the composite data distribution is highly skewed or lognormal. This is common with gold, copper, and other metals.

To use lognormal kriging, the two or three parameter lognormal model must first be determined as described in Section 15.11. The variogram of the logarithms of the data is then found and the variogram parameters entered into CSMine. Lognormal kriging can then be selected from the 'Block Values by Kriging' dialog box.

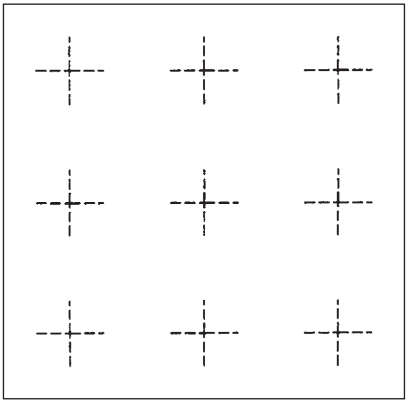


Figure 15.11. The 3 by 3 point definition grid used for block kriging in CSMine.

The kriging weights w_i are obtained by solving the kriging equations (Equation (15.8)). The three-parameter lognormal block kriging estimate with the additive constant β is found by

$$Z_L^* = \exp\left[\sum_{i=1}^{n} w_i \ln(Z(x_i) + \beta) + \frac{1}{2}\left(\sigma_v^2 - \sum_{i=1}^{n} w_i \sigma_{bx_i} - \lambda\right)\right] - \beta \qquad (15.12)$$

and the lognormal estimation variance is found by

$$\sigma_{e_L}^2 = \left(Z_L^*\right)^2 \left[\exp\left(\sigma_b^2 - \sum_{i=1}^{n} w_i \sigma_{bx_i} + \lambda\right) - 1\right] \qquad (15.13)$$

The actual kriging is performed on the lognormal transforms of the original composites. Do not replace the composite values in the composite file by the logarithms of the composites. CSMine performs the lognormal transformation automatically on the original non-transformed composite data.

Universal kriging can be used with data that have a strong trend or drift. Estimating the position of hangingwall or footwall surfaces are cases where universal kriging might be used. In addition to the variogram model, the trend model must also be determined. Universal kriging is considered a very advanced modeling technique and should be avoided if simpler models give acceptable results. For a detailed description of universal kriging, refer to (David, 1977, pp. 266–274).

In addition to the block grade estimate, CSMine also lists the number of samples used in the estimate and the estimation variance for each block. The estimation variance with block kriging is that of a two dimensional block.

15.4.4 *Creating a block model*

First a composite file must be present in memory. Composites may be calculated from drill hole data or read from disk as explained in Section 15.3. Next the Block mode must be selected. Refer to Section 15.1.3 for a description of how to do this.

From the 'Block Mode' menu:

SELECT: the 'Calculate' command

To calculate block values using the Inverse Distance Squared method, from the drop down menu:

SELECT: the 'Values by IDS' command

To calculate block values using the kriging method, from the drop down menu:

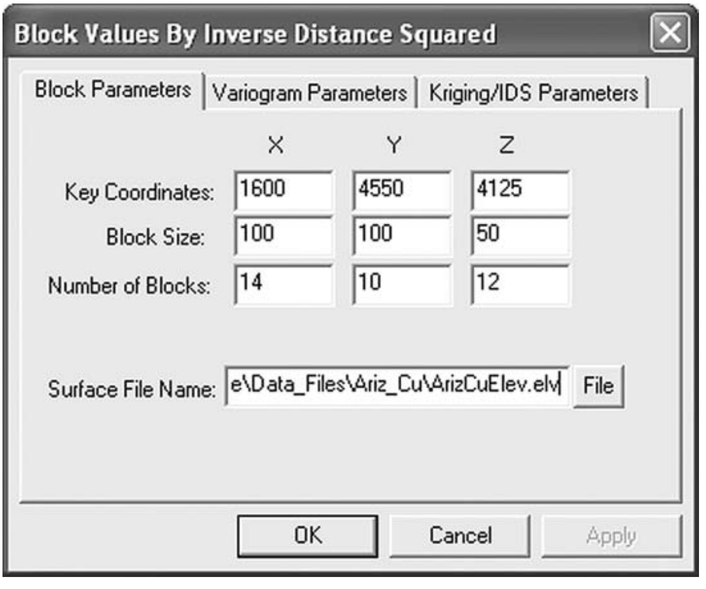
SELECT: the 'Values by Kriging' command

Depending on which option is chosen, the dialog box for the selected calculation method will be displayed. The 'Block Values by Kriging' and 'Block Values by Inverse Distance Squared' dialog boxes are basically the same. The only difference is that when IDS interpolation is selected, the menu values specific to kriging are written with a dimmer video color to indicate that these values need not be filled in for IDS interpolation. In each case the dialog box consists of three tabs. Where the items displayed differ, the tabs for both IDS and kriging will be shown in the discussion that follows.

When the program is first loaded, the menu parameters are initialized as shown for the 'ArizCu.dhf' data set. These values may be altered at any time. The most recent values will always be displayed whenever the dialog box is entered during a CSMine session. Changing the dialog box values does not alter the block model values currently stored in memory unless the block model is recalculated by clicking the 'OK' button.

15.4.4.1 *Block Parameters tab*

The values displayed in the 'Block Parameters' tab are the same for both IDS and kriging.



Key Coordinates –x, y, z: The coordinates of the key block (K_x, K_y, K_z) are used to calculate coordinates of all remaining blocks. In *i, j, k* notation the key block is at position 1, 1, 1. In *x, y, z* notation these are the actual coordinates of the center of the key block. Coordinates increase in the X and Y directions but decrease in the Z direction.

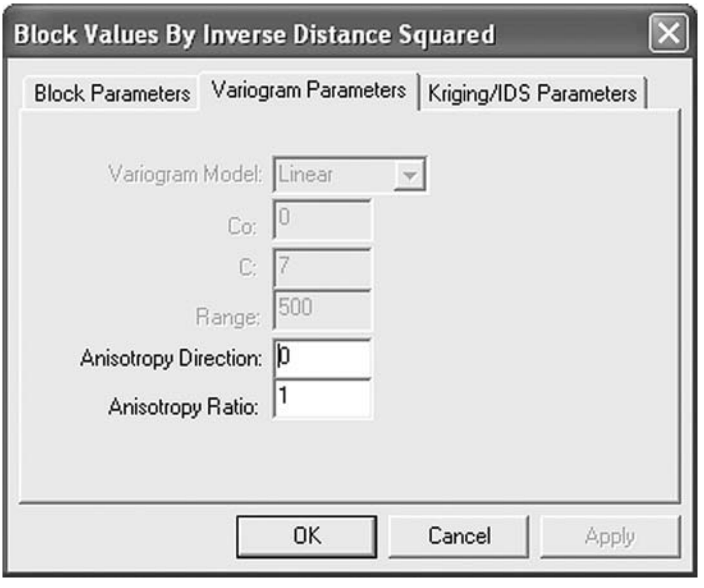
Block Size –x, y, z: The width, length, and height of a block (B_x, B_y, B_z).

Number of Blocks –x, y, z: The number of blocks along the X, Y, and Z axis (N_x, N_y, N_z). The size of the block model is restricted to a maximum of 8000 blocks in plan ($N_x \times N_y$), 255 blocks in the Z direction (N_z) and a maximum total number of blocks of 32,000 ($N_x \times N_y \times N_z$).

Surface File Name: The name of the surface topography file, (see Section 15.4.2). Leave the file name blank if no file is to be used. Press the File button to bring up the standard Windows file dialog that can be used for selecting the surface topography file. The surface topography file is read each time the block model is calculated. When using an external drive, be sure the disk with the indicated surface topography file is accessible.

15.4.4.2 *Variogram Parameters tab*

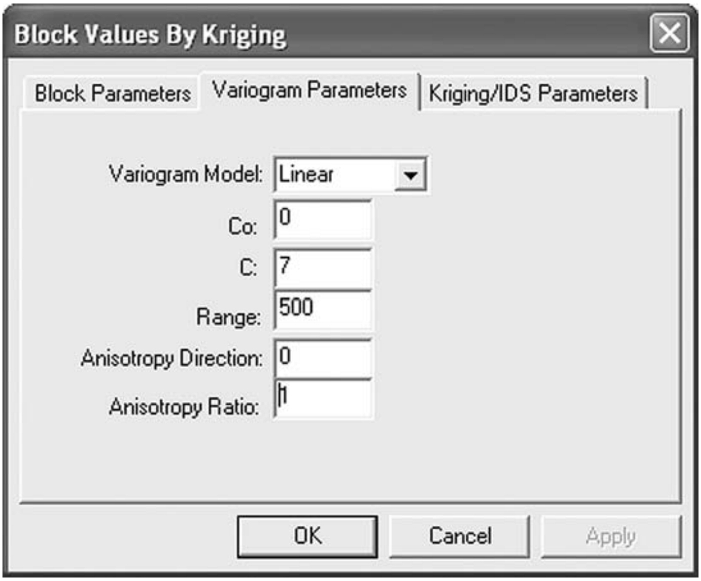
The 'Variogram Parameters' tab is used to define the anisotropy direction and ratio for both IDS and kriging, and to define the variogram parameters for kriging.



Variogram Model: The drop down menu is used to select the variogram model. Choices are currently limited to the 'Spherical', 'Linear', or 'Exponential' models.

C_0: The nugget effect *Co* for the selected variogram model.

C: The *C* value for the selected variogram model. For the linear model, the *C* value is the slope of the linear variogram.



Range: The range value (a) for the selected variogram model. For the linear model, this specifies the maximum distance over which the variogram is defined.

Anisotropy Direction: The angle in degrees the major axis of the search ellipse makes with the X-axis, measured counter clockwise from the positive X direction (East).

Anisotropy Ratio: The ratio between the minor and major axis of the search ellipse (minor/major, 0 . . . 1). A value of one indicates a circle, in which case the anisotropy direction is not used.

15.4.4.3 *Kriging/IDS Parameters tab*

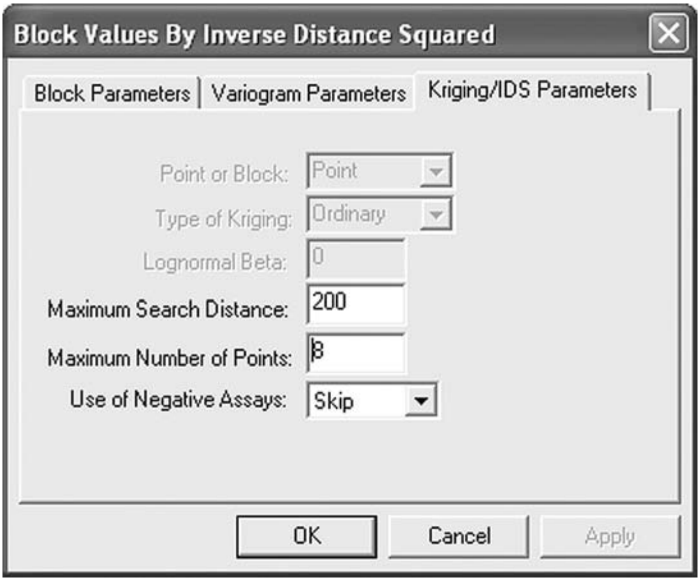
The 'Kriging/IDS Parameters' tab is used to define the type of kriging calculation to perform, and to define the search distance and number of points to use for both IDS and kriging.

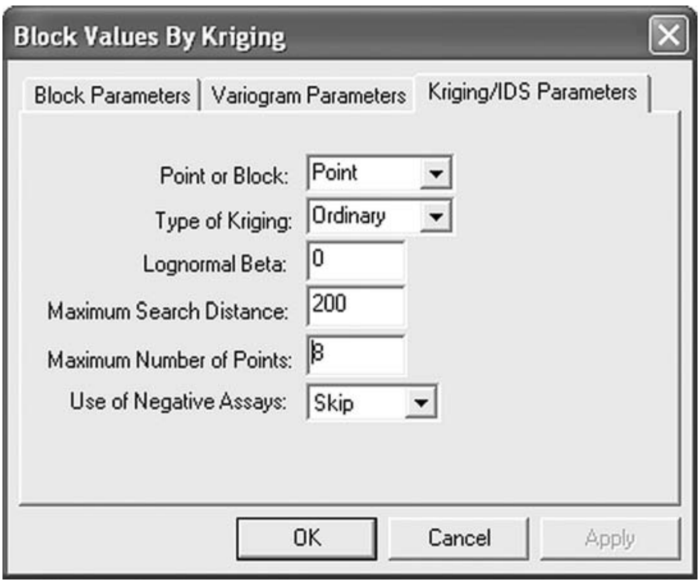
Point or Block: Specifies whether point or block kriging is to be performed. Select 'Point' or 'Block' from the drop down menu.

Type of Kriging: Specifies the type of kriging to be performed. The choice can be selected from the drop down menu. Choices available are:

Ordinary:	ordinary kriging.
Lognormal:	lognormal kriging.
Universal 1:	universal kriging with drift order 1.
Universal 2:	universal kriging with drift order 2.

Lognormal Beta: The lognormal beta (β) parameter to be used when lognormal kriging is selected. The value for the log normal beta must be calculated in a separate step as described in Section 15.11.





Max Search Distance: The length of the major axis of the search ellipse. Composites that are farther from the block center than the maximum search distance are not used in estimating a block's value.

Max Number of Points: The maximum number of composites to use when estimating a block's value. The *n* closest composites that are found using the search ellipse defined by the anisotropy direction and ratio are used.

Use of Negative Assays: The drop down menu is used to select 'Skip', 'Zero' or 'Use'. CSMine uses −1 to indicate an undefined composite. Undefined composites should not be used in estimating block values. However, it is possible to use CSMine with data files with valid negative valued samples by selecting the 'Use' option. It is also possible to convert undefined assays to a value of 0 using the 'zero' option.

15.4.4.4 *Kriging/IDS dialog commands*

To calculate block values and close the dialog box:

SELECT: the 'OK' button

The program will then read the surface topography file specified. When using an external drive, be sure the disk with the indicated surface topography file is accessible. An error message will be printed if a surface topography file is specified and cannot be read. Numbers will be displayed along the bottom of the screen indicating which block the program is currently calculating. These numbers will be cleared when the program has completed the calculations.

To calculate block values and without closing the dialog box:

SELECT: the 'Apply' button

The calculations will proceed as described above, however the dialog box will remain open. This option will only become available if a dialog box item is modified.

To close the dialog box without calculating block values:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

15.4.5 *Saving a block file*

The blocks currently stored in memory may be written to disk for storage or processing by other programs. The program must be in the Block mode. If it is not, then change modes as described in Section 15.1.3. From the 'Block Mode' menu:

SELECT: the 'File' command

To save the blocks to a previously defined file, from the drop down menu:

SELECT: the 'Save' command

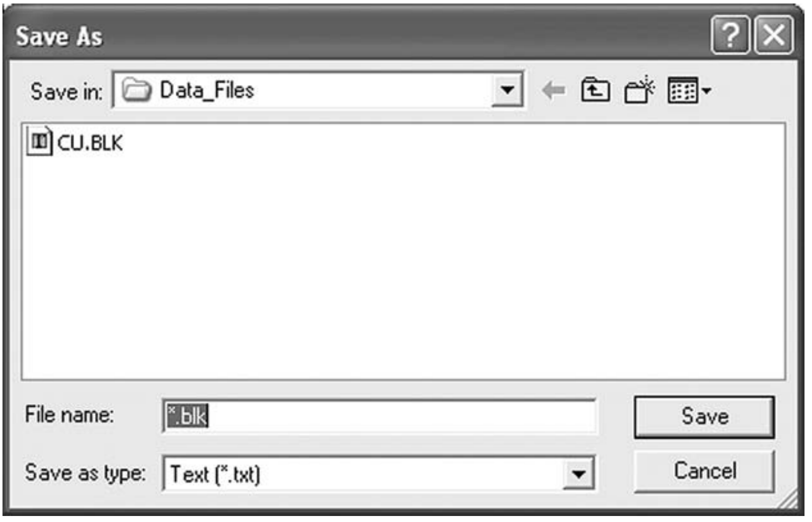
If the block file name has been previously defined through the previous use of a 'Save' or 'Save As' command, or through a previously read project file, then the block file will be saved to the defined file. If no block file name is currently defined, the 'Save As' command will automatically be invoked.

To save the blocks to a new file, from the drop down menu:

SELECT: the 'Save As' command

The standard Windows file dialog box will appear, from which the user may select an existing block file with the extension '.blk', or enter the name of a new file to write the blocks to.

Sorted or Restricted block files are saved in their unsorted and unrestricted state.



To save a sorted or restricted block file, from the drop down menu:

SELECT: the 'Save Restricted As' command

The standard Windows file dialog box will appear, from which the user may select an existing block file with the extension '.blk', or enter the name of a new file to write the blocks to. Block models written using this command cannot be read back into CSMine with the 'Open' command. They can however be imported into a spreadsheet program such as Excel.

15.4.6 *Reading a block file*

A block file that has been saved on disk can be read back into memory. The program must be in the Block mode. If it is not, then change modes as described in Section 15.1.3. From the 'Block Mode' menu:

SELECT: the 'File' command

From the drop down menu

SELECT: the 'Open' command

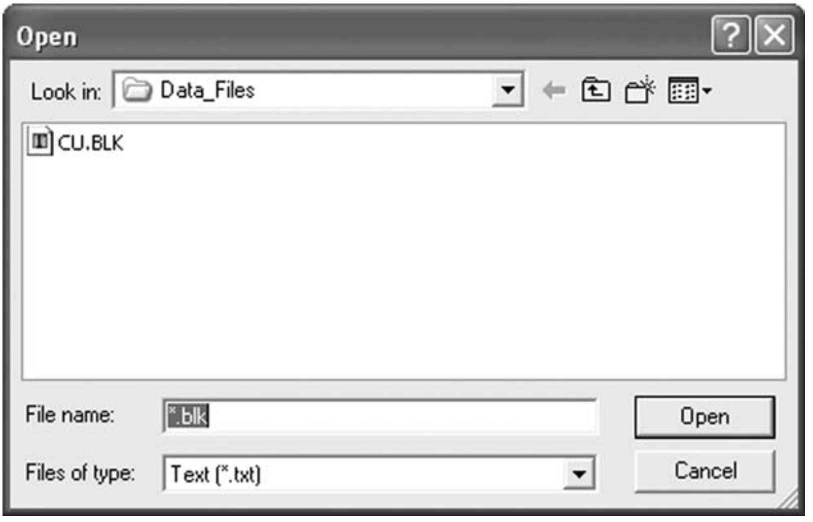
The standard Windows file dialog box will then appear from which the name of an existing block file to read can be selected.

If there were any problems reading the file an error message will be displayed. If the file was successfully read, the block records will be displayed in the data window.

15.4.7 *Block file description*

The block model file is stored in *i, j, k* format similar to the way it appears on the computer screen. The first line of the file must contain the file description header. The file header consists of:

Key *x*: the x coordinate of the key block center (K_x).
Key *y*: the y coordinate of the key block center (K_y).



Key *z*: the z coordinate of the key block center (K_z).
Size *x*: the block size in the X direction (B_x).
Size *y*: the block size in the Y direction (B_y).
Size *z*: the block size in the Z direction (B_z).
Num *x*: the number of block in the X direction (N_x).
Num *y*: the number of block in the Y direction (N_y).
Num *z*: the number of block in the Z direction (N_z).
Num *v*: the number of values per block.
Num *e*: 1 if the last value per block is the economic value for the block, 0 otherwise. Economic values are described in Section 15.5.

The remaining lines of the file contain the *i*, *j*, *k* block indices and the data values as described below:

i – the count along the X axis from the key block.
j – the count along the Y axis from the key block.
k – the count along the Z axis from the key block.
$V_1, \ldots, V_v$ – from 1 to v values per block.

An example block file is shown below:

1600.00	4550.00	4125.00	100.00	100.00	50.00	14	10	12	2	1
1 1 1	−2.00000	0.00								
2 1 1	−2.00000	0.00								
3 1 1	−2.00000	0.00								
4 1 1	−2.00000	0.00								
5 1 1	−2.00000	0.00								
6 1 1	−2.00000	0.00								
7 1 1	−2.00000	0.00								
8 1 1	−2.00000	0.00								
9 1 1	−2.00000	0.00								

10	1 1	−2.00000	0.00
11	1 1	−1.00000	−75.00
12	1 1	−2.00000	0.00
13	1 1	−1.00000	−75.00
14	1 1	−1.00000	−75.00
:			
:			
:			
1	7 5	0.01254	−90.00
2	7 5	0.01075	−90.00
3	7 5	0.02033	−90.00
4	7 5	0.14416	59.62
5	7 5	0.10156	2.11
6	7 5	0.11835	24.77
7	7 5	0.10806	10.88
8	7 5	0.16165	83.23
9	7 5	0.11429	19.29
10	7 5	0.09888	−1.51
11	7 5	0.08334	−22.49
12	7 5	0.03138	−90.00
13	7 5	−1.00000	−90.00
14	7 5	−1.00000	−90.00

15.5 ECONOMIC BLOCK VALUES

After mineral grades for each block in the model have been calculated, the economic values are assigned. Once this has been done, the economic model can be used to determine the final pit contours as discussed in Section 15.6.

The economic modeling portion of the program is based on an interactive cost data input menu. With the menu system, one can quickly change economic parameters and recalculate the final pit limits. This allows one to study how the pit limits are affected by changing economic conditions such as varying mineral price or mining costs.

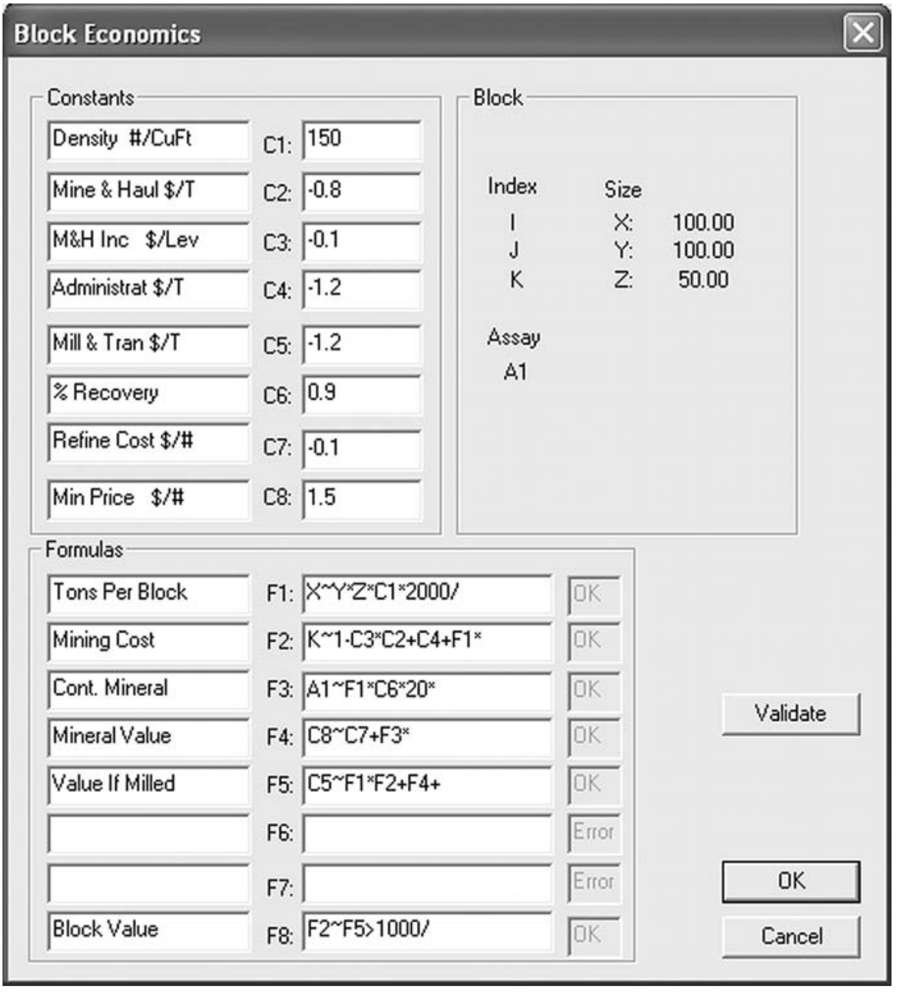
15.5.1 *How economic values are calculated*

The 'Block Economics' dialog box is used for entering formulae and for controlling the calculation of the economic block values.

The screen is divided into two parts. The top half contains constants and the bottom half contains formulae. The formulae are used to calculate the final block value.

The upper left part contains user definable labels for the constant registers which may be used in any of the formulae. The upper center part contains the user definable constant registers. The upper right part contains constant registers that are part of the data and cannot be changed. They may, however, be used in the calculation formulas.

The lower left part contains user definable labels for the formulae. The lower right part contains the actual formulas.



The syntax used to represent formulae is Reverse Polish Notation (RPN) and is very similar to that used with Hewlett Packard calculators. RPN is a very compact notation that allows for the unambiguous representation of any formula without the use of parenthesis.

A formula may consist of up to 16 functions. A function consists of a numeric constant or register letter followed by an operation. Valid numeric constants are any positive real or integer number. Valid operations are:

∼ Enter.
+ Addition.
− Subtraction.
* Multiplication.
/ Division.
> Greatest.
< Least.

Valid registers are:

C1, . . . , C8:	Defined constant registers.
I, J, K:	*i*, *j*, or *k* block index.
X, Y, Z:	Block dimensions.
A1, A2:	Block assay values.
F1, . . . , F8:	Formula registers.

Formulae are evaluated by performing each function operation on the current formula sum and the preceding register or numeric constant value. Each formula must begin with the 'Enter' operation '$\sim$' to initialize the formula sum. Consider the example

$$F1 = \frac{5 * 4}{2}$$

In the RPN notation the right hand side could be expressed as

$$5 \sim 4 * 2/$$

In words this means that the number 5 is entered and the initial sum formed. This is then multiplied by the number 4 and the new sum formed, finally the current sum is divided by 2. The steps are

5$\sim$	Sum = 5
4*	Sum = 20
2/	Sum = 10

Formula F8 is always used to assign each block a value. It is mandatory that formula F8 contain a valid formula. The other formula registers may be used to store intermediate results. Formula registers can be used in formulae that come after each formula. For example, formula F2 can use register F1, but formula F1 can not use register F2.

Invalid or undefined formulae will have the word 'Error' to the far right of the formula register number. Using an invalid formula register in a subsequent formula will make the subsequent formula invalid. All formulas are checked for validity when the economics menu is entered or when a formula is edited. Editing of formulas is discussed in Section 15.5.3.1.

15.5.2 *Evaluation of the default formulas*

The basic steps used to determine a block's value are:

(1) Calculate the cost to mine the block.
(2) Calculate the net value of the block if milled.
(3) Assign the block the maximum of these two values.

In terms of the RPN formulas used the procedure can be expressed as follows:

Assume:

Block size: 100 ft × 100 ft × 50 ft.
Block grade: 0.1%.
Block is on level 2.

All other values are shown in the dialog box in Section 15.5.1.

The first step is to calculate the tons per block. If one were to do this by hand, one uses the block dimensions B_x, B_y and B_z and the density expressed in compatible units. In the English system of units, if the density entered as constant C1 is expressed in lbs/ft^3, then the conversion factor of 2000 is needed to convert lbs/block to tons/block. The formula would be

$$F1 = \text{tons/block} = \frac{B_x(\text{ft}) * B_y(\text{ft}) * B_z(\text{ft}) * C1(\text{lbs/ft}^3)}{2000\ (\text{lbs/ton})}$$

Expressed in RPN it becomes

F1) Tons per block $= B_x \sim B_y * B_z * C1 * 2000/$

Substituting one finds that

$F1 = 100 \sim 100 * 50 * 150 * 2000/ = 37{,}500$ tons

The second step is to calculate the overall mining cost of the block. In this example there is a basic Mining and Haulage (M&H) cost per ton associated with each block. It is given by constant C2. In addition we have included the possibility that these costs can increase with depth in the pit. This would account for the increased haul distances and an increase in rock hardness, for example. This is handled through an incremental cost increase per level M&H Inc (ton/level). It is given by constant C3. The level is specified by the K index. Since there is no additional charge for Bench 1 ($K = 1$), the M&H Inc/level is

$(K - 1) \times C3$

There is also a General and Administration (G&A) cost assigned per ton. It is denoted by constant C4. Thus the mining cost per ton is

Mining cost/ton $= (K - I)C3 + C2 + C4$

Notice that all the costs have been assigned negative signs. To get the desired mining costs per block one simply multiplies the cost per ton by the tons per block. Thus

Mining cost/block $= ((K - 1)\ C3 + C2 + C4)F1$

For this example

Mining cost/block $= [(2 - 1)(-0.10) - 0.80 - 1.20] * 37{,}500 = -78{,}750$

In RPN this is written as

F2) Mining cost/block $= K \sim 1 - C3 * C2 + C4 + F1 *$

The third step is to calculate the metal recovered per block. It involves the use of the block grade, the recovery, the block tonnage, and depending upon the units being used for the various quantities, a units conversion factor. One must first consider the way the grade (assay) is expressed in the drill hole file since this is carried through the compositing and the block grade assignments. Often, and this is the case here, the grades (A1) are expressed in percent. The recovery is sometimes expressed as a ratio and sometimes as a percent. Here the recovery (C6) is given as a ratio

$C6 = 0.90$

The contained mineral quantity per block is then

$=$ Block grade $*$ Tons/block $*$ Recovery $*$ Conversion factor

For this example the recovered metal (lbs/ton) is

$$= \frac{A1}{100} * C6 * 2000$$

The factor of A1/100 simply converts the grade expressed as a percent into a ratio. The factor of 2000 is needed to obtain lbs per ton. The desired quantity of lbs recovered per block is just the lbs recovered per ton times the tons per block. It is

$$\text{Metal recovery (lbs/block)} = \frac{A1}{100} * C6 * 2000 * F1$$

This can be simplified to

$$\text{Metal recovery (lbs/block)} = A1 * F1 * C6 * 20$$

Substituting the grade for this example ($A1 = 0.1$) one finds that

$$\text{Metal recovery (lbs/block)} = 0.1 * 37,500 * 0.9 * 20 = 67,500\,\text{lbs}$$

In RPN the formula is

$$\text{F3) Contained mineral quantity (lbs/block)} = A1 \sim F1 * C6 * 20 *$$

The fourth step is to calculate the revenue of the contained metal in the block. It is to be expressed in \$/block. In this example, it has been assumed that the mill concentrate is sold and that the revenue received is

$$\text{Revenue (\$/1b)} = \text{Market price (\$/lb)} - \text{Refining cost (\$/lb)}$$

The market price (Min price) is expressed by constant C8 and the refining cost by constant C7. Thus the revenue received is

$$\text{Revenue (\$/lb)} = C8 - C7$$

To obtain the revenue one multiplies by the lbs of metal recovered from the block. Thus

$$\text{Block revenue (\$/block)} = (C8 - C7)F3$$

For the example

$$\text{Block revenue (\$/block)} = (1.50 - 0.10)67{,}500 = \$94{,}500$$

In RPN the formula is expressed as:

$$\text{F4) Contained mineral value} = C8 \sim C7 + F3 *$$

The fifth step is to calculate the net value of the block if it is sent to the mill rather than to the waste dump. The block having a 'mill destination' will incur the milling and possibly concentrate transportation costs (Milling & Transportation). These are represented by constant C5 and expressed in \$/ton of block material. Thus the milling and transportation cost/block would be

$$\text{Milling \& Transportation (\$/block)} = C5 * F1$$

The value of the block if milled is

$$\begin{aligned}\text{Value (\$/block)} = {} & \text{Milling \& Transportation (\$/block)} \\ & + \text{Mining cost (\$/block)} \\ & + \text{Value of recovered metal (\$/block)}\end{aligned}$$

The formula would be

$$\text{Value (\$/block)} = \text{C5} * \text{F1} + \text{F2} + \text{F4}$$

Substituting the values from the current example

$$\text{Value (\$/block)} = (-1.20 * \$37,500) - \$78,750 + \$94,500 = -\$28,650$$

In RPN this formula becomes

$$\text{F5) Value of block if milled = C5} \sim \text{F1} * \text{F2} + \text{F4} +$$

The final step is to assign a value to the block so that pit limits can be evaluated. The block has one value as waste (given by formula F2) and another as mill ore (given by formula F5). The value to be assigned is the least negative of the two. This is expressed as choose

$$\text{Maximum} \begin{bmatrix} \text{Mining cost of block} \\ \text{or} \\ \text{Value of block if milled} \end{bmatrix}$$

For this example it is to choose

$$\text{Maximum} \begin{bmatrix} -\$78{,}750 \\ -\$28{,}650 \end{bmatrix}$$

Obviously the value of $-\$28,650$ is assigned. To avoid the use of such large numbers in subsequent economic evaluations, it has been found convenient to divide the block values by a factor of 1000. This means that the economic block values are expressed in thousands of dollars ($1000's). This factor of 1000 can be easily added at the end by hand. Hence the value assigned to the block is $-\$28.65$. In RPN notation the choice of which value to assign is expressed by

$$\text{F8) Block value} = \text{F2} \sim \text{F5} > 1000/$$

15.5.3 *Creating an economic block model*

This section describes the steps required to perform the economic calculations. First a block file must be present in memory. A block file can be read from disk or calculated from composites. Next the program must be switched to the Block mode. Refer to Section 15.1.3 for a description of how to do this. From the 'Block Mode' menu:

SELECT: the 'Calculate' command

From the drop down menu:

SELECT: the 'Economic Block Values' command

The 'Economic Block Values' dialog box will then be displayed as shown in Section 15.5.1.

The economic block values are automatically written to disk when a block file is saved. These values are also read back into memory when the file is read from the disk file.

Blocks that lie above the original pit surface defined by the surface topography file are assigned a value of zero.

15.5.3.1 *Economic Block Values dialog box command buttons*

The dialog control keys are displayed along the bottom of the dialog box.

To validate the RPN formula strings

SELECT: the 'Validate' button

Each of the 8 formulae will be validated. If no errors were detected, the text 'OK' will be displayed to the right of the formula. If a formula is found to be invalid, then the text 'Error' will be displayed to the right of the formula. An error message showing the formula string and indicating the character position in the formula string where the error occurred will be displayed for the first formula found to be invalid.

To calculate the block values:

SELECT: the 'OK' button

The program will then calculate the value of each block based on the values and formulae entered into the menu. The result in formula F8 is used for each block value. The number of the block currently being evaluated is displayed at the bottom of the screen. This number is cleared away and the menu control keys redisplayed when all blocks have been evaluated. Simply changing menu values has no effect on the block value stored in memory. The program must be explicitly told to recalculate the block value (through the 'OK' command) in order to incorporate any changes.

To exit the 'Block Economics' dialog box without updating the economic block values:

SELECT: the 'Cancel' button

15.6 PIT MODELING

The block model developed using the program basically consists of a large rectangular block sub-divided into many smaller rectangular blocks. Many of the blocks are unmineable because they do not lie within feasible pit limits. The pit modeling or data restriction portion of the program permits the screening out of blocks that do not lie within certain pit limits. Blocks that have been screened out are not displayed in the Data window nor are included in summary calculations.

The program uses four types of pit limit restrictions. These are:

(1) Surface topography limits.
(2) Geometric pit limits defined by pit slopes.
(3) Limits defined by the union of all cones which apex with a positive economic block.
(4) Limits defined by the 3-D Floating Cone algorithm.

In addition to screening blocks from the data window, the pit restrictions also control which data are displayed in graphical plots of the block data. A complete discussion on block plots is contained in Section 15.7. Examples of a special type of block plot called the Pit Plot are used in the discussions that follow.

The remainder of this chapter explains each of the pit limit restrictions and how the limits are controlled.

15.6.1 *Surface topography restrictions*

The surface topography constraints reflect superimposing the original surface topography on to the block model. The surface topography is determined from the surface topography file read before the block model is created. The surface topography is also automatically recreated when the block model file is read from disk. The −2 values assigned to blocks indicate that more than half the block height is above the known surface.

When the surface constraints have been turned on, blocks lying above the pit surface are screened out. Blocks that have been screened out are not displayed in the data window nor are included in summary calculations. It could be argued that this feature of the program is unnecessary since the surface constraints should always be on. However, the ability to turn all constraints on and off has been included so that pit plots reflecting each of the four pit limits can be created. A pit plot of the surface topography for the 'ArizCu.dhf' data set is shown in Figure 15.12. The numbers printed for each grid reflect the bench elevation index of the highest block at that location. In other words, if each block in the model had the bench number written on its top surface, a bird's eye view of the original pit would appear as shown.

15.6.2 *Geometric pit limit restriction and pit slopes*

The geometric pit limits are those defined by the original surface topography and the pit slopes. These limits represent the largest pit which is physically mineable within the given block model but which does not include any economic considerations.

The pit slopes are defined in terms of the maximum number of blocks that can be mined in the downward direction (n) after moving one block horizontally. The slopes are entered as a ratio of 1:n. Note that the actual slope angle in degrees is determined by the block dimensions since the slope entered is only in terms of the number of blocks. A pit slope can be defined for each of the four cardinal directions. Entering the pit slopes into the program is described in Section 15.6.5.

Figure 15.13 shows a pit plot with geometric pit limits imposed for the 'ArizCu.dhf' data set. All slopes are given as 1:2. The numbers at each grid shows the bench number of the highest block that would remain if the pit defined by the geometric limits was mined.

When the pit slopes are changed, the pit restrictions are automatically updated since a change in pit slopes will alter all feasible pits. If the floating cone pit limit constraint is on, some delay may be encountered due to the enormous number of calculations that must be performed.

15.6.3 *Positive apexed cone limits*

The positive apexed cone limiting pit is the first of the four pit limits that incorporate block economics. This pit is defined by the union of all cones which apex at a positive valued block. The cones are determined such that they are compatible with the current pit slopes. Furthermore, the positive apexed cone limits cannot extend outside of the geometric limits.

The pit defined by this technique can be thought of as an outer bound for the true optimum economic pit. Blocks that lie outside these limits need not be considered in any further economic pit limit analysis. The primary purpose of these limits is to speed up the floating cone calculation routine. However, it was felt that knowing these limits may be of some value in the planning process.

Pit View
Surface Indicies

CSMine
June 3, 2005

	3	2	2	1	2	1	1	1	1	1	2	1	1	1
5350	3	2	2	1	2	1	1	1	1	2	3	2	2	2
	3	3	3	2	2	2	2	2	2	2	3	3	3	3
5150	3	4	3	2	2	2	2	2	3	3	3	3	4	4
	3	3	4	3	2	2	3	3	4	4	4	4	4	4
4950	3	3	4	3	3	3	3	3	4	4	4	4	4	4
	3	3	3	4	4	4	4	4	4	4	3	3	4	4
4750	3	3	3	4	4	4	4	4	3	3	2	3	3	4
	2	3	3	4	4	4	3	2	3	2	1	2	2	2

Pit View
Geometric Pit Limits

CSMine
June 3, 2005

	3	2	2	1	2	1	1	1	1	1	2	1	1	1
5350	3	4	3	3	3	3	3	3	3	3	3	3	3	2
	3	5	5	5	5	5	5	5	5	5	5	5	4	3
5150	3	5	7	7	7	7	7	7	7	7	7	6	5	4
	3	5	7	9	9	9	9	9	9	9	8	7	6	4
4950	3	5	7	8	10	10	9	9	9	9	8	8	6	4
	3	5	6	8	8	8	8	7	7	7	7	6	6	4
4750	3	4	6	7	7	6	6	6	5	5	5	5	4	4
	2	4	5	5	5	5	4	4	4	3	3	3	3	2

An outer economic bound pit plot for the 'ArizCu.dhf' data set is shown in Figure 15.14. The numbers indicate the bench number of the highest block that would remain if the pit defined by the outer economic pit limits were removed. For most grids the number shown is less than the number in the geometric limits plot, indicating that the outer economic pit bound contains less blocks than the amount which is physically possible.

15.6.4 *Three-dimensional floating cone*

Algorithms for determining final pit limits can be placed in two classes, those that generate the true optimum economic pit limits and those that do not. True optimum pit limits are those that maximize the value of the pit under the specified economic conditions. Such algorithms are characterized by large memory requirements and very long execution times.

An alternative method for generating the final economic pit limits, known as the Floating Cone method, is used in the CSMine program. Floating Cone algorithms generally require less memory and if properly programmed execute quite rapidly. The algorithm used here will take only a few seconds to find the economic pit limits with the Cu.dhf data set and default program settings. The problem with Floating Cone algorithms is that they may not find the true maximum valued pit. This results from the fact that the pit limits are determined from the union of all positive cones in the pit rather than treating the entire block model as a whole.

Care has been taken to minimize the potential for miscalculating the final pit contours by implementing the three-dimensional Floating Cone algorithm as follows:

```
For each bench do
    For each block in the current bench do
        Calculate the value of the cone over the block
        If positive then mine cone and return to start (Bench 1)
```

Figure 15.15 shows the final pit limits generated by the cone miner. The pit generated lies inside that determined from the positive apexed cone pit limits.

15.6.5 *Entering pit slopes*

This section describes how pit slopes are entered into the program. First the program must be switched to the Block mode. Refer to Section 15.1.3 for a description of how to do this. From the 'Block Mode' menu:

SELECT: the 'Slopes' command

The 'Pit Slopes' dialog box will then appear.

Pit slopes are entered as a ratio of 1:*n*. The value for *n* can be different for the North, East, South, and West face. The actual pit slope can be calculated using the value for *n* and the block size. For example, if the block size of 100 ft long × 100 ft wide × 50 ft high is used with a slope of 1:2, the resulting pit slope would be 45°.

To apply any changes made to the pit slopes and close the dialog box:

SELECT: the 'OK' button

If the pit slopes were changed, the pit limits are automatically updated for all pit restrictions currently turned on since a change in pit slopes will alter all feasible pits. If the floating cone

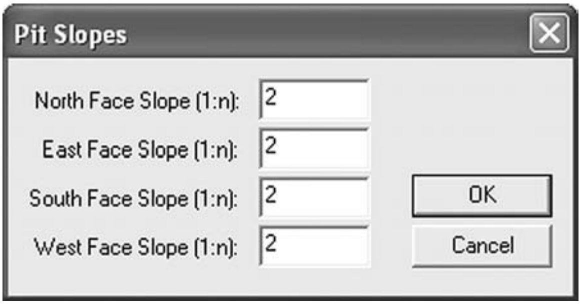
Pit View | CSMine
Outer Economic Bound | June 3, 2005

	3	2	2	1	2	1	1	1	1	1	2	1	1	1
5350	3	3	3	3	3	3	3	3	3	3	3	2	2	2
	3	4	5	5	5	5	5	5	5	3	3	3	3	3
5150	3	5	6	7	6	6	6	6	5	5	5	3	4	4
	3	5	7	8	8	7	6	6	6	7	5	4	4	4
4950	3	5	6	7	9	8	7	7	7	7	5	4	4	4
	3	4	5	7	8	8	7	7	5	5	5	3	4	4
4750	3	3	5	6	7	6	6	6	5	3	3	3	3	4
	2	3	4	5	5	5	4	4	4	3	3	3	2	2

Pit View
Floating Cone Bound

CSMine
June 3, 2005

	3	2	2	1	2	1	1	1	1	1	2	1	1	1
5350	3	2	2	1	2	2	3	3	2	2	3	2	2	2
	3	3	3	3	3	4	4	4	4	2	3	3	3	3
5150	3	4	4	5	5	5	6	6	4	3	3	3	4	4
	3	3	5	6	7	7	6	6	5	5	5	4	4	4
4950	3	3	5	7	8	8	7	7	7	5	5	4	4	4
	3	3	5	7	7	6	6	6	5	5	3	3	4	4
4750	3	3	5	5	6	5	4	4	4	3	3	3	3	4
	2	3	3	4	4	4	3	2	3	2	1	3	2	2



constraint is on, some delay may be encountered due to the large number of calculations that must be performed.

To close the dialog box without applying any changes made:

SELECT: the 'Cancel' button

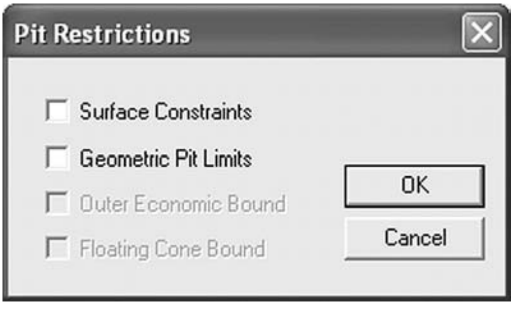
The dialog box will be closed and any changes made will be discarded.

15.6.6 *Turning pit restrictions on and off*

This section describes how pit limit restrictions are turned on and off. First the program must be switched to the Block mode. Refer to Section 15.1.3 for a description of how to do this. From the 'Block Mode' menu:

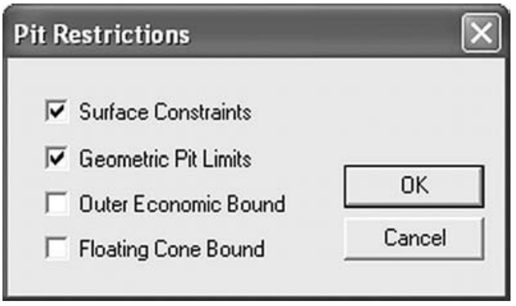
SELECT: the 'Restrictions' command

The 'Pit Restrictions' dialog box will then appear.



If economic values for the blocks have not been calculated as described in Section 15.5, then the 'Outer Economic Bound' and 'Floating Cone Bound' options will be dim. If economic block values have been calculated, then all four pit restrictions will be available.

To turn a constraint on, click the mouse in the small box next to the constraint name. A small check mark will then appear indicating that the constraint has been turned on. In the example shown above, the 'Surface Constraint' and 'Geometric Pit Limits' constraints have been turned on. Clicking on a constraint that is currently on will toggle it to the off state and the check mark will disappear.



To apply any changes made to the pit restrictions and close the dialog box:

SELECT: the 'OK' button

The dialog box will be closed and the Block mode data window will be updated to show those block records that meet the current constraints. If the floating cone constraint is turned on, some delay may be encountered while new pit limits are calculated.

To close the dialog box without applying any changes made:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made will be discarded.

15.7 BLOCK PLOTS

Once a block model has been created or read from disk, Block Plots through any bench, north section, or east section may be made. The original block grades or economic values may be selected for plotting. Only one value at a time may be plotted. The final appearance of the block plot is also controlled by the current data restriction settings.

The program must be in the Block mode. If it is not, then change modes as described in Section 15.1.3. From the 'Block Mode' menu:

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Block Plot' command

A Block Plot will be displayed using the current control variable settings.

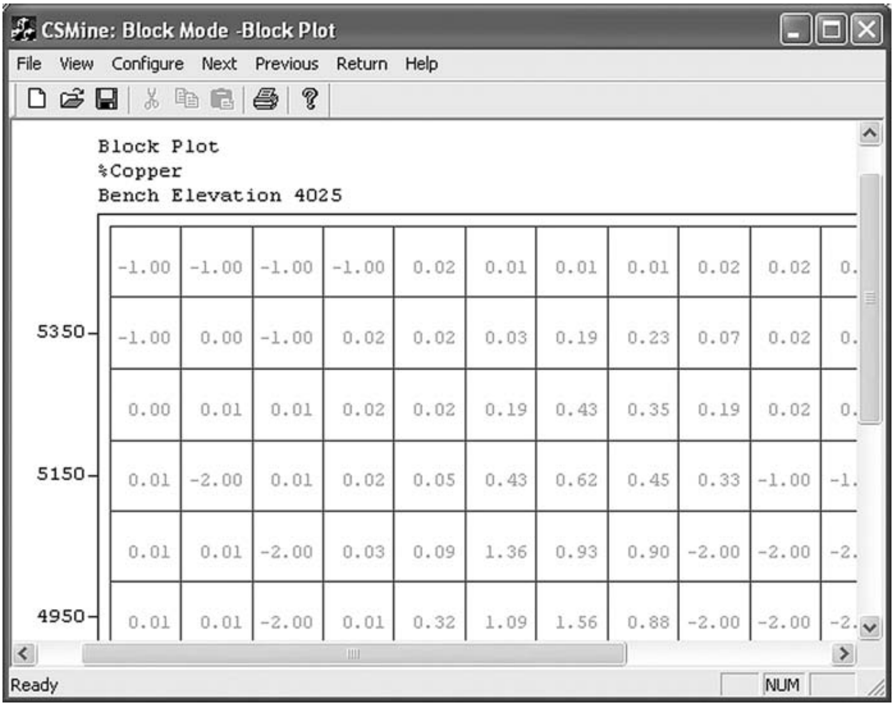
15.7.1 *The Configure command*

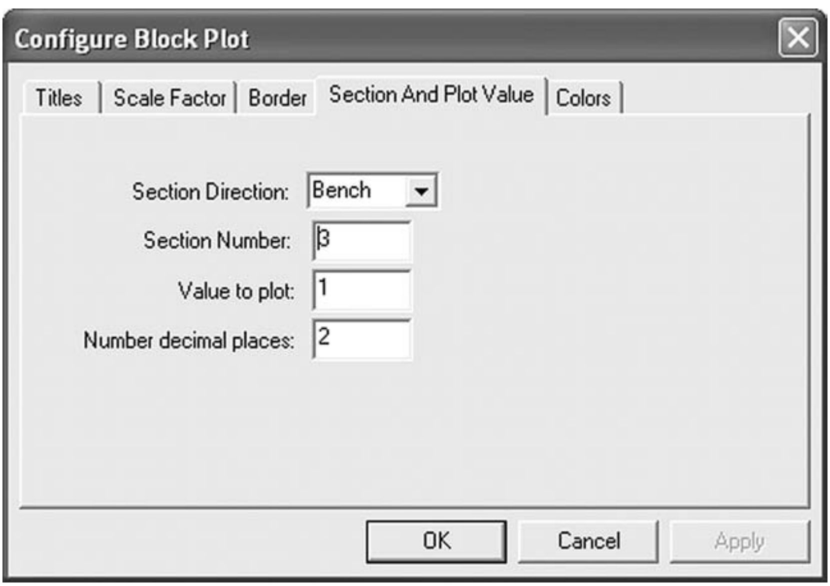
To change the Block Plot display control variables:

SELECT: the 'Configure' command

The 'Configure Block Plot' dialog box will then be displayed. The dialog box consists of five tabs. The 'Titles', 'Scale Factor', and 'Border' tabs are the same as those described for the Drill Hole Plan Map in Section 15.2.3.1. Refer to this section for a description.

The Section and Plot Value tab is used to define the section and value to plot.


CSMine: Block Mode -Block Plot
File View Configure Next Previous Return Help
Block Plot
%Copper
Bench Elevation 4025
-1.00 -1.00 -1.00 -1.00 0.02 0.01 0.01 0.01 0.02 0.02 0.
5350
-1.00 0.00 -1.00 0.02 0.02 0.03 0.19 0.23 0.07 0.02 0.
0.00 0.01 0.01 0.02 0.02 0.19 0.43 0.35 0.19 0.02 0.
5150
0.01 -2.00 0.01 0.02 0.05 0.43 0.62 0.45 0.33 -1.00 -1.
0.01 0.01 -2.00 0.03 0.09 1.36 0.93 0.90 -2.00 -2.00 -2.
4950
0.01 0.01 -2.00 0.01 0.32 1.09 1.56 0.88 -2.00 -2.00 -2.
Ready
NUM


Configure Block Plot
Titles
Scale Factor
Border
Section And Plot Value
Colors
Section Direction: Bench
Section Number: 3
Value to plot: 1
Number decimal places: 2
OK
Cancel
Apply

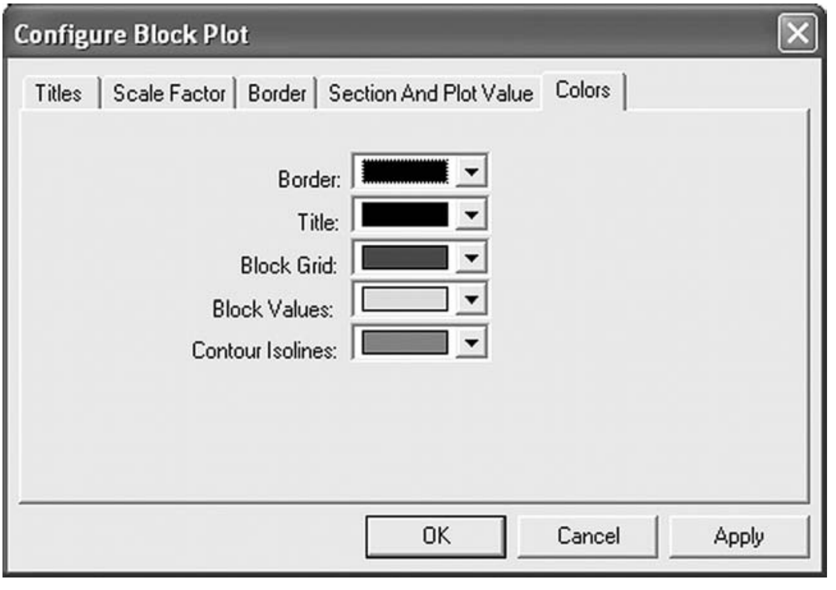
Section Direction: This determines whether North, East, or Bench sections are to be plotted. Use the drop down menu to select one of these three options. Bench plots are horizontal sections with a constant z coordinate. North specifies a vertical section as seen looking from the South to North, with the y coordinate of the section constant. East specifies a vertical section as seen looking from the East to West, with the x coordinate of the section constant.

Section Number: This is the i (East section, constant x), j (North section, constant y), or k (Bench, constant z) block index of the section or bench to plot. For example, section Bench 4 is the horizontal section through the fourth bench of the block model. Section North 4 is the vertical columns of blocks through the fourth row of the block model.

Value To Plot: This indicates which value from the block model to plot. To plot block grades, enter 1. If kriging was used to estimate the block values, a value of 2 indicates the kriging variances are to be plotted, and a value of 3 indicates the number of samples used in estimating each block is to be plotted. If economic block values rather than grade block values are being used, a value of 2 indicates that they are to be plotted.

Number Decimal Places: This is the number of digits to plot to the right of the decimal point when plotting values. For example, 2 would plot a number as N.nn, (15.43 for example).

The 'Colors' tab is used to define the colors to be used with various graphics elements. Clicking on the arrow beside a color box will cause a drop down menu with 48 predefined colors that can be selected.



Border: The color to use when displaying the plot border an axis labels.

Title: The color to use when displaying the plot title and comment.

Block Grid: The color to use when displaying the block grid.

Block Values: The color to use when displaying the block values.

Contour Isolines: The color to use when displaying the contour isolines and interval labels. This option is used with Contour Map plotting as described in Section 15.8.

15.7.1.1 *The Configure dialog commands*

To update the Block Plot and close the dialog box:

SELECT: the 'OK' button

The 'Configure Block Plot' dialog box will be closed and the 'Block Plot' redisplayed using the current control variable settings.

To update the Block Plot without closing the dialog box:

SELECT: the 'Apply' button

The Block Plot will be redisplayed using the current control variable settings and the 'Configure Block Plot' dialog box will remain open. This option will only become available if a dialog box item has been modified.

To close the dialog box without updating the Block Plot:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

15.7.2 *The Next command*

The 'Next' command is used to select the next bench or section for display. It has the same effect as increasing by one the 'Section Number' value located in the 'Section and Plot Value' tab of the 'Configure Block Plot' dialog box.

15.7.3 *The Previous command*

The 'Previous' command is used to select the previous bench or section for display. It has the same effect as decreasing by one the 'Section Number' value located in the 'Section and Plot Value' tab of the 'Configure Block Plot' dialog box.

15.7.4 *The Return command*

The 'Return' command is used to close the Block Plot window and return to the Block mode main data window.

15.7.5 *Controlling which blocks are plotted*

The current data restriction settings determine which blocks are plotted. Also, it is possible to highlight the pit outline with the two economic pit restrictions. Refer to Section 15.6 for a description of the four data restrictions and Section 15.6.6 for a description of how to turn them on and off.

Figure 15.16 shows a block plot through section North 4 of the block model created from the 'ArizCu.dhf' data set with the default program settings and all data restrictions turned

Block Plot
%Copper
Section North 4850

CSMine
June 3, 2005

4125	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00
	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00
	0.01	0.02	0.03	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	-2.00	0.06	0.05	-2.00	-2.00
	0.02	0.02	0.03	0.05	0.13	0.19	0.20	0.12	0.10	0.11	0.14	0.09	0.03	0.02
3925	0.01	0.02	0.04	0.16	0.46	0.35	0.32	0.25	0.10	0.10	0.09	0.07	0.06	0.06
	0.02	0.03	0.06	0.46	0.35	0.17	0.22	0.21	0.13	0.12	0.07	0.05	0.05	0.05
	0.03	0.11	0.10	0.15	0.18	0.24	0.13	0.12	0.20	0.17	0.05	0.03	0.04	0.05
	0.31	0.26	0.15	0.16	0.18	0.12	0.08	0.05	0.03	0.03	0.01	0.01	0.01	-1.00
3725	0.22	0.13	0.09	0.08	0.09	0.13	-1.00	0.04	0.04	0.03	0.03	0.03	0.03	-1.00
	0.11	0.08	0.06	0.05	0.04	-1.00	-1.00	-1.00	-1.00	0.02	0.02	0.02	0.02	-1.00
	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	0.02	0.02	0.02	0.02	-1.00
	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	0.02	0.02	0.02	0.02	-1.00
	1600		1800		2000		2200		2400		2600		2800	

off. The plot contains a large number of blocks with −2.00 plotted inside. The number −2.00 indicates the block is above the surface. To remove these blocks, turn on the 'Surface Constraints' restriction.

Figure 15.17 shows the resulting plot with surface constraints turned on. This drawing still contains a large number of blocks with a −1 inside, indicating that the grades for these blocks are undefined. Most of these blocks can be removed from the plot by turning on the 'Geometric Pit Limits' restriction. When this constraint is on, only those blocks that fall within the physically mineable pit (as defined by the current pit slopes) are plotted. Figure 15.18 shows the resulting plot with the geometric pit limits constraint turned on.

The final two data restrictions control whether the pit limits are to be highlighted on the block plot when the economic block values are plotted. If the 'Outer Economic Bound' constraint is turned on, a double line will be plotted at the pit boundary. All the blocks above this line are within the pit resulting from the union of all positive apexed cones. An example plot is shown in Figure 15.19. Similar results are obtained when the 'Floating Cone Bound' constraint is turned on (Figure 15.20).

15.8 CONTOUR PLOTS

Once a block model has been created or read from disk, contour plots through any bench, north section, or east section may be made. The original block grades or economic values may be selected for contouring. Only one value at a time may be contoured.

The program must be in the Block mode. If it is not, then change modes as described in Section 15.1.3. From the 'Block Mode' menu:

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Contour Plot' command

A Contour Plot will be displayed using the current control variable settings.

15.8.1 *The Configure command*

To change the Contour Plot display control variables:

SELECT: the 'Configure' command

The 'Configure Contour Plot' dialog box will then be displayed. The dialog box consists of six tabs. The 'Titles', 'Scale Factor', and 'Border' tabs are the same as those described for the Drill Hole Plan Map in Section 15.2.3.1. Refer to this section for a description. The 'Section and Plot Value' and 'Colors' tabs are the same as described for the Block Plot in Section 15.7.1. Refer to this section for a description.

The 'Contour Increments' tab is used to define the section and value to plot.

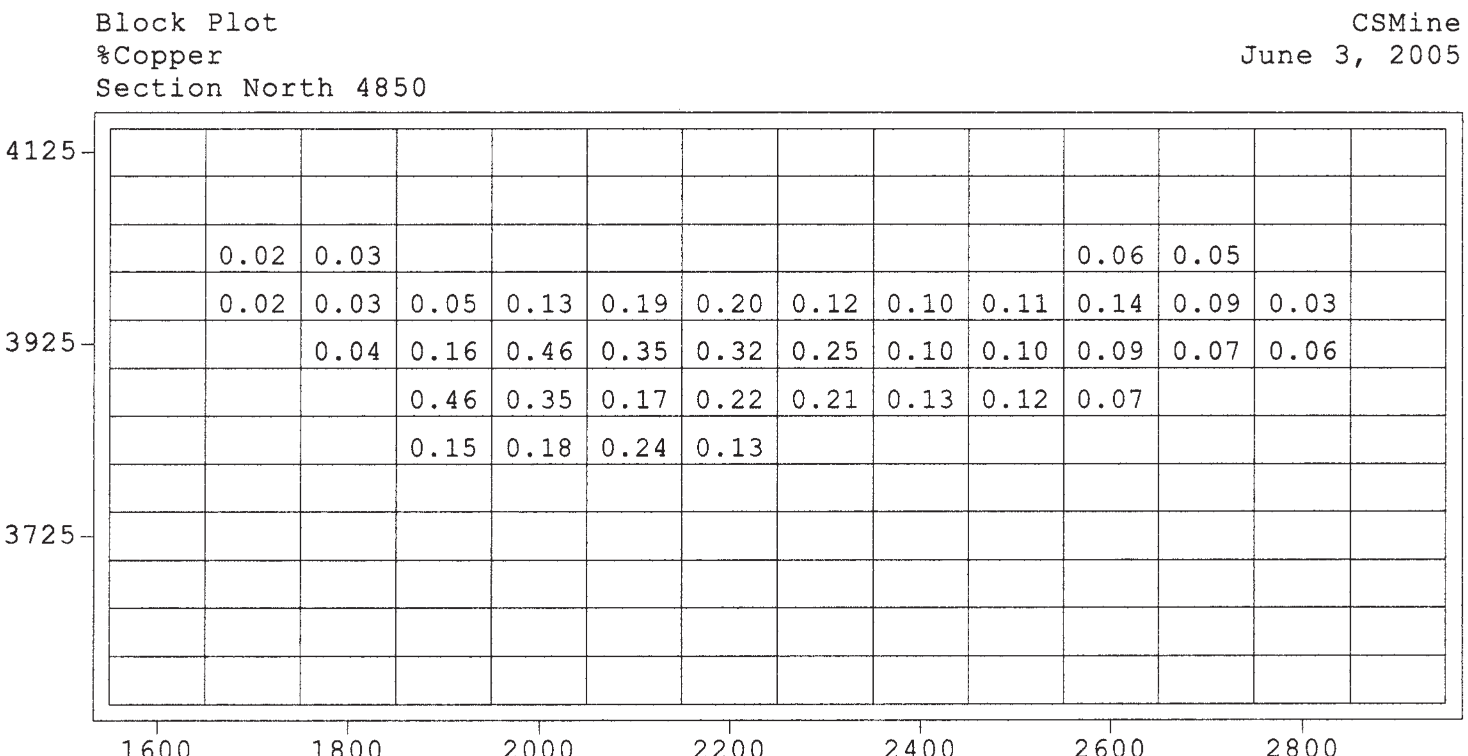
Lowest Contour Value: This is the value of the first contour to plot.

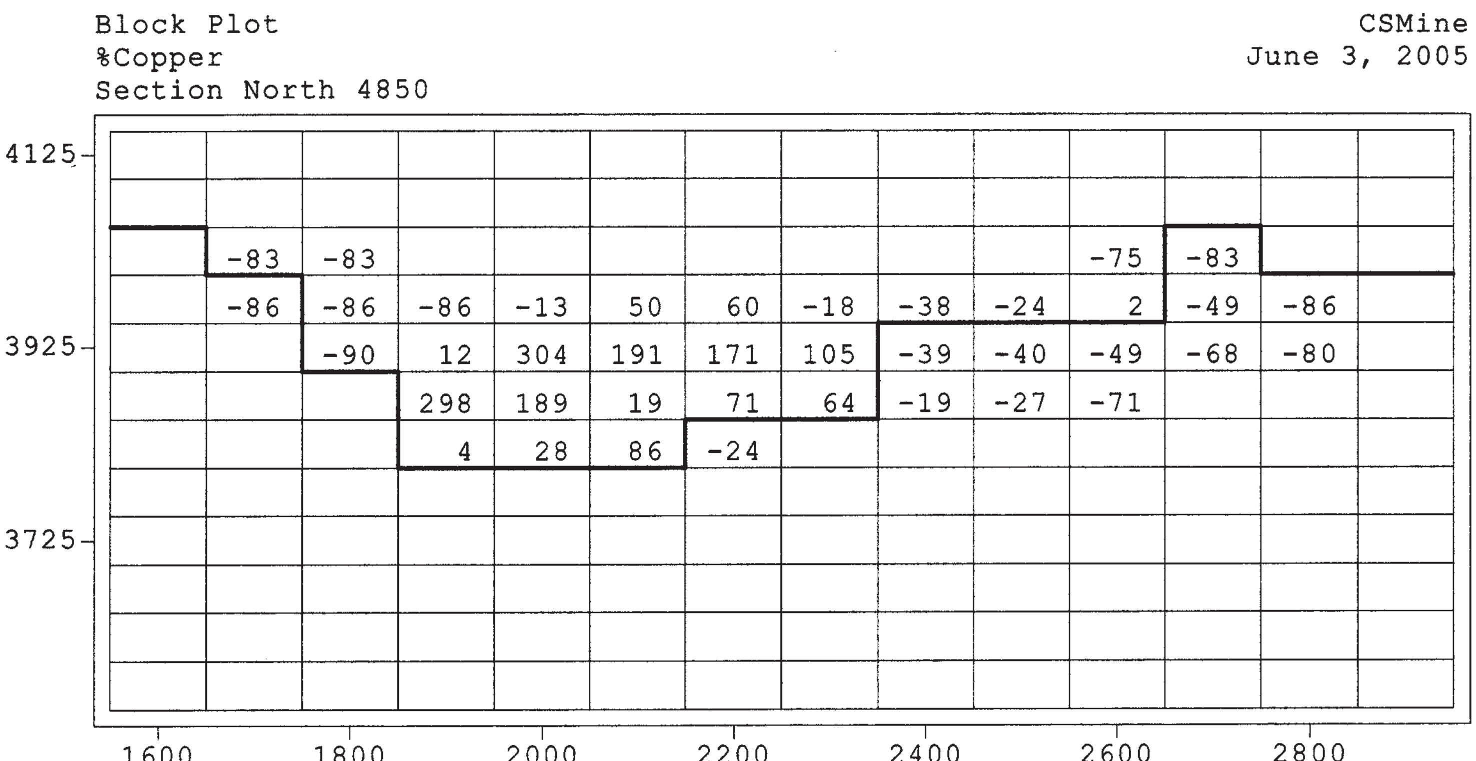
Contour Increment: This is the value between plotted contours. The contour interval is added to the lower contour value and contours plotted until the upper contour value is exceeded. This value must be greater than 0.

Block Plot
%Copper
Section North 4850

CSMine
June 3, 2005

4125														
	0.01	0.02	0.03								0.06	0.05		
	0.02	0.02	0.03	0.05	0.13	0.19	0.20	0.12	0.10	0.11	0.14	0.09	0.03	0.02
3925	0.01	0.02	0.04	0.16	0.46	0.35	0.32	0.25	0.10	0.10	0.09	0.07	0.06	0.06
	0.02	0.03	0.06	0.46	0.35	0.17	0.22	0.21	0.13	0.12	0.07	0.05	0.05	0.05
	0.03	0.11	0.10	0.15	0.18	0.24	0.13	0.12	0.20	0.17	0.05	0.03	0.04	0.05
	0.31	0.26	0.15	0.16	0.18	0.12	0.08	0.05	0.03	0.03	0.01	0.01	0.01	-1.00
3725	0.22	0.13	0.09	0.08	0.09	0.13	-1.00	0.04	0.04	0.03	0.03	0.03	0.03	-1.00
	0.11	0.08	0.06	0.05	0.04	-1.00	-1.00	-1.00	-1.00	0.02	0.02	0.02	0.02	-1.00
	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	0.02	0.02	0.02	0.02	-1.00
	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	0.02	0.02	0.02	0.02	-1.00
	1600		1800		2000		2200		2400		2600		2800	


Block Plot
%Copper
Section North 4850
CSMine
June 3, 2005
4125
3925
3725
0.02 0.03 0.06 0.05
0.02 0.03 0.05 0.13 0.19 0.20 0.12 0.10 0.11 0.14 0.09 0.03
0.04 0.16 0.46 0.35 0.32 0.25 0.10 0.10 0.09 0.07 0.06
0.46 0.35 0.17 0.22 0.21 0.13 0.12 0.07
0.15 0.18 0.24 0.13
1600
1800
2000
2200
2400
2600
2800


Block Plot
%Copper
Section North 4850
CSMine
June 3, 2005
4125
3925
3725
-83 -83 -75 -83
-86 -86 -86 -13 50 60 -18 -38 -24 2 -49 -86
-90 12 304 191 171 105 -39 -40 -49 -68 -80
298 189 19 71 64 -19 -27 -71
4 28 86 -24
1600
1800
2000
2200
2400
2600
2800

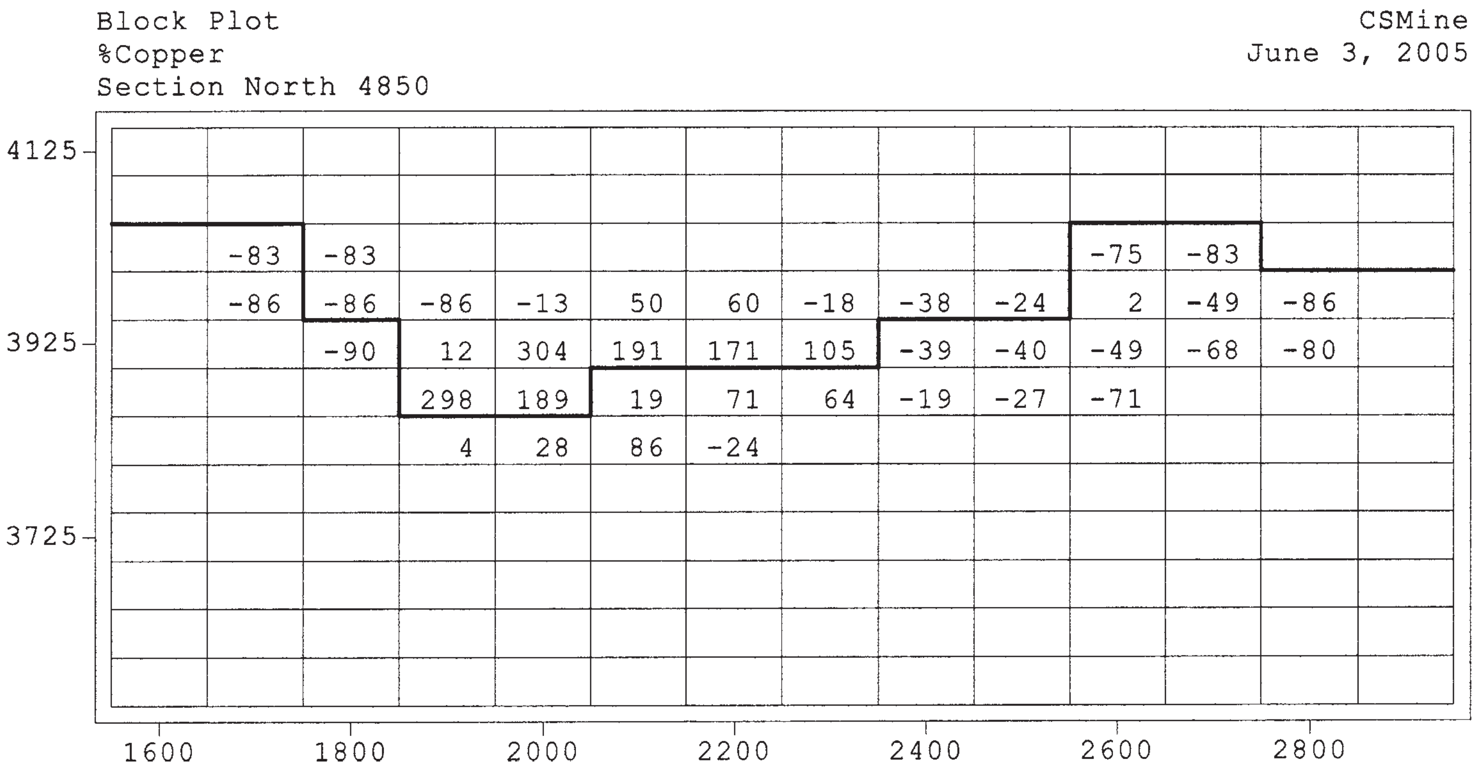
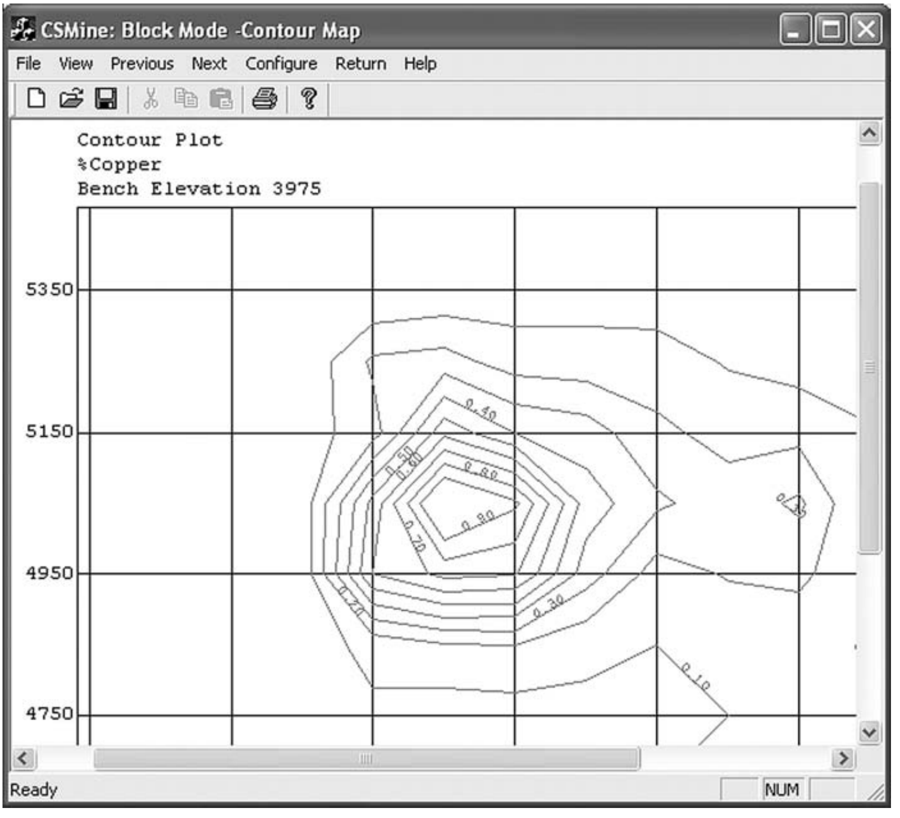
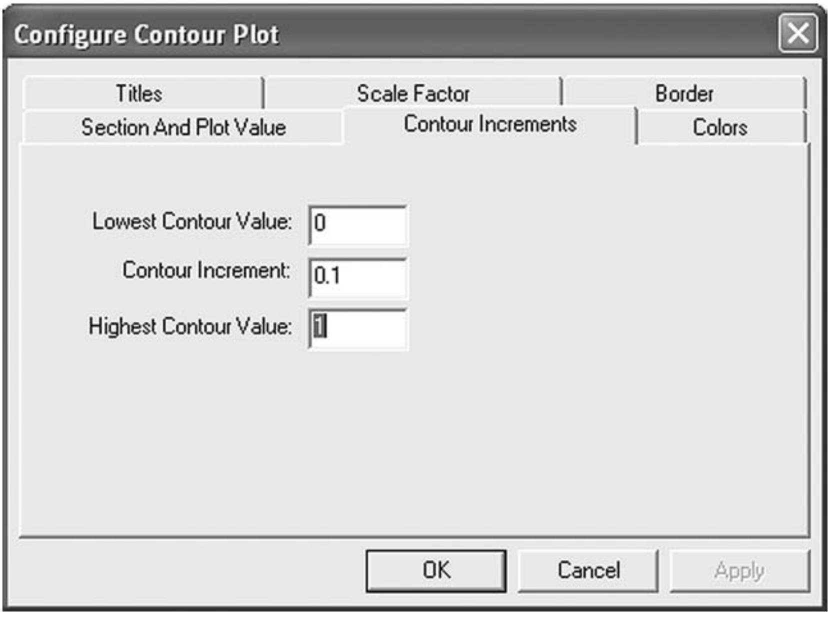


Figure 15.20. Block plot with floating cone bound.


CSMine: Block Mode -Contour Map
File View Previous Next Configure Return Help
Contour Plot
%Copper
Bench Elevation 3975
5350
5150
4950
4750
0.40
0.50
0.60
0.80
0.90
0.70
0.20
0.30
0.10
Ready
NUM


Configure Contour Plot
Titles
Scale Factor
Border
Section And Plot Value
Contour Increments
Colors
Lowest Contour Value: 0
Contour Increment: 0.1
Highest Contour Value: 1
OK
Cancel
Apply

Highest Contour Value: This is the value of the last contour to plot.

15.8.1.1 *The Configure dialog commands*

To update the Contour Plot close the dialog box:

SELECT: the 'OK' button

The 'Configure Contour Plot' dialog box will be closed and the 'Contour Plot' redisplayed using the current control variable settings.

To update the Contour Plot without closing the dialog box:

SELECT: the 'Apply' button

The Contour Plot will be redisplayed using the current control variable settings and the 'Configure Contour Plot' dialog box will remain open. This option will only become available if a dialog box item has been modified.

To close the dialog box without updating the Contour Plot:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

15.8.2 *The Next command*

The 'Next' command is used to select the next bench or section for display. It has the same effect as increasing by one the 'Section Number' value located in the 'Section and Plot Value' tab of the 'Configure Contour Plot' dialog box.

15.8.3 *The Previous command*

The 'Previous' command is used to select the previous bench or section for display. It has the same effect as decreasing by one the Section Number value located in the 'Section and Plot Value' tab of the 'Configure Contour Plot' dialog box.

15.8.4 *The Return command*

The 'Return' command is used to close the Contour Plot window and return to the Block mode main data window.

15.9 PLOTTING PIT PROFILES

Pit profile plots can be used to view the geometry of the pit with the various pit restrictions discussed in Section 15.6.

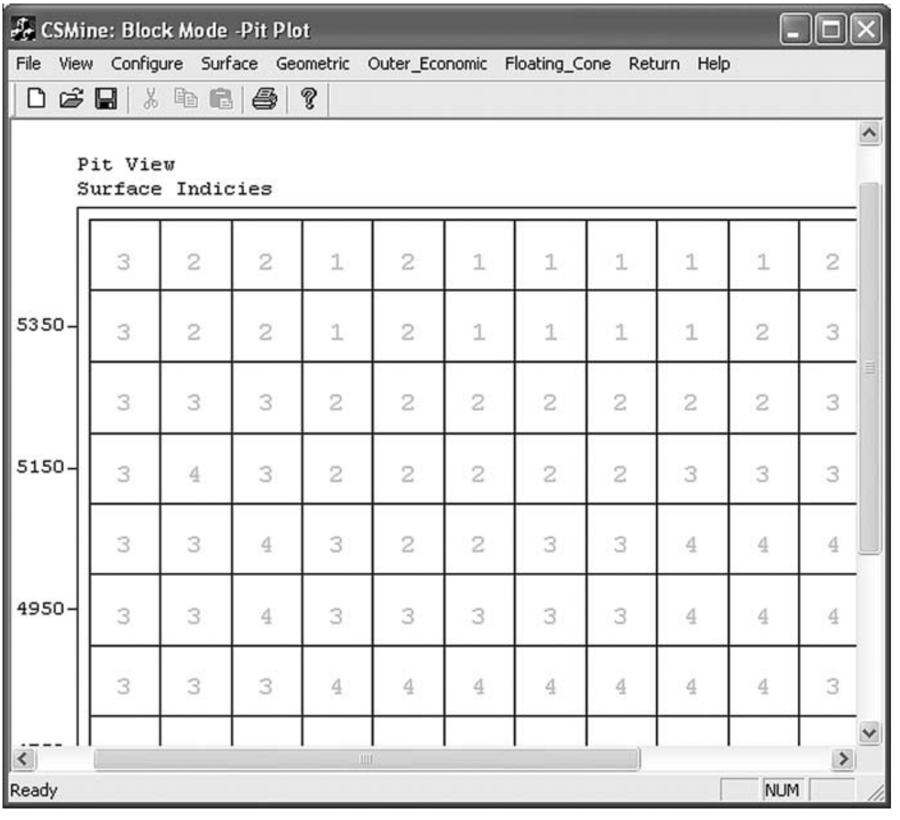
The program must be in the Block mode. If it is not, then change modes as described in Section 15.1.3. From the 'Block Mode' menu:

SELECT: the 'Plot' command

From the drop down menu:

SELECT: the 'Pit Plot' command

A Pit Plot showing the surface indices will be displayed using the current control variable settings.



15.9.1 *The Configure command*

To change the Pit Plot display control variables:

SELECT: the 'Configure' command

The 'Configure Pit Plot' dialog box will then be displayed. The dialog box consists of three tabs. The 'Scale Factor', and 'Border' tabs are the same as those described for the Drill Hole Plan Map in Section 15.2.3.1. Refer to this section for a description. The 'Colors' tab are the same as described for the Block Plot in Section 15.7.1. Refer to this section for a description.

15.9.1.1 *The Configure dialog commands*

To update the Pit Plot and close the dialog box:

SELECT: the 'OK' button

The 'Configure Pit Plot' dialog box will be closed and the Pit Plot redisplayed using the current control variable settings.

To update the Pit Plot without closing the dialog box:

SELECT: the 'Apply' button

The Pit Plot will be redisplayed using the current control variable settings and the 'Configure Pit Plot' dialog box will remain open. This option will only become available if a dialog box item has been modified.

To close the dialog box without updating the Pit Plot:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

15.9.2 *The Surface command*

The 'Surface' command is used to display a Pit Plot showing the results of applying the surface topography constraint as discussed in Section 15.6.1.

15.9.3 *The Geometric command*

The 'Geometric' command is used to display a Pit Plot showing the results of applying the 'Geometric Pit Limits' constraint as discussed in Section 15.6.2.

15.9.4 *The Outer_Economic command*

The 'Outer_Economic' command is used to display a Pit Plot showing the results of applying the 'Outer Economic Bound' constraint as discussed in Section 15.6.3.

15.9.5 *The Floating_Cone command*

The 'Floating_Cone' command is used to display a Pit Plot showing the results of applying the 'Floating Cone Bound' constraint as discussed in Section 15.6.4.

15.9.6 *The Return command*

The 'Return' command is used to close the Pit Plot window and return to the Block mode main data window.

15.10 BLOCK REPORTS

A simple report giving the total tons and grade for the block model can be displayed and printed. The report can be configured to calculate tons and grades for up to three grade categories. If economic block values have been calculated, then the total value of all blocks within each category will also be displayed.

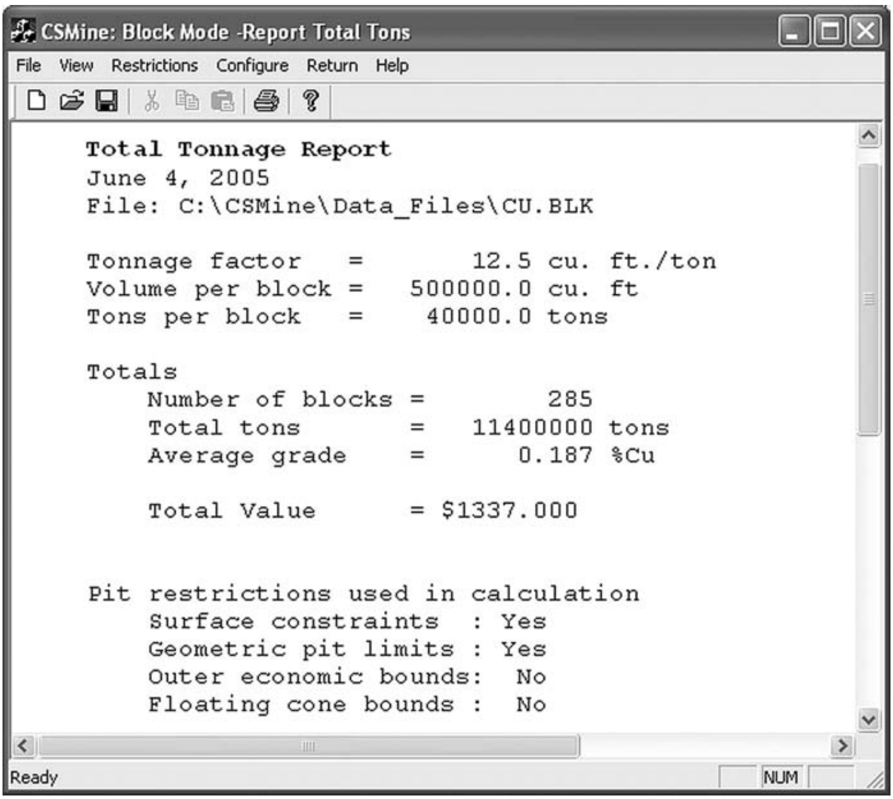
The program must be in the Block mode. If it is not, then change modes as described in Section 15.1.3. From the 'Block Mode' menu:

SELECT: the 'Reports' command

From the drop down menu:

SELECT: the 'Total Tons' command

A Total Tons Report will be displayed using the current control variable settings.



15.10.1 *The Restrictions command*

The Total Tons Report will calculate the total tons and average grade for the pit defined by the four pit restrictions. The pit restrictions used are displayed at the bottom of the report.

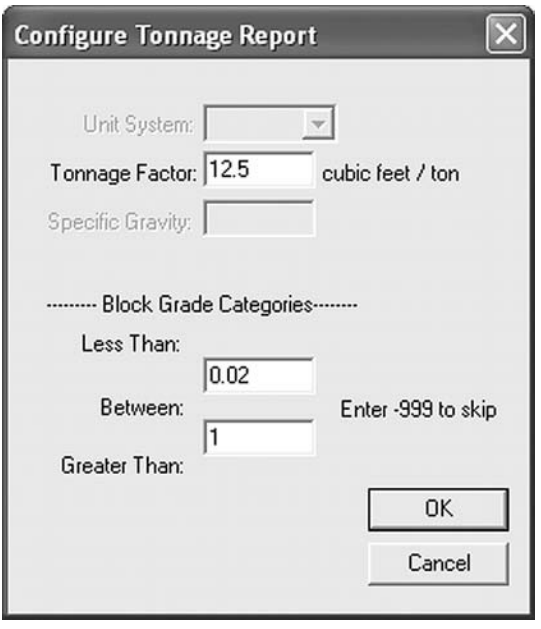
The 'Surface Constraint' is always turned on. Other constraints may be turned on and off using the 'Pit Restrictions' dialog box that is opened using the 'Restrictions' command. The 'Restrictions' command is the same as described in Section 15.6.6. Refer to this section for a description.

15.10.2 *The Configure command*

To change the Total Tons Report display control variables:

SELECT: the 'Configure' command

The 'Configure Tonnage Report' dialog box will then be displayed.



The top section of the dialog box is used to specify whether the calculations are to be performed using the English or metric unit systems. If the English unit system is chosen, then the 'Tonnage Factor' is used to calculate the tons per block. When the Metric system is chosen, then the 'Specific Gravity' is used to calculate the metric tons per block.

The lower portion of the dialog box is used to specify up to two grade categories. A value of −999 should be entered to indicate a grade category should not be used. If no grade categories are used, then the report will only contain the block totals.

If one category is defined, the report will contain two sub sections. These are (1) a sub section for all blocks with a grade less than the grade category, and (2) a sub section with all blocks with a grade greater than or equal to the grade category.

If two categories are defined, the report will contain three sub sections. These are (1) a sub section for all blocks with a grade less than the grade category 1, (2) a sub section with all blocks with a grade greater than or equal to grade category 1 and less than grade category 2, and (3) a sub section with all blocks with a grade greater than or equal to the grade category 2.

An example report with two grade categories defined for the 'ArizCu.dhf' data set is shown below.

Total Tonnage Report
June 4, 2005
File: C:\CSMine\Data Files\CU.BLK

Tonnage factor = 12.5 cu. ft./ton
Volume per block = 500000.0 cu. ft
Tons per block = 40000.0 tons

%Cu < 0.020
Number of blocks = 32
Total tons = 1280000 tons
Average grade = 0.004 %Cu
Total Value = $76.000

%Cu > = 0.020 and < 1.000
Number of blocks = 247
Total tons = 9880000 tons
Average grade = 0.179 %Cu
Total Value = $1224.000

%Cu > = 1.000
Number of blocks = 6
Total tons = 240000 tons
Average grade = 1.473 %Cu
Total Value = $37.000

Totals
Number of blocks = 285
Total tons = 11400000 tons
Average grade = 0.186 %Cu
Total Value = $1337.000

Pit restrictions used in calculation
Surface constraints : Yes
Geometric pit limits : Yes
Outer economic bounds: No
Floating cone bounds : No

15.10.2.1 *The Configure dialog commands*

To update the Total Tons Report and close the dialog box:

SELECT: the 'OK' button

The 'Configure Tonnage Report' dialog box will be closed and the Total Tons Report redisplayed using the current control variable settings.

To close the dialog box without updating the Total Tonnage Report:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

15.10.3 *The Return command*

The 'Return' command is used to close the Total Tons Report window and return to the Block mode main data window.

15.11 SUMMARY STATISTICS

In this section, the summary statistics available in the CSMine program are explained. Two example data sets found on the distribution disk can be used while learning the various statistical features of the program. These data sets, which are described in detail below, are used in the examples throughout this chapter.

15.11.1 *The EX1.CMP data set*

The EX1.CMP data set contains a number of surface elevation measurements of the top of a geologic structure. The data were collected on roughly a 12.5 m × 12.5 m grid. The elevation measurements follow a normal distribution as will be shown later in this chapter. Figure 15.21 shows a contour map of the elevation data. The small crosses are the sample locations.

15.11.2 *The EX2.CMP data set*

The EX2.CMP data set contains data from the 'ArizCu.dhf' porphyry copper deposit located in Arizona. The data have been regularized into 50 ft composites from a series of vertical drill hole core samples. Since bench compositing was used the center coordinates of the composites fall on even 50 ft increments starting from an elevation of 4125 ft and continuing down to 3575 ft. This cuts the samples into a series of 12 horizontal slices or benches. The data values given are %copper. As is common with metals, the sample values follow a lognormal distribution. Figure 15.22 shows a contour map of the copper grades for the fourth bench (elevation 3975 ft). The small crosses show the locations of the samples.

15.11.3 *Summary statistics description*

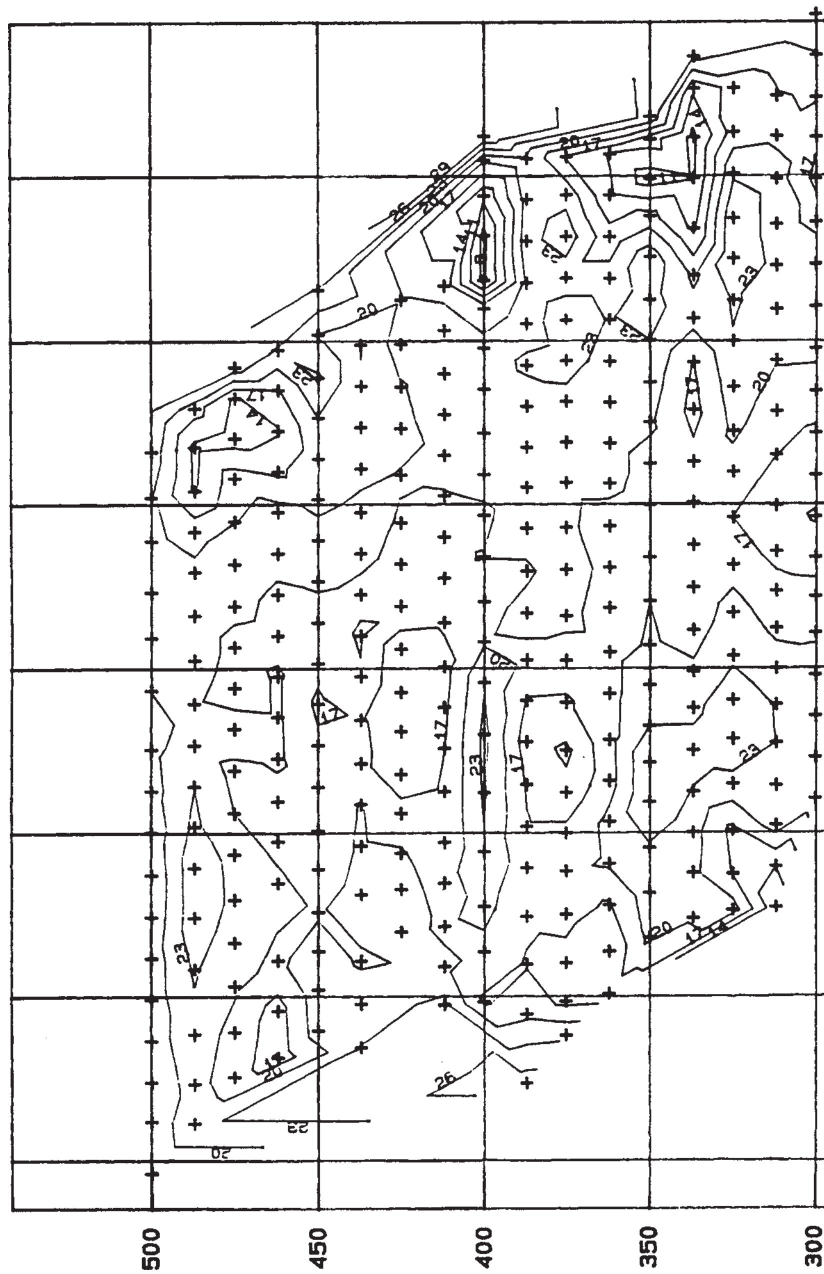
Figure 15.23 shows the results of the summary statistics screen from the data file EX1.CMP. The upper right part of the screen shows a histogram of the data. The lower part lists the table of values used in the histogram. The upper left part lists some summary statistics.

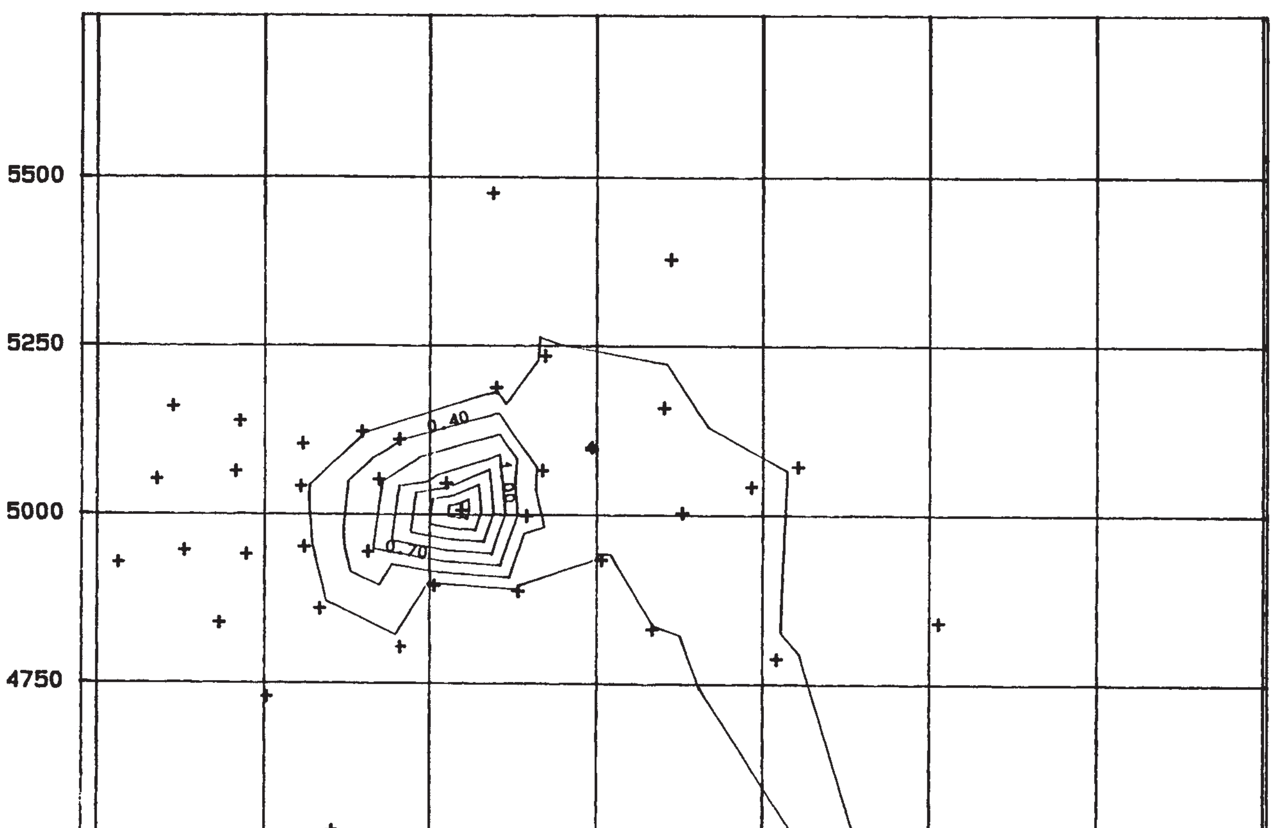
15.11.3.1 *Histogram calculation table*

The histogram is calculated by dividing the data into 10 cells of equal size. The cell size is found by dividing the maximum data value minus the minimum data value by 10:

$$\text{Cell size} = \frac{\text{Data maximum} - \text{Data minimum}}{10}$$

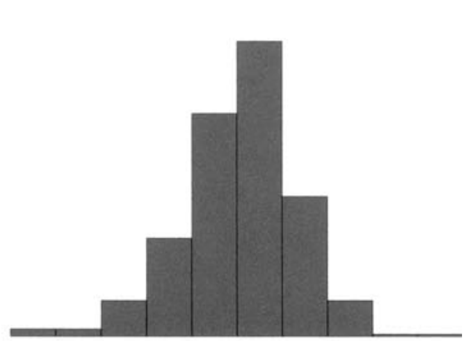
The data interval represented by each cell can then be calculated. These are listed in the column labeled 'interval'. The number of points that fall within each interval is calculated and listed in the column labeled 'frequency'. The relative frequency for each cell, labeled


500
450
400
350
300


5500
5250
5000
4750
0.40
0.70

File Name: C:\CSMine\Data_Files\EX1.CMP June 4, 2005
Variable : Height

Number of Samples	: 328
Arithmetic Average	: 19.6602
Variance	: 9.9672
Standard Deviation	: 3.1571
Coefficient of Skewness	: -0.5217
Coefficient of Kurtosis	: 1.8375
Geometric Mean	: 19.3717
Median	: 19.9691
10% Trim Mean	: 19.7437
Midrange	: 19.4759
Mean Absolute Deviation	: 2.3817



Cell	Interval	Frequency(f)	Relative f	Cummulative f
1	7.05 .. 9.54	3	0.91	0.91
2	9.54 .. 12.02	3	0.91	1.83
3	12.02 .. 14.51	14	4.27	6.10
4	14.51 .. 16.99	38	11.59	17.68
5	16.99 .. 19.48	86	26.22	43.90
6	19.48 .. 21.96	114	34.76	78.66
7	21.96 .. 24.45	54	16.46	95.12
8	24.45 .. 26.93	14	4.27	99.39
9	26.93 .. 29.41	1	0.30	99.70
10	29.41 .. 31.90	1	0.30	100.00

'relative f', is calculated by dividing the number of samples falling within an interval by the total number of samples:

$$\text{relative } f = \frac{\text{Number of samples in interval}}{\text{Total number of samples}}$$

The relative frequency value for each cell is plotted in the histogram drawing in the upper left window. The cumulative frequency for a cell, labeled 'cumulative f', is the number of samples found up to and including the number in the current cell divided by the total number of samples. The cumulative frequency for the last cell should of course be one hundred.

15.11.3.2 *Relative frequency histogram*

The relative frequency histogram is displayed in the upper right window. The values plotted are the relative frequencies for each cell listed in the histogram calculation table. The histogram plot allows one to visually inspect the data frequency distribution. This makes it possible to quickly determine if the distribution is normal, lognormal, uniform, or something else. A more exact means of determining if the distribution is normal or lognormal will be given later.

15.11.3.3 *Statistics*

The window in the upper left lists some important summary statistics. The top part lists some standard statistics and the bottom part lists some robust statistics. The meaning of each of these statistics as well as how they are calculated are discussed next.

Number of samples: The number of samples is simply the total number of samples that were used in the calculation of the statistics. It is denoted by the symbol n.

Arithmetic average: The arithmetic average or mean of the samples is simply the average value of all the samples. The mean is usually denoted by the symbol μ and can be found by summing the sample values and dividing by the total number of samples:

$$\mu = \frac{1}{n}\sum_{i=1}^{n} x_i \tag{15.14}$$

where x_i is the value of sample i and n is the number of samples.

Variance: The variance is a measure of the spread of the data. The variance is often denoted by the symbol s^2 and can be calculated by the formula

$$s^2 = \frac{1}{n}\sum_{i=1}^{n} (x_i - \mu)^2 \tag{15.15}$$

It is common to divide by $n - 1$ instead of just n when calculating the variance

$$s^2 = \frac{1}{n-1}\sum_{i=1}^{n} (x_i - \mu)^2 \tag{15.16}$$

For large values of n, the difference in the two formulas is negligible. CSMine uses formula (15.15) for calculating the variance.

Standard deviation: The standard deviation is simply the square root of the variance and is usually denoted by the symbol s. For the normal distribution, 68% of the values lie within

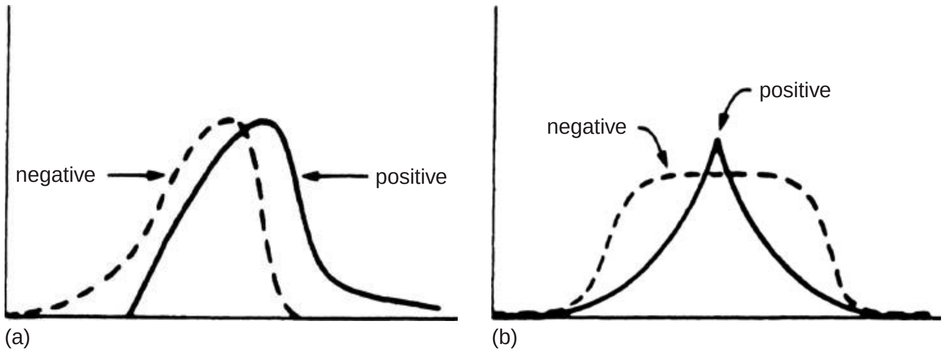


Figure 15.24. Diagrammatic representation of frequency curves illustrating: (a) skewness, (b) kurtosis.

one standard deviation of the mean and 95% of the values lie within two standard deviations of the mean.

Coefficient of skewness: The coefficient of skewness indicates if a distribution is symmetric (zero), tails to the left (negative), or tails to the right (positive), (Figure 15.24). The formula is

$$\text{Skew} = \frac{1}{n}\sum_{i=1}^{n}\left(\frac{x_i - \mu}{s}\right)^3 \tag{15.17}$$

The skewness of a distribution will almost never be exactly zero even if it is symmetric. A rule of thumb is that the skewness is insignificant unless it is many times the value of $\sqrt{6/n}$.

Coefficient of kurtosis: The coefficient of kurtosis is a dimensionless quantity which measures the flatness or peakedness of a distribution relative to a normal distribution (see Fig. 15.24). It is calculated by

$$\text{K}urt = \frac{1}{n}\sum_{i=1}^{n}\left(\frac{x_i - \mu}{s}\right)^4 - 3 \tag{15.18}$$

The -3 term makes the value zero for a normal distribution.

Geometric mean: The geometric mean is the inverse log of the average of the logs of the samples. It is calculated by

$$Gm = \exp\left(\frac{1}{n}\sum_{i=1}^{n}\ln x_i\right) \tag{15.19}$$

Median: The median of a distribution is the value for which larger and smaller values are equally probable. It is estimated by finding the value that has an equal number of samples above it and below it. Since this is not possible when n is even, the average of the two central values is then used.

$$Med = \begin{cases} x_{(n+1)/2}, & n \text{ odd} \\ 1/2(x_{n/2} + x_{(n/2)+1}), & n \text{ even} \end{cases} \tag{15.20}$$

The median is a more robust estimator of the central tendency of a mildly skewed distribution than the mean.

10% trim mean: The 10% trim mean is the mean calculated by leaving out the lower 5% and upper 5% of the samples. This value can give an indication of how sensitive the mean is to outliers.

Midrange: The midrange is the average of the largest and smallest samples:

$$\text{Midrange} = \frac{1}{2}(x_1 + x_n) \tag{15.21}$$

Mean absolute deviation: A more robust estimator of the width (standard deviation) of a distribution, is the average deviation or mean absolute deviation defined by:

$$\text{Adev} = \frac{1}{n}\sum_{i=1}^{n} |x_i - \mu| \tag{15.22}$$

15.11.4 *Is a distribution normal?*

The easiest way to determine if a distribution is normal is to plot the cumulative frequencies from a cumulative frequency histogram on probability paper. Probability paper is gridded and labeled such that when the cumulative frequencies are plotted for each cell, they will plot roughly as a straight line if the distribution is normal. Figure 15.25 shows the plot of the cumulative frequency table results from Figure 15.23 using the EX1.CMP data file. As can be seen from the graph, the cumulative frequencies plot roughly as a straight line. Thus it can be concluded that this distribution is normal.

15.11.5 *Is a distribution lognormal?*

The lognormal distribution is quite important in mineral deposit assessment due to the fact that nearly all metal quantity-grade relationships follow the lognormal distribution. Figure 15.26 shows the summary statistics from the data file EX2.CMP.

Compare this with the results from data set EX1 shown in Figure 15.23. Clearly data set EX2 is highly positively skewed. Since there are very few high valued samples in the tail of the distribution, it would seem likely that these values should be removed. Figure 15.27 shows the summary statistics from the EX2 data set with all the samples with grades above 0.25% removed.

Notice that the histogram is still highly positively skewed. This is a typical behavior of a lognormal distribution. Cutting away the tail of the distribution still leaves a lognormal distribution.

To illustrate the difference in the way a lognormal distribution appears when the cumulative frequencies are plotted on probability paper, a plot of the data from Figure 15.26 is shown in Figure 15.28. Notice that the cumulative frequencies do not form a straight line.

The lognormal distribution gets its name due to the fact that the natural logarithms of the sample values form a normal distribution. Figure 15.29 shows the summary statistics of the natural logarithms of the EX2.CMP data set and Figure 15.30 shows a plot on probability paper of the cumulative frequencies from Figure 15.29.

Notice that the histogram is not truly bell shaped and that the frequencies do not plot as a straight line. This indicates that the logs of the samples do not quite follow a normal

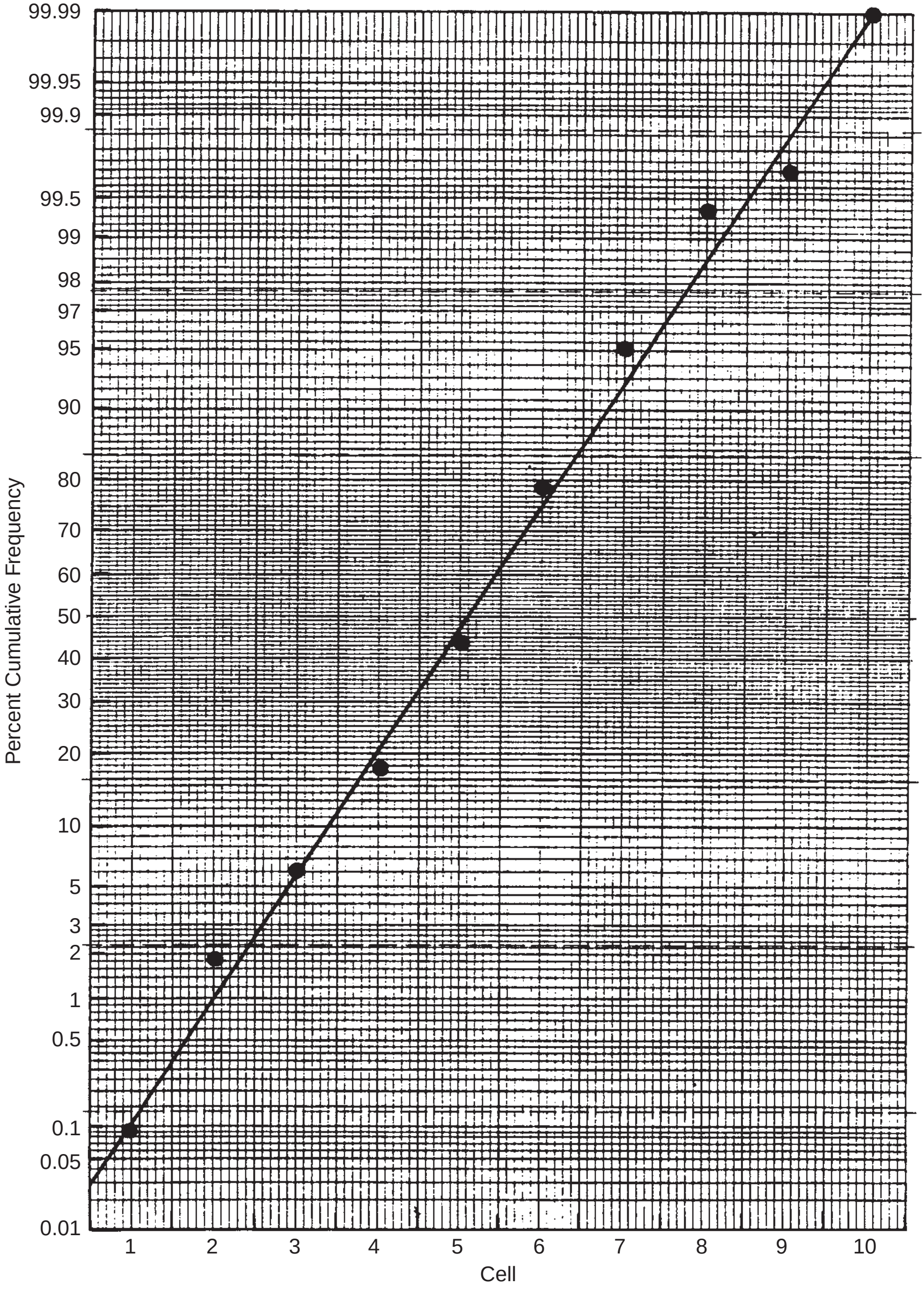
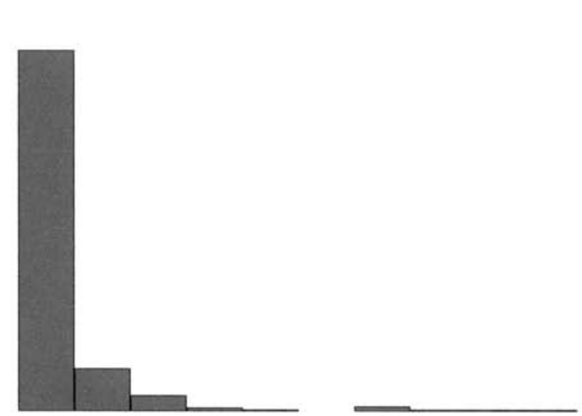


Figure 15.25. Cumulative frequency results for the EX1.CMP data set as taken from Figure 15.23 plotted on probability paper.

File Name: C:\CSMine\Data_Files\EX2.CMP June 4, 2005
Variable : %Cu

Number of Samples	:	243
Arithmetic Average	:	0.1624
Variance	:	0.1037
Standard Deviation	:	0.3221
Coefficient of Skewness	:	4.3094
Coefficient of Kurtosis	:	21.1888
Geometric Mean	:	0.0630
Median	:	0.0550
10% Trim Mean	:	0.1057
Midrange	:	1.2152
Mean Absolute Deviation	:	0.1732

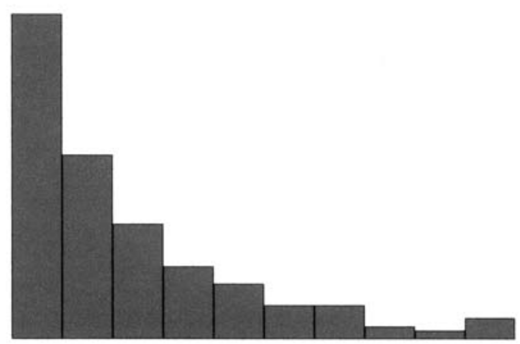


Cell	Interval	Frequency(f)	Relative f	Cummulative f
1	0.00 .. 0.25	201	82.72	82.72
2	0.25 .. 0.49	24	9.88	92.59
3	0.49 .. 0.73	9	3.70	96.30
4	0.73 .. 0.97	2	0.82	97.12
5	0.97 .. 1.22	1	0.41	97.53
6	1.22 .. 1.46	0	0.00	97.53
7	1.46 .. 1.70	3	1.23	98.77

```
File Name: C:\CSMine\Data_Files\EX2_trimmed.CMP   June 4, 2005
Variable : %Cu

Number of Samples          : 202
Arithmetic Average         :  0.0611
Variance                   :  0.0030
Standard Deviation         :  0.0551
Coefficient of Skewness    :  1.4402
Coefficient of Kurtosis    :  1.5774

Geometric Mean             :  0.0411
Median                     :  0.0400
10% Trim Mean              :  0.0554
Midrange                   :  0.1272
Mean Absolute Deviation    :  0.0431
```



Cell	Interval	Frequency(f)	Relative f	Cummulative f
1	0.00 .. 0.03	76	37.62	37.62
2	0.03 .. 0.05	43	21.29	58.91
3	0.05 .. 0.08	27	13.37	72.28
4	0.08 .. 0.10	17	8.42	80.69
5	0.10 .. 0.13	13	6.44	87.13
6	0.13 .. 0.15	8	3.96	91.09
7	0.15 .. 0.18	8	3.96	95.05
8	0.18 .. 0.20	3	1.49	96.53

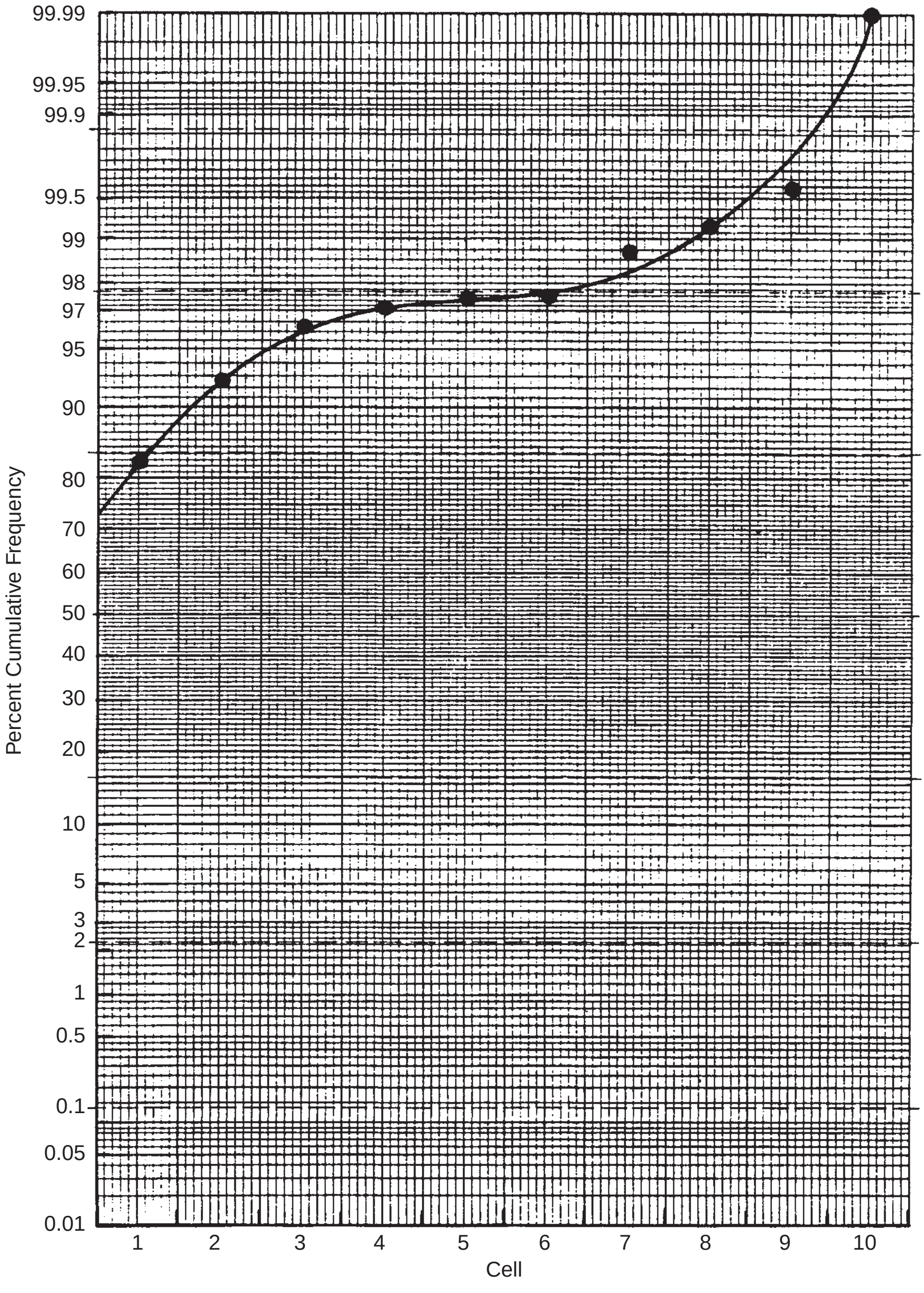
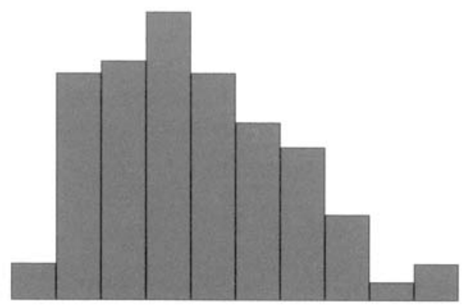


Figure 15.28. Cumulative frequency results for the EX2.CMP data set as taken from Figure 15.26 plotted on probability paper.

File Name: C:\CSMine\Data_Files\EX2.CMP June 4, 2005
Variable : Ln(%Cu), beta = 0.0000

Number of Samples	:	243
Arithmetic Average	:	-2.7642
Variance	:	1.6687
Standard Deviation	:	1.2918
Coefficient of Skewness	:	0.5113
Coefficient of Kurtosis	:	-0.2373
Geometric Mean	:	0.0000
Median	:	-2.9004
10% Trim Mean	:	-2.8146
Midrange	:	-2.2062
Mean Absolute Deviation	:	1.0563



Cell	Interval	Frequency(f)	Relative f	Cummulative f
1	-5.30 .. -4.68	6	2.47	2.47
2	-4.68 .. -4.06	37	15.23	17.70
3	-4.06 .. -3.44	39	16.05	33.74
4	-3.44 .. -2.82	47	19.34	53.09
5	-2.82 .. -2.21	37	15.23	68.31
6	-2.21 .. -1.59	29	11.93	80.25
7	-1.59 .. -0.97	25	10.29	90.53
8	-0.97 .. -0.35	14	5.76	96.30
9	-0.35 .. 0.27	3	1.23	97.53
10	0.27 .. 0.89	6	2.47	100.00

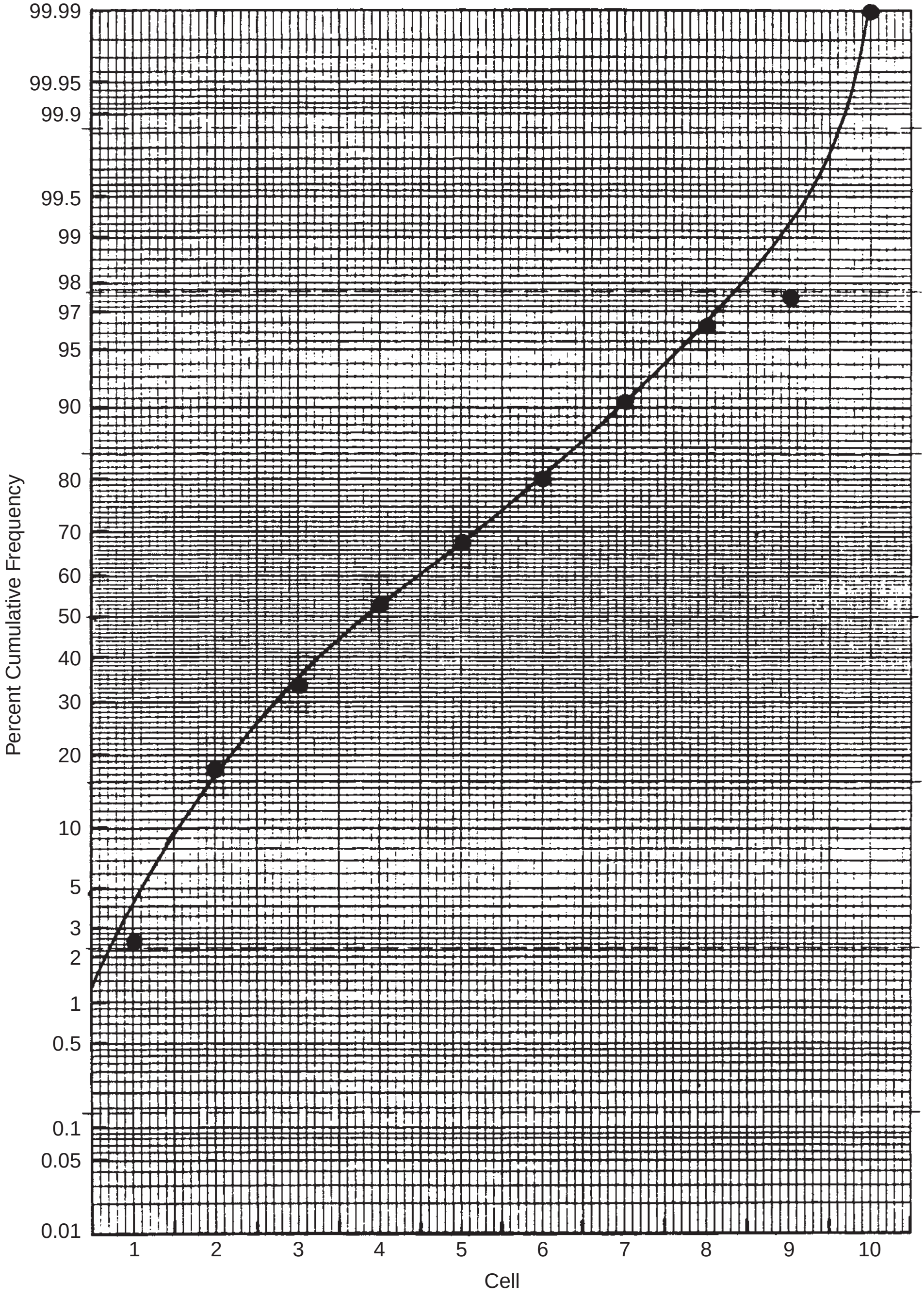


Figure 15.30. Cumulative frequency results for the natural logs of the EX2.CMP data set as taken from Figure 15.29 plotted on probability paper.

distribution. Sometimes adding a constant to the sample values before taking the logs will result in a normal distribution. This is the well-known three-parameter lognormal model:

$$\ln(x_i + \beta)$$

where x_i is the value of sample i and β is the additive constant.

The parameter β can be found by the following formula:

$$\beta = \frac{m^2 - f_1 f_2}{f_1 + f_2 - 2m} \tag{15.23}$$

where f_1 is the inverse natural logarithm of the sample value corresponding to a cumulative frequency of p, f_2 is the inverse natural logarithm of the sample value corresponding to a cumulative frequency of $1 - p$, and m is the inverse natural logarithm of the sample value corresponding to the 50% cumulative frequency.

The value f_1 often corresponds to a frequency of 15% and f_2 corresponds therefore to a frequency of 85%. Values for p between 5 to 20% are recommended.

From Figure 15.30 the values for f_1, f_2, and m are:

$$f_1 = \exp(-4.37) = 0.013$$

$$f_2 = \exp(-1.59) = 0.204$$

$$m = \exp(-3.25) = 0.039$$

The value for β can then be calculated as

$$\beta = \frac{0.039^2 - 0.13 * 0.204}{0.013 + 0.204 - 2 * 0.039} = -0.00813$$

The summary statistics from the three parameter natural logarithm transform of data set EX2 with $\beta = -0.0081$ are shown in Figure 15.31 and the probability plot for this distribution is shown in Figure 15.32. It can be seen that the cumulative frequencies plot nearly as a straight line. Note that the lowest three sample values were removed to improve the appearance of the histogram.

15.11.6 *The Transform command*

The ‘Transform’ command is used to calculate the log transforms for the composite data. The parameter β as calculated from Equation 15.23 and explained in Section 15.11.5 can be entered for each assay value in the composite data set. The log transforms can also be switched on and off. When log transforms are switched on, a new data column will be displayed with the calculated log transform for each data record. The log transforms if they exist will automatically be used with the ‘Statistics’ command as explained in Section 15.11.7, and with the ‘Variogram’ command as explained in Section 15.12.4.

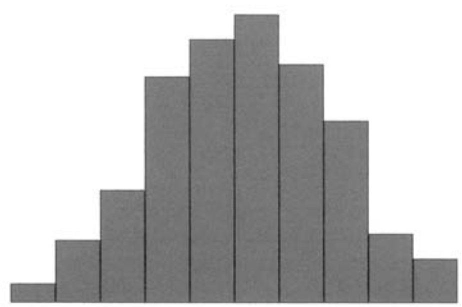
The program must be switched to the Composite mode. Refer to Section 15.1.3 for a description of how to do this. From the ‘Composite Mode’ menu:

SELECT: the ‘Transform’ command

The ‘Log Transforms’ dialog box will then be displayed.

File Name: C:\CSMine\Data_Files\EX2.CMP June 4, 2005
Variable : Ln(%Cu), beta = -0.0081

Number of Samples	:	240
Arithmetic Average	:	-3.0675
Variance	:	2.6363
Standard Deviation	:	1.6237
Coefficient of Skewness	:	-0.0637
Coefficient of Kurtosis	:	-0.2321
Geometric Mean	:	0.0000
Median	:	-3.0376
10% Trim Mean	:	-3.0597
Midrange	:	-3.2680
Mean Absolute Deviation	:	1.3044



Cell	Interval	Frequency(f)	Relative f	Cummulative f
1	-7.42 .. -6.59	3	1.25	1.25
2	-6.59 .. -5.76	10	4.17	5.42
3	-5.76 .. -4.93	18	7.50	12.92
4	-4.93 .. -4.10	36	15.00	27.92
5	-4.10 .. -3.27	42	17.50	45.42
6	-3.27 .. -2.44	46	19.17	64.58
7	-2.44 .. -1.61	38	15.83	80.42
8	-1.61 .. -0.78	29	12.08	92.50
9	-0.78 .. 0.05	11	4.58	97.08
10	0.05 .. 0.88	7	2.92	100.00

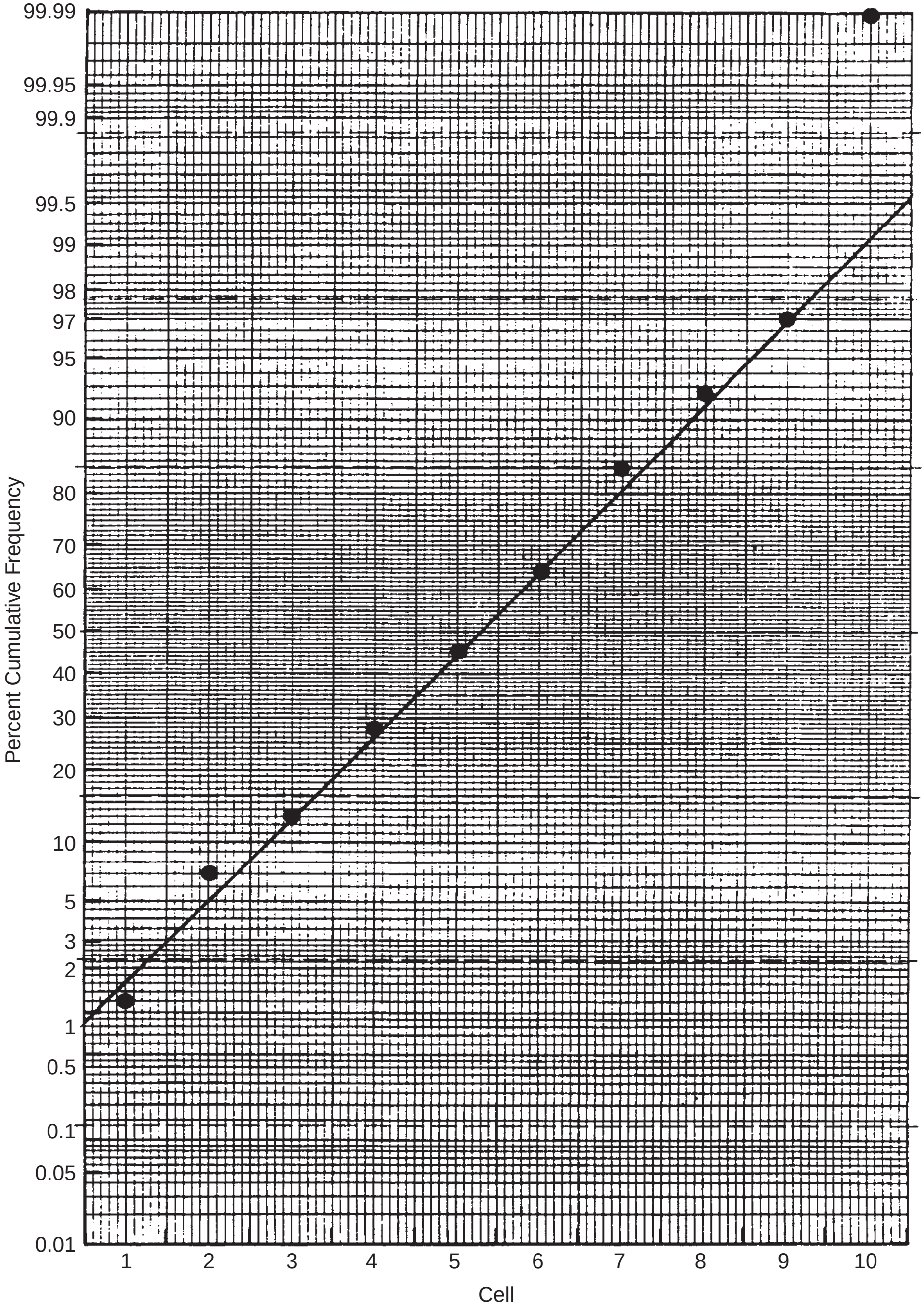
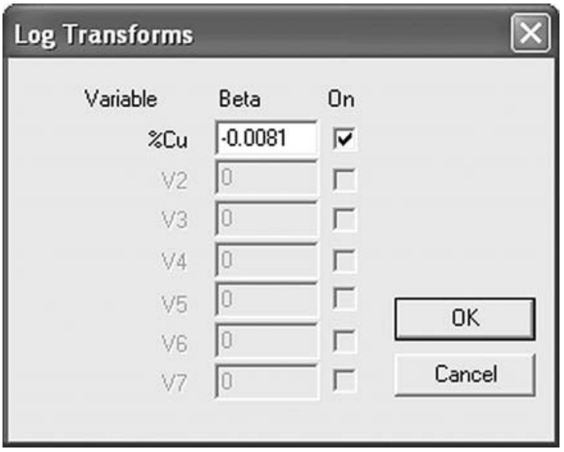


Figure 15.32. Cumulative frequency plot of the EX2.CMP data set with three parameter lognormal model results from Figure 15.31 plotted on probability paper.



Beta: The parameter β as calculated from Equation 15.23 and explained in Section 15.11.5.

On: Click in the check box to turn the calculation of the log transform for the specified variable on and off.

15.11.6.1 *The Transform dialog commands*

To update the composite data to reflect any changes made to the log transform control variables:

SELECT: the 'OK' button

The 'Log Transform' dialog box will be closed and the main data window updated to reflect any changes. If the log transform has been set to 'On' for a variable, a new data column will be displayed with the calculated log transform for each data record.

To close the dialog box without updating the data:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made to dialog box values will be discarded.

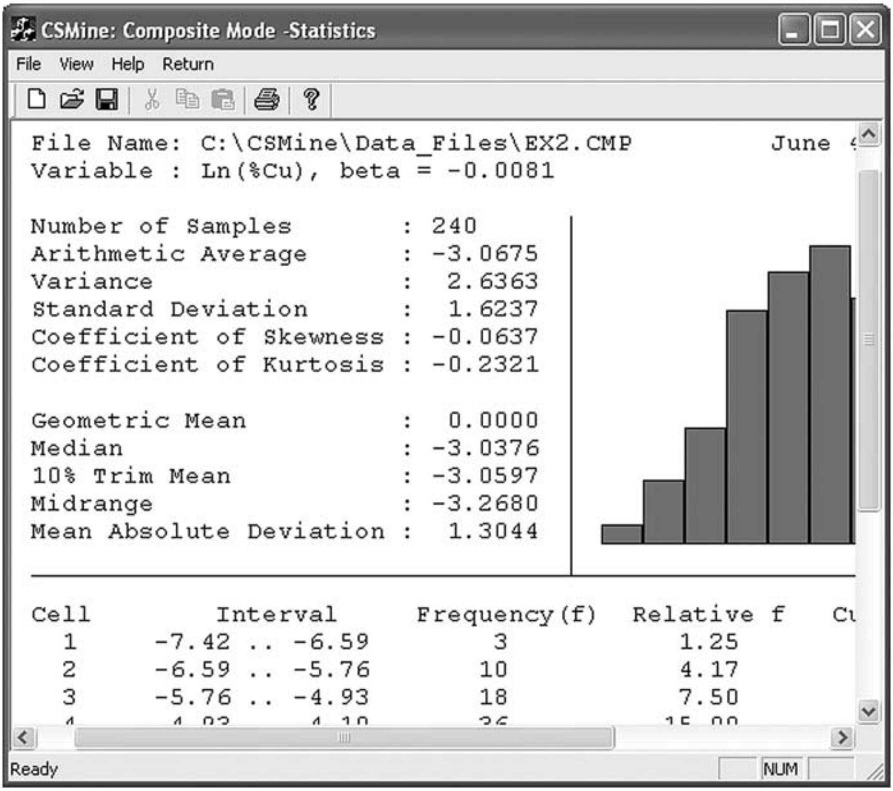
15.11.7 *The Statistics command*

The 'Statistics' command is used to display the summary statistics for the composite data. If the log transform for an assay variable has been turned on as explained in Section 15.11.6, then the log transform will be automatically used when calculating statistics.

The program must be switched to the Composite mode. Refer to Section 15.1.3 for a description of how to do this. From the 'Composite Mode' menu:

SELECT: the 'Statistics' command

The Statistics window will then be displayed.



15.11.7.1 *The Return command*

The ‘Return’ command is used to close the Statistics window and return to the Composite mode main data window.

15.12 VARIOGRAM MODELING

15.12.1 *Introduction*

The variogram is the geostatistical tool used to measure the degree of spatial correlation between samples in the region being studied. If the variable being considered is a regionalized variable, then samples taken close to one another on average will be more similar than samples taken far apart.

By definition, the value of the variance $2\gamma(h)$ for a given distance h, is the square of the expected difference between the values of the samples separated by distance h.

$$2\gamma(h) = \mathrm{E}[Z(x) - Z(x+h)]^2 \tag{15.24}$$

It is usual to drop the multiplier 2. Hence the quantity $\gamma(h)$ should be called the semivariance. However, today the ‘semi’ is generally dropped. The plots that are eventually made are called variograms rather than semi-variograms.

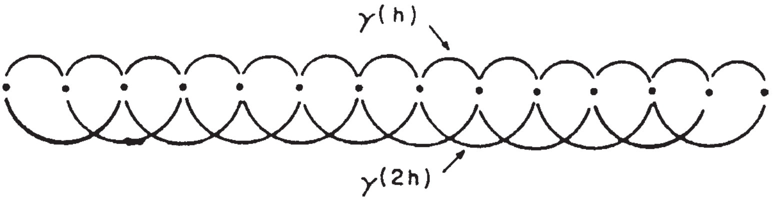


Figure 15.33. Diagrammatic representation of the one-dimensional variogram calculation, after Barnes (1980).

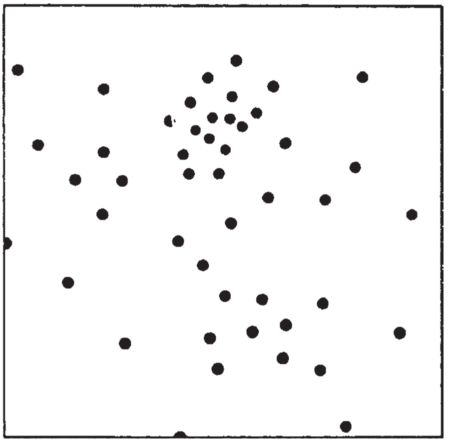


Figure 15.34. Points located randomly in a plane.

From the data at hand, the experimental variogram may be calculated by:

$$\gamma(h) = \frac{1}{n(h)} \sum_{\mathrm{i}=1}^{n(h)} [Z(x) - Z(x+h)]^2 \tag{15.25}$$

Thus for a given distance h, the values of all the samples which are separated by h are subtracted from each other, and the result squared, accumulated and divided by the number of pairs found. This gives one point on the variogram. The distance h is increased and the process is repeated. Figure 15.33 illustrates this concept on a simple one-dimensional data set.

It is much more usual that the samples are distributed more or less randomly in the plane as shown in Figure 15.34. In this case, the variance is calculated using the procedure outlined in Figure 15.35.

The horizontal direction α is specified along with the horizontal half window size dα, and the distance increment size dh. All pairs of samples x_i, x_j such that the vector from x_i to x_j falls within the interval $(\alpha - \mathrm{d}\alpha)$ to $(\alpha + \mathrm{d}\alpha)$ are treated as if they were in the same direction. All pairs of samples falling within a distance $(h - \mathrm{d}h)$ to $(h + \mathrm{d}h)$ of each other are then accumulated as with the one-dimensional case. The final distance h for each distance class is then adjusted to the average separation distance for the n pairs found within each class. A horizontal half window of 90° will effectively cover the entire plane, and is usually called the average variance.

Three-dimensional variance calculation can be carried out by simply extending the two-dimensional concept as shown in Figure 15.36.

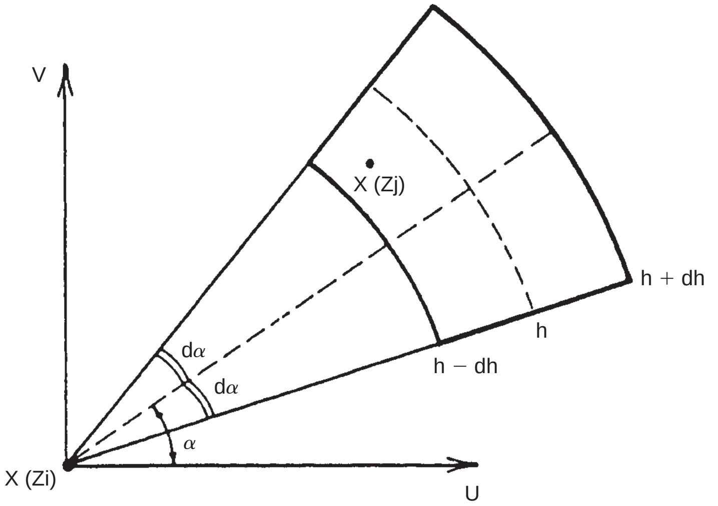


Figure 15.35. Variance calculation for 2-D random points, after Rendu (1981).

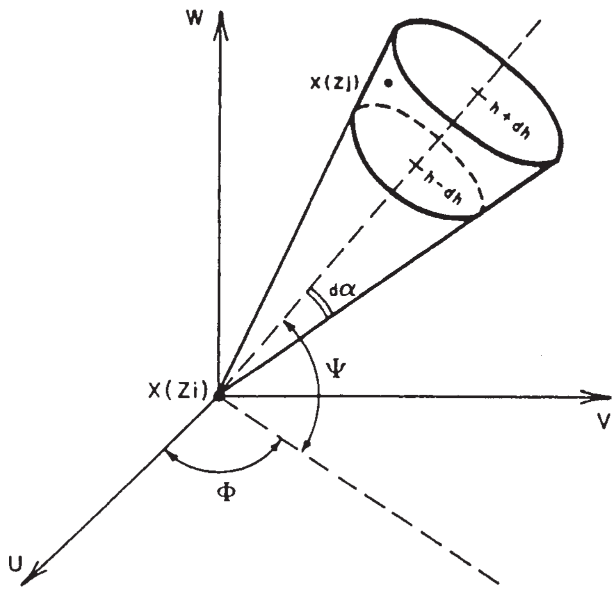


Figure 15.36. Three-dimensional variance calculation, after Rendu (1981).

With three-dimensional orebodies, the samples are usually first grouped to form composites of equal length. In open-pit mining, it is common to calculate the composites such that their length is equal to the bench height as shown in Figure 15.37. In this case it is much quicker to calculate the variances for each bench as is done in the two-dimensional case and then average them. This method is the one used in the CSMine program.

The $\gamma(h)$ values are then plotted as a function of h to form a variogram. The next step is to fit an equation to the experimental variogram. The most common variogram model is called the spherical model shown in Figure 15.38.

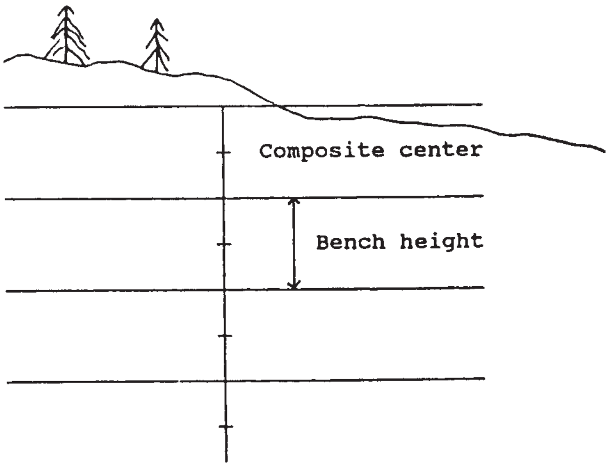


Figure 15.37. Diagrammatic representation of bench compositing.

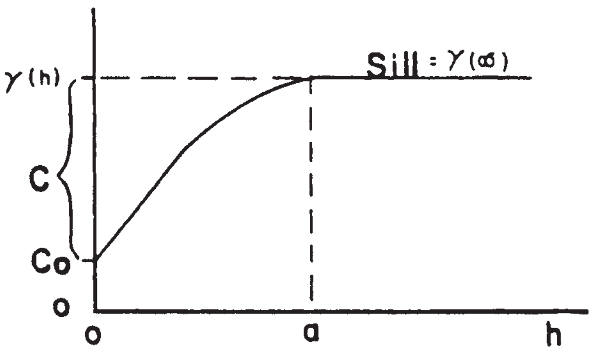


Figure 15.38. Diagrammatic representation of spherical variogram, after Barnes (1980).

The equation for the spherical variogram is:

$$\gamma(h) = \begin{cases} C\left(1.5\dfrac{h}{a} - 0.5\left(\dfrac{h}{a}\right)^3\right) + C_0, h < a \\ C + C_0, & h \geq a \\ C_0, & h = 0 \end{cases} \tag{15.26}$$

With the spherical model, the calculated variance between samples increases with increasing separation distances up to a distance a, called the range, where it levels off to a constant value. Samples with a separation distance less than the range are spatially correlated, and those with separation distances greater than the range are statistically independent. The point at which the variogram $\gamma(h)$ levels off is called the sill, and is equal to the overall variance of the sample population. The sill is composed of two components; C and C_0. C_0 is called the nugget effect and is a measure of the systematic or random error of the samples. In theory it should equal zero for regionalized variables, but in practice it often is not. The

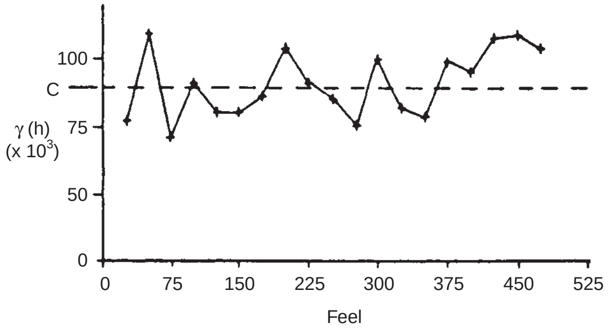


Figure 15.39. Random variogram model.

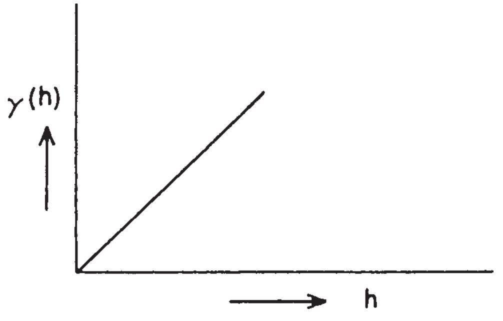


Figure 15.40. The Linear variogram model.

variance between points separated by distance h is calculated by Equation (15.26) and the covariance (σ) is found by the formula

$$\sigma = C + C_0 + \gamma(h) \tag{15.27}$$

If the nugget effect becomes large enough to be equal to the variance of the population, the variogram model is called the random model, shown in Figure 15.39. In this case there is no spatial correlation between samples, and geostatistics may not be used. The expected value of a point is then simply the arithmetic mean of the population and the estimation variance is the population variance.

The linear model shown in Figure 15.40, is another model commonly encountered in practice. The equation for the linear variogram is:

$$\gamma(h) = C_0 + Ch \tag{15.28}$$

In this case C is the slope of the line and C_0 is again the nugget effect. Basically the linear model may be thought of as a spherical model for which the range has not yet been found. A linear model is therefore valid only up to a maximum distance ($d_{\max}$). This distance is usually taken as the long axis of the geometric coordinates of the sample point distribution.

The variance between points separated by distance h is calculated by Equation (15.28) and the covariance is found by the formula

$$\sigma = C(d_{\max}) + C_0 - \gamma(h) = C(d_{\max} - h) \tag{15.29}$$

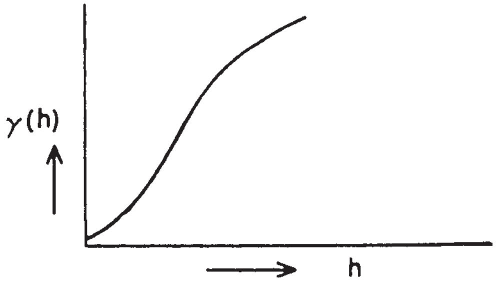


Figure 15.41. The Gaussian variogram model.

Figure 15.42. The exponential variogram model.

Another common model is the Gaussian model

$$\gamma(h) = C\left(1 - \exp\left(\frac{-h^2}{a^2}\right)\right) \tag{15.30}$$

as shown in Figure 15.41. The tangent of the curve near the origin is horizontal indicating that there is little difference between samples at short separation distances.

The exponential variogram model shown in Figure 15.42.

$$\gamma(h) = C_0 + C\left(1 - \exp\left(-\frac{|h|}{a}\right)\right) \tag{15.31}$$

has an infinite range and is therefore used to model mineral deposits that vary in a highly uniform manner.

The parabolic variogram model

$$\gamma(h) = ah^{\lambda} \tag{15.32}$$

shown in Figure 15.43 is often found with elevation data and indicates the presence of a strong trend.

15.12.2 *Experimental variogram modeling*

In this section, the fitting of some variogram models to the data sets provided with the CSMine program is illustrated. The variogram for the EX1.CMP data set can be calculated using the following parameters:

Horizontal direction (α)	: 0°
Horizontal half window (dα)	: 90°
Cell size	: 13 m
Number of cells	: 15

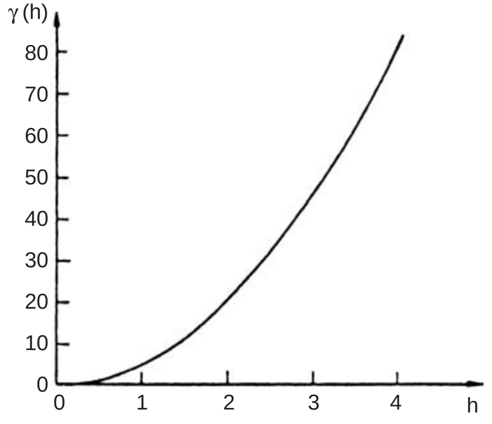


Figure 15.43. The parabolic variogram model.

Refer to Figure 15.35, which shows how a two-dimensional variance may be calculated. The cell size h is calculated by dividing the maximum separation distance by the number of cells. Figure 15.44 shows that the experimental variogram for the EX1.CMP data set is nearly an ideal spherical variogram. Equally important is Table 15.1 showing the results of the variogram calculation.

In this case it is relatively easy to fit a spherical model to the experimental data. The spherical model curve fits the calculated points nicely. At a distance of about 150 m the experimental variogram begins to deviate from the spherical model. This is typical behavior and poses no real practical problems. The reason for this is that when kriging, the closest 10 or so samples would be used to estimate the value of an unknown point. Since the average spacing for this data set is 12.5 m, points at a distance greater than about 50 m from the point being estimated would rarely if ever be used.

The EX2.CMP data set is not as easy to model. Figure 15.45 shows a plot of an experimental variogram and Table 15.2 lists the calculation results. A spherical model has been fitted to the data, but this choice could easily be debated. It is very likely that the random model should be chosen to describe these data, in which case classical statistical methods rather than kriging should be used to estimate the values at unknown points.

The experimental variogram in Figure 15.45 was calculated using the following values:

Horizontal direction (α)	: 0°
Horizontal half window ($d\alpha$)	: 90°
Cell size	: 33.33 ft
Number of cells	: 18
Bench height	: 50 ft

In Section 15.11.5 it is shown that the EX2.CMP data are well described by a three parameter lognormal distribution with a β value that was found to be -0.00813. The variogram of the log transforms shown in Figure 15.46 and the results of the calculations are listed in Table 15.3. Figure 15.46 shows clearly why lognormal geostatistics should be used on highly skewed lognormally distributed data. While the variogram is not textbook perfect, it clearly shows a high degree of spatial continuity between samples over a far greater range than the variogram made using the non-transformed data.

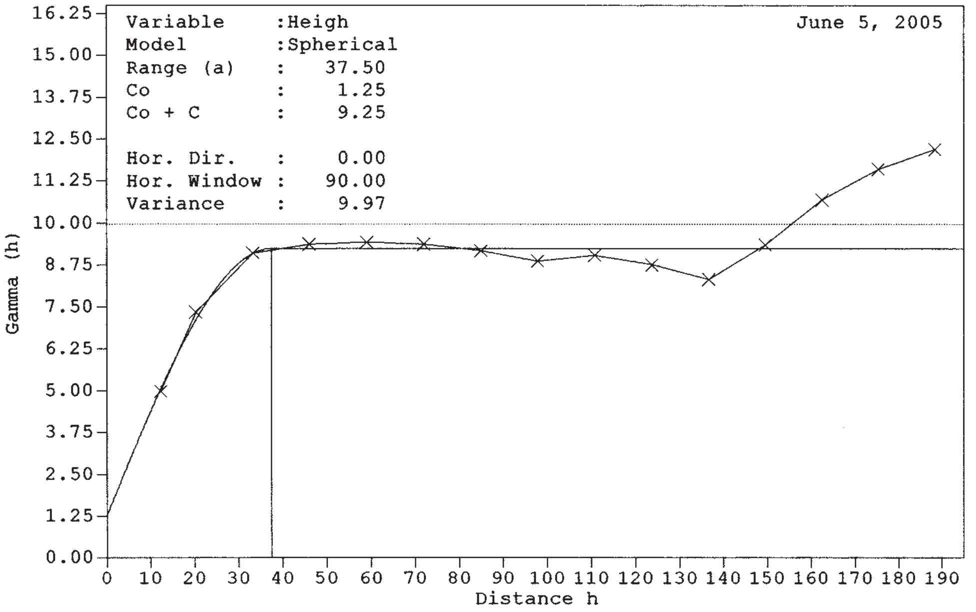


Figure 15.44. Experimental variogram from the EX1.CMP data set.

Table 15.1. Experimental variogram results for Figure 15.44.

Interval (m)	Number of Pairs	Average Distance (m)	$\gamma(h)$ (m^2)
0–13	312	12.3432	4.9848
13–26	1524	20.3445	7.3547
26–39	2130	33.3181	9.1180
39–52	2831	46.1182	9.3761
52–65	3284	59.0908	9.4368
65–78	3682	72.0803	9.3816
78–91	3795	84.9674	9.1864
91–104	4043	97.8362	8.8725
104–117	4049	110.8046	9.0487
117–130	3993	123.8049	8.7642
130–143	3806	136.7141	8.3251
143–156	3587	149.6336	9.3669
156–169	3249	162.5283	10.6979
169–182	2918	175.3840	11.6034
182–195	2481	188.3028	12.2020

The experimental variogram in Figure 15.46 was calculated using the following values:

Horizontal direction (α) : 0°
Horizontal half window (dα) : 90°
Cell size : 27.77 ft
Number of cells : 18
Bench height : 50 ft

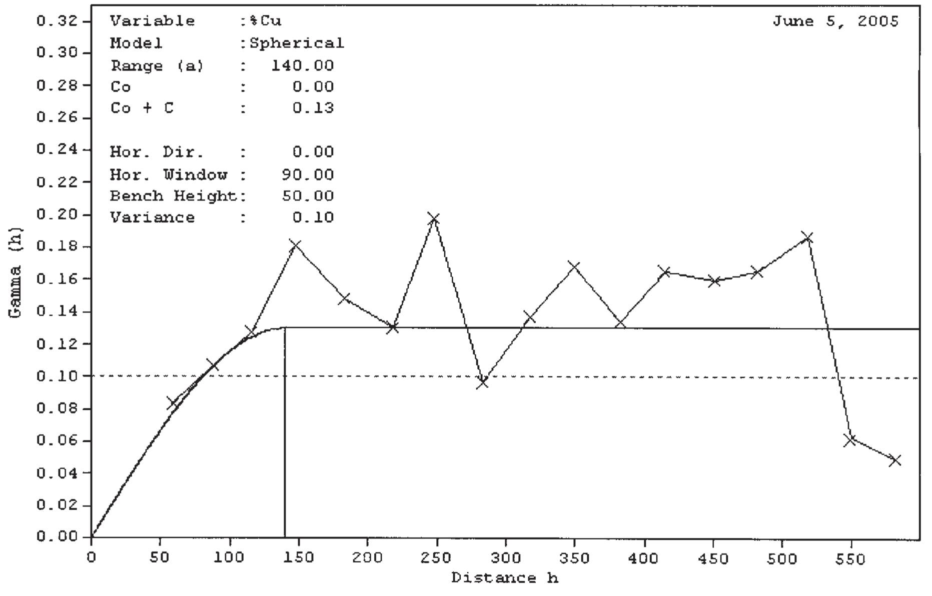


Figure 15.45. Experimental variogram from the EX2.CMP data set using non-transformed values.

Table 15.2. Experimental variogram results for Figure 15.45.

Interval (ft)	Number of Pairs	Average Distance (ft)	$\gamma(h)$ ((%Cu)^2)
0–33	0	0.0000	0.0000
33–67	21	59.1214	0.0839
67–100	95	88.6523	0.1071
100–133	167	116.3872	0.1276
133–167	152	147.5843	0.1811
167–200	186	183.5876	0.1482
200–233	205	218.4827	0.1303
233–267	230	247.9248	0.1977
267–300	165	283.4211	0.0969
300–333	214	317.5676	0.1367
333–367	191	350.1292	0.1682
367–400	122	382.9236	0.1338
400–433	180	415.4561	0.1654
433–467	153	451.0725	0.1599
467–500	124	482.5904	0.1647
500–533	121	518.7781	0.1869
533–567	114	549.9813	0.0625
567–600	140	582.1724	0.0498

15.12.3 *Anisotropy*

It is common for a mineral deposit to exhibit a greater degree of spatial continuity in one direction than another. For example, a large vein deposit will likely show greater grade continuity along the long axis or strike of the vein than across the vein. This phenomenon is

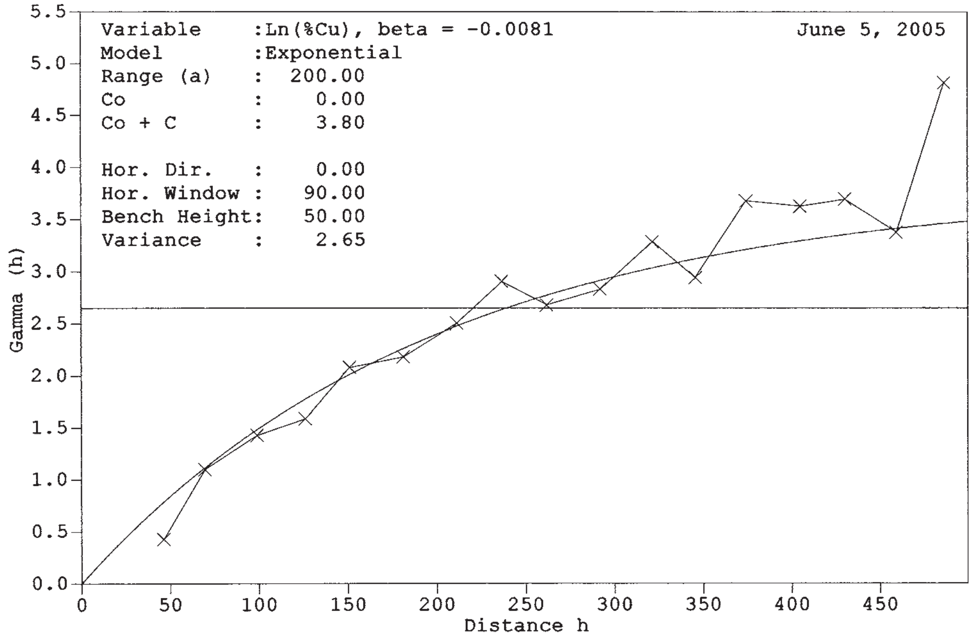


Figure 15.46. Experimental variogram from the EX2.CMP data set using log-transformed values, $\beta = -0.00813$.

Table 15.3. Experimental variogram results for Figure 15.46.

Interval (ft)	Number of Pairs	Average Distance (ft)	$\gamma(h)$ ($\ln(\%Cu)^2$)
0–28	0	0.0000	0.0000
28–56	5	46.5296	0.4323
56–83	40	69.7514	1.1009
83–111	133	99.0170	1.4288
111–139	135	126.2399	1.5898
139–167	117	151.0229	2.0844
167–194	160	181.3071	2.1841
194–222	153	211.5556	2.5058
222–250	201	236.7484	2.9081
250–278	143	262.1205	2.6801
278–305	149	292.0799	2.8334
305–333	165	321.3900	3.2907
333–361	140	345.7399	2.9455
361–389	133	374.1866	3.6802
389–417	127	405.0089	3.6303
417–444	122	430.2263	3.6988
444–472	128	459.3388	3.3815
472–500	91	486.4958	4.8191

called anisotropy. The anisotropy can easily be found by calculating variograms in different directions. The CSMine program is set up to calculate from one to six variograms at a time. The program can be set to automatically calculate variograms in six directions which completely cover the area of interest by setting the control variables to:

Number of variograms	: 6
Horizontal direction of first variogram	: 0°
Horizontal direction increment size	: 30°
Horizontal half window size	: 12.5°

This would calculate variograms at 0, 30, 60, 90, 120, and 150 degrees with each one having a horizontal window of 25 degrees. A horizontal half window size of 90 degrees will cover the entire plane and is often called the average variogram. Reducing the half window to 12.5 degrees insures that the samples in one direction are not used in calculating the variances in another direction. Note that this will reduce the total number of pairs for each point on the variogram. If the data set does not contain a large number of points, it may not be possible to determine conclusively if there is any anisotropy. Neither of the data sets included with the CSMine program showed clear signs of anisotropy. However Figure 15.47 taken from Knudsen & Kim (1978) shows how the variograms of a deposit with anisotropy might appear. All of the variograms should have the same nugget effect and sill. Only the ranges should be different in the different directions.

With the CSMine program, each variogram can be viewed and the model parameters set. A rose diagram may then be displayed and plotted as shown in Figure 15.48. The anisotropy, if it exists, can easily be seen from such a diagram.

15.12.4 *The Variogram command*

The 'Variogram' command is used to calculate and display variograms for the composite data. Composite data must be present in memory in order to calculate a variogram. See Section 15.3 for a description on how to calculate composites or read an existing composite file. If the log transform for an assay variable has been turned on as explained in Section 15.11.6, then the log transform will be automatically used when calculating variograms. Variograms that were previously calculated and saved may be read and modeled without the need for the composite data to be present in memory.

The program must be switched to the Composite mode. Refer to Section 15.1.3 for a description of how to do this. From the 'Composite Mode' menu:

SELECT: the 'Variogram' command

The Variogram window will then be displayed.

The variogram window will initially be blank if no variograms are currently stored in memory.

15.12.4.1 *The calculate Variogram command*

In order to calculate variograms, a composite file must be present in memory. Composites may be calculated from drill hole data or read from disk as explained in Section 15.4.3. Next

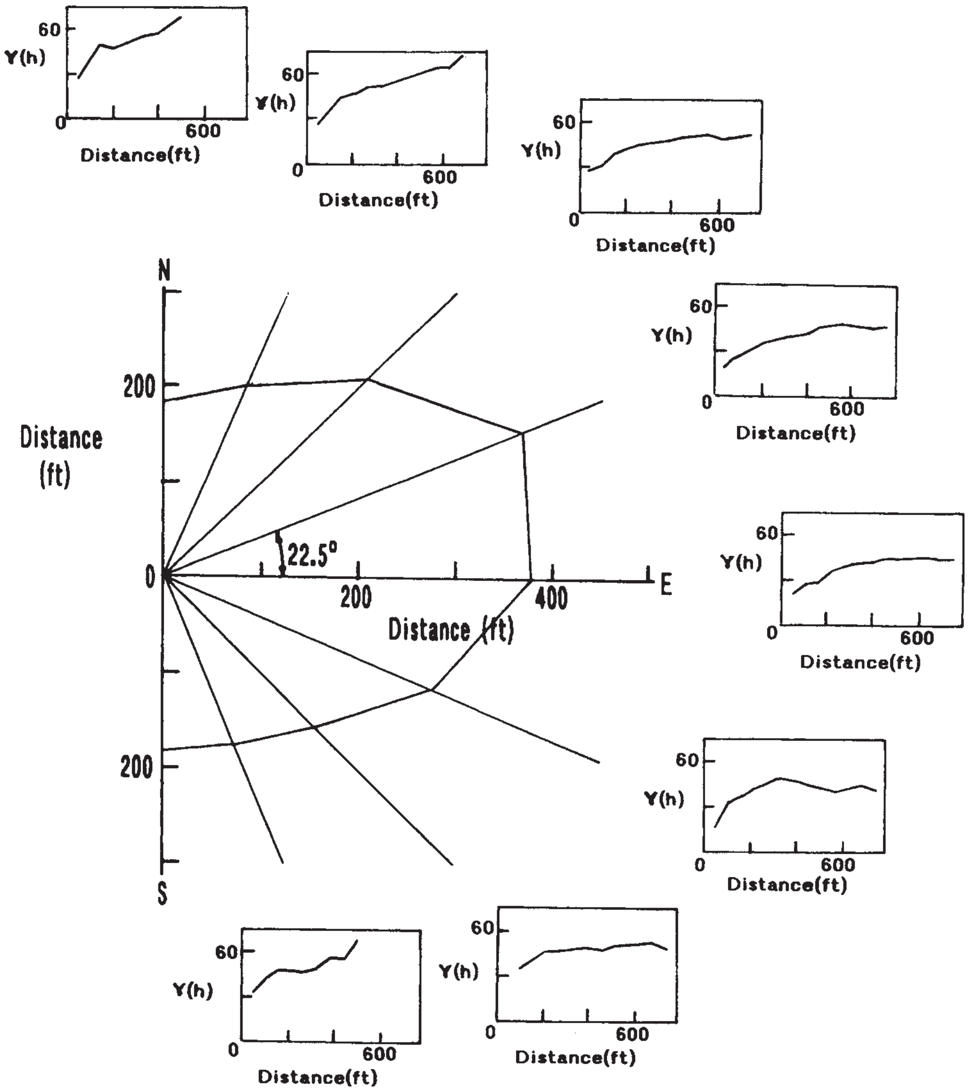


Figure 15.47. Variograms indicating signs of anisotropy, after Knudsen & Kim (1978).

the 'Variogram' command must be selected from the 'Composite Mode' menu as explained in Section 15.12.4.

From the Variogram main data window:

SELECT: the 'Calculate' command

The 'Calculate Variogram' dialog box will then be displayed.

Number of variograms to calculate: A number between 1 and 6 indicating the number of variograms to calculate.

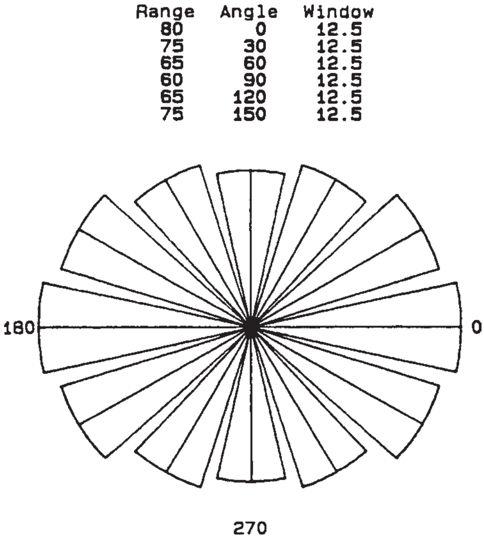
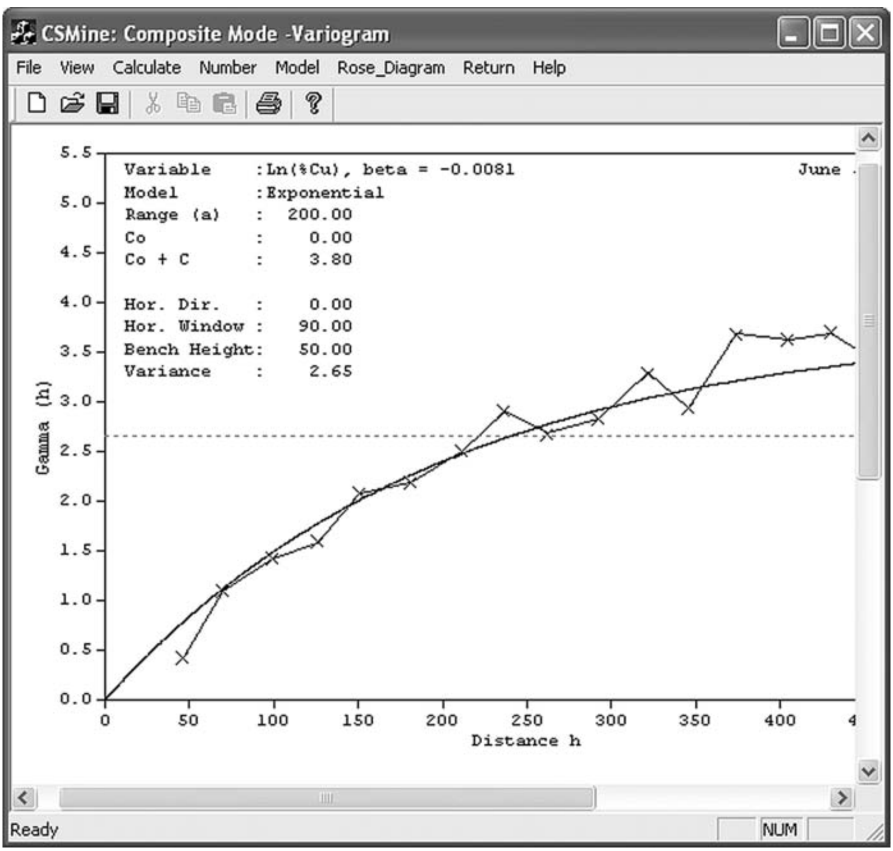
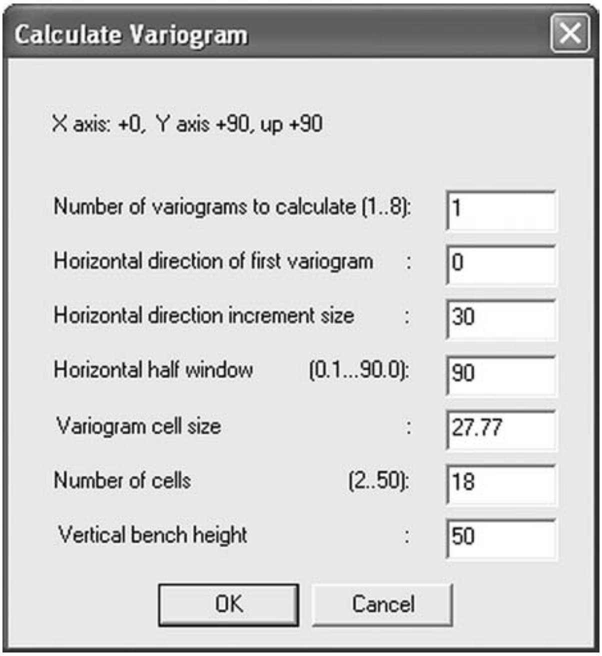


Figure 15.48. An example of a rose diagram showing a moderate degree of anisotropy.





Horizontal direction of first variogram: The horizontal direction α of the first variogram in degrees measured counter clockwise from the horizontal axis.

Horizontal direction increment size: The size in degrees for the horizontal direction angle increment size. This value is ignored if only one variogram is to be calculated (*Number of variograms to calculate* = *1*).

Horizontal half window: The size in degrees from, 0.1° to 90°, of the horizontal half window dα. If one variogram is to be calculated, (*Number of variograms to calculate* = *1*), a horizontal half window of 90° could be used to give a total horizontal window size of 180°, resulting in coverage of the entire plane. The resulting variogram is called the average variogram. In order to determine if an anisotropy exists as discussed in Section 15.12.3, the following settings can be used:

Number of variograms	: 6
Horizontal direction of first variogram	: 0°
Horizontal direction increment size	: 30°
Horizontal half window size	: 12.5°

This would calculate variograms at 0, 30, 60, 90, 120, and 150 degrees with each one having a horizontal window of 25 degrees.

Variogram cell size: The total distance in data units, usually meters or feet, for the horizontal interval dh = (*Variogram cell size*)/2. All pairs of samples falling within a distance (h − dh) to (h + dh) of each other are then accumulated for each cell.

Number of cells: The number of cells to use for each distance interval. This value must be in the range of 2 to 50.

Vertical bench height: With three-dimensional data sets, the vertical bench height defines the thickness of the slice used to group the three-dimensional composites into two-dimensional slices. This value is ignored with one- or two-dimensional data sets.

To calculate the variograms specified by the 'Calculate Variogram' dialog items and close the dialog box:

SELECT: the 'OK' button

The requested variograms will be calculated and displayed in the main Variogram window. Default values for the variogram models are reset each time variograms are calculated. Refer to Section 15.12.4.3 for a discussion on how to fit a variogram model to the experimental variogram data.

To close the dialog box without applying any changes made:

SELECT: the 'Cancel' button

The dialog box will be closed and any changes made will be discarded.

15.12.4.2 *The Number command*

If more that one variogram has been calculated, the 'Number' command can be used to step the display sequentially through the variograms available. To change the display to the next variogram number:

SELECT: the 'Number' command

The next variogram number in sequential order will then be displayed. When the last variogram is currently being displayed, invoking the 'Number' command will return the display to the first variogram.

15.12.4.3 *The Model command*

The 'Model' command is used to configure one of the standard variogram models to the experimental variogram data and to set values used to control the graphical display.

From the Variogram menu:

SELECT: the 'Model' command

The 'Model Variograms' dialog box will then be displayed.

The dialog box consists of three tabs. The 'Define Model' tab is used to define the variogram model parameters. The relationship between the C_0, C, and range (a) and the various variogram models is given in Table 15.4.

Model: The drop down menu is used to select a variogram model from the following list: 'Spherical', 'Linear', 'Gaussian', 'Exponential', or 'Random'.

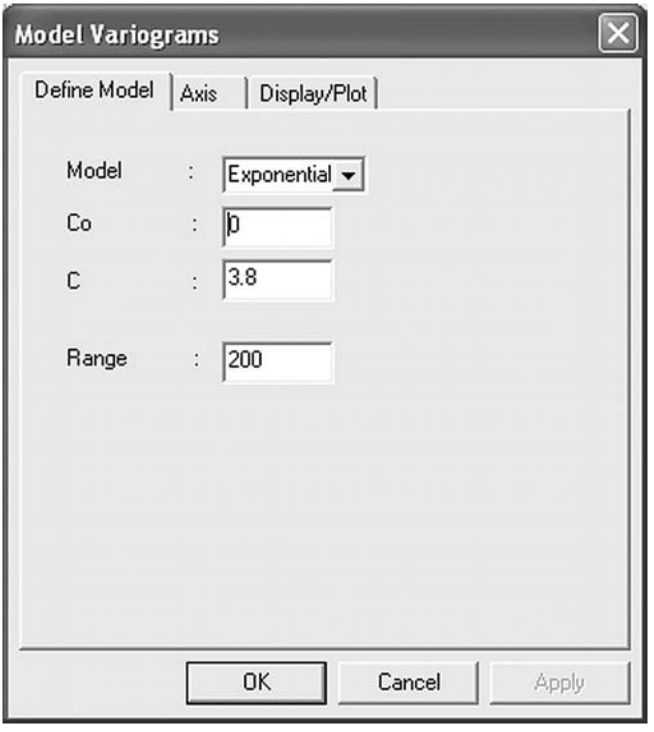
C_0: The C_0 value for the selected variogram model.
C: The C value for the selected variogram model.

Range: The range (a) value for the selected model.

The 'Axis' tab is used to define parameters that control the labeling of the axis and scale factor of the variogram plot.

Table 15.4. The relationship between the C_0, C, and range (a) and the various variogram model.

Model	Equation	Range	Nugget	Sill
Linear	$\gamma(h) = C_0 + Ch$	-	C_0	C
Spherical	$\gamma(h) = C_0 + C(1.5(h/a) - 0.5(h/a)^3)$	a	C_0	$C + C_0$
Exponential	$\gamma(h) = C_0 + C(1 - \exp(-\lvert h \rvert /a))$	a	C_0	C
Gaussian	$\gamma(h) = C(1 - \exp(-h^2/a^2))$	a	-	C



X tic distance: The distance in data units (usually meters or feet) between the labeled X-axis tics.

 # X decimals: The number of decimal places to use for the X-axis label.

Y scale: The number of data units per inch to use for the Y-axis scale factor. No scale factor is entered for the X-axis since the variogram plot is automatically adjusted to fit within the plot border in the X direction.

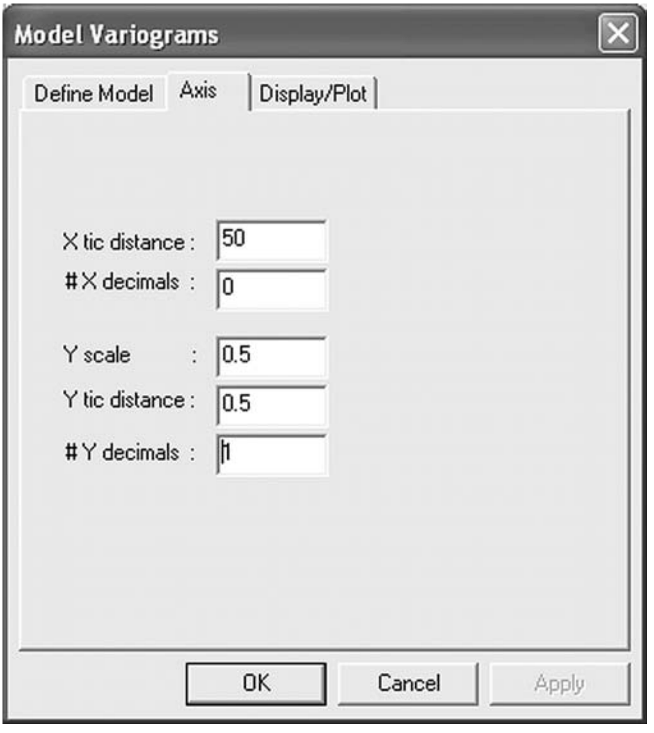
Y tic distance: The distance in data units between the labeled Y-axis tics.

 # Y decimals: The number of decimal places to use for the Y-axis label.

The 'Display/Plot tab' is used to indicate which text and graphic items should be displayed. A check next to the item name indicates that the item should be displayed.

To update the Variogram display and close the dialog box:

SELECT: the 'OK' button



The 'Model Variograms' dialog box will be closed and the Variogram window redisplayed using the current control variable settings.

To update the Variogram display without closing the dialog box:

SELECT: the 'Apply' button

The Variogram window will be redisplayed using the current control variable settings and the Model Variograms dialog box will remain open. This option will only become available if a dialog box item has been modified.

To close the dialog box without updating the Variogram window:

SELECT: the 'Cancel' button

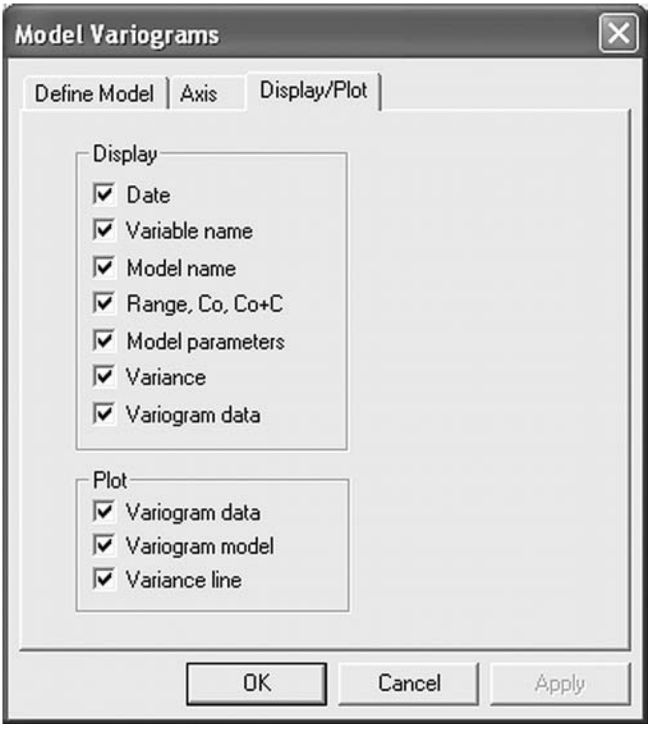
The dialog box will be closed and any changes made to dialog box values will be discarded.

15.12.4.4 *The Rose_Diagram command*

The 'Rose_Diagram' command is used to display a rose diagram such as that shown in Figure 15.48 when multiple variograms have been calculated. The Rose diagram is useful for determining the anisotropy as discussed in Section 15.12.3.

From the 'Variogram' menu:

SELECT: the 'Rose_Diagram' command



The screen will then be updated to display the Rose Diagram for the series of variograms currently stored in memory.

To exit the Rose Diagram window and return to the Variogram window:

SELECT: the 'Return' command

15.12.4.5 *Saving variograms*

The variograms currently stored in memory may be written to disk for storage. From the 'Variogram' menu:

SELECT: the 'File' command

To save the variograms to a previously defined file, from the drop down menu:

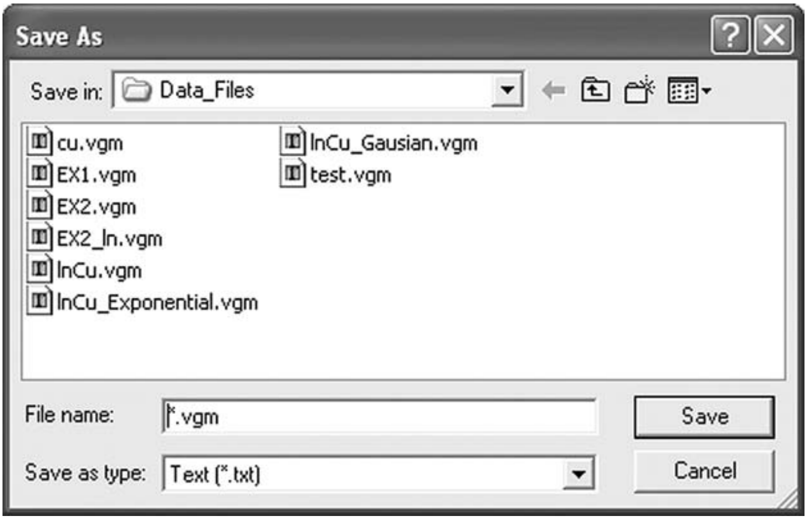
SELECT: the 'Save' command

If the variogram file name has been previously defined through the previous use of a Save or 'Save As' command, or through a previously read project file, then the variogram file will be saved to the defined file. If no variogram file name is currently defined, the Save As command will automatically be invoked.

To save the variograms to a new file, from the drop down menu:

SELECT: the 'Save As' command

The standard Windows file dialog box will appear from which the user may select an existing block file with the extension '.vgm', or enter the name of a new file to write the variograms to.



15.12.4.6 *Reading stored variograms*

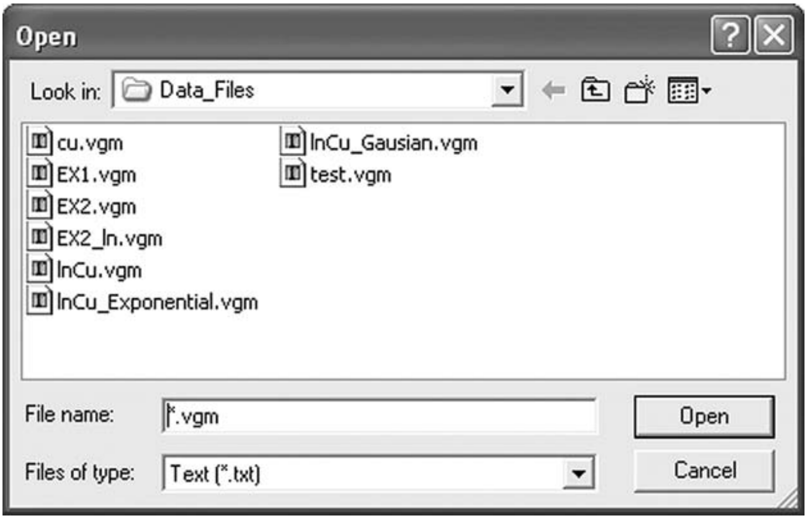
A variogram file that has been saved on disk can be read back into memory. From the 'Variogram' menu:

SELECT: the 'File' command

From the drop down menu

SELECT: the 'Open' command

The standard Windows file dialog box will then appear from which the name of an existing variogram file to read can be selected.



If there were any problems reading the file an error message will be displayed. If the file was successfully read, the first variogram stored in the file will be displayed in the data window.

15.12.4.7 *The Return command*

The 'Return' command is used to close the Variogram window and return to the Composite mode main data window.

REFERENCES

Barnes, M.P. 1980. *Computer-Assisted Mineral Appraisal and Feasibility*. American Institute of Mining, Metallurgical and Petroleum Engineers, Inc

Clark, I. 1979. *Practical Geostatistics*. Applied Sciences Publishers. London.

David, M. 1977. *Geostatistical Ore Reserve Estimation*. Elsevier, Amsterdam.

Joumel, A. & C.J. Gand Huijbregts 1978. *Mining Geostatistics*. Academic Press, London.

Knudsen, H.P. & Y.C. Kim 1978. *A Short Course on Geostatistical Ore Reserve Estimation*. University of Arizona.

Krige, D.G. 1978. *Lognormal-de Wijsian Geostatistics for Ore Evaluation*. South African Inst. Min. Metall. Monograph Series, Johannesburg.

Rendu, J.M. 1981. *An Introduction to Geostatistical Methods of Mineral Evaluation*. South African Inst. Min. Metall. Monograph Series, Johannesburg.

CHAPTER 16

The MicroMODEL V8.1 mine design software

16.1 INTRODUCTION

The MicroMODEL V8.1 program is a powerful, interactive, three-dimensional deposit modeling and mine planning system. The system has been written by a team of experts who have considerable experience in computer modeling and mine planning on a wide variety of mineral deposits. The result is a versatile and easy to use package offering three-dimensional deposit block modeling and open pit evaluation capabilities. There is a User's Manual comprised of two volumes (Volume I and Volume II) which is included with the software.

It has been found by experience, however, that the "New-User" requires a little extra help in getting started. Once properly introduced to the software, the "New-User" can easily learn the different features by reading the Manual.

Introduction to the software is via a small data set (Ariz_Cu) included as part of this book. If the "New-User" is familiar with the menu-driven open pit design software "CSMine" included in this book, then MicroMODEL should be very easy to learn. MicroMODEL is the logical next step up from CSMine. You are encouraged to review the pit planning concepts described in this book as you learn to use this new software.

You will be using a special Academic Version of the software prepared especially for teaching. It has all of the features of the regular software with the exception that it will only function when used with the six included sample projects. Also, only a single rock type is allowed. If the model limits (origin, block size, extents), number of drill holes, or number of sample intervals are changed in any way, the program will cease to function on that project. Table 16.1 lists the parameters which cannot be changed for each of the six available projects. The drill hole interval data has been pre-loaded for all of the projects, and a surface topography grid has already been created.

The Academic Version provides a "soft way" to learn this very powerful tool. As the student will find, this software is organized in the way that the planning engineer evaluates an orebody and hence is engineer-friendly. The cost of the full software package is much less than other commercial software packages intended for the same purpose. The CD contains the program MicroMODEL 8.1 and the required data files for running the six sample projects.

Table 16.1. Demo project model parameters.

Project	Origin East	Origin North	Origin Elev	Ncol	Nrow	Nlev	Column Size	Row Size	Level Size	Number DH	Number Samples
MMDemo	3500	4300	2800	80	68	66	25	25	15	58	2397
Seam Demo	0	0	5900	57	53	30	100	100	10	17	51
Azul	69300	52140	3900	115	83	70	20	20	10	489	33878
Andina	4000	4300	2500	130	170	70	10	10	10	76	2099
Ariz_Cu	1200	4200	3500	40	30	30	50	50	25	40	2017
Norte_Cu	600	0	2000	210	160	103	10	10	10	144	22869

16.2 PROGRAM OVERVIEW

16.2.1 *Introduction*

MicroMODEL is comprised of several modules which assist the user in the analysis of mineral deposits amenable to block modeling methods.

This software package enables the user to enter drill hole data, and then statistically analyze and display this data. Next, a two-dimensional model of the surface topography is built. Upon completion of the topographic model, a three-dimensional geologic model of the deposit is constructed either from the drill hole information or digitized polygons. Drill hole data are then composited, for statistical reasons. Next, the necessary grade models are built as three-dimensional grids using polygonal, Inverse Distance to a Power, (IDP) or kriging estimation methods.

Finally, the user can evaluate material within digitized polygons for contained volumes, tonnages, grades and stripping ratios. A floating cone algorithm is available to aid in open pit design.

The main modules within MicroMODEL are:

- Data Entry
- Surface Modeling
- Rock Modeling
- Drill Hole Compositing
- Grade Modeling
- Pit Generation and Reserves Evaluation
- File Management
- Grade Thickness Modeling
- Special Tools
- PolyMap Interface

The first six of these will be briefly discussed.

16.2.2 *Data Entry Module overview*

The Data Entry Module is invoked when the user needs to input, output, plot, statistically analyze, or manipulate the sample drill hole database. A sample database is generally comprised of drill hole information as it appears in the drill hole logs.

Prior to entering the drill hole data, the Project Information File must be established. This file contains information that each program will need for execution such as model orientation and dimensions, number and names of the labels, etc. It is mandatory to have run the Project Information Program before any other programs can be run.

The remaining programs in this module enable the user to enter and analyze sample drill hole data. Drill holes are allowed at any orientation with up to 1000 downhole directional surveys per drill hole. (Drill hole collar card plus 999 downhole survey cards). Each drill hole interval is specified by its interval from and to, rock type (code), and assay values. Continuous intervals and sampling are not necessary, but are highly recommended. Simply enter unsampled values for all intervals that are not sampled.

Two methods of entering drill hole data into a MicroMODEL database are available:

- Enter data in three separate data files (collar, survey, assay). These files are converted to the standard input format and then read into MicroMODEL. Geology can alternatively be entered as a fourth ASCII input file.
- Enter data in a single, standard MicroMODEL format. This method is not recommended and has been retained only so that legacy projects with the original input format are still compatible with the current version of the software.

The user can specify the record formats of the ASCII data file that is to be read.

At any point, the sample drill hole database can be output to a printer in a report style format allowing the user to verify the input data. This program can also be used to create a free format ASCII file which can be subsequently edited and reread into MicroMODEL. This ASCII file also serves as a backup of the drill hole database.

It is highly recommended that the user maintain all original data in some sort of a spreadsheet format, since almost every user has a copy of Microsoft Excel or similar program. Use the program that reads separate data files to create a standard MicroMODEL input file.

Three plotting programs are available to produce maps of the drill hole sample data. The first program plots drill hole collar locations along with several user specified lines of information. The second program plots drill hole mid-bench pierce points and corresponding data for any user specified level of the model. The third program plots drill hole cross sections at any orientation and vertical exaggeration. This program can be instructed to plot a drill hole interval value numerically, or as a histogram along the left or right side of each drill hole. Optionally, filled color bars may be plotted to depict different rock types or grade ranges. The cross-sectional topographic profile can also be plotted with this program. Each of the map plotting programs enables the user to design the format of the plot to suit the users particular needs.

As with all plot files created by MicroMODEL, the graphical display plots can be previewed on the computer screen before being sent to the printer or plotter.

By utilizing the programs in the Data Entry Module, the user can enter, display, and verify the drill hole database. Once the user is confident that a "clean" drill hole database exists, it can be used in subsequent rock modeling, compositing, and grade modeling.

16.2.3 *Surface Modeling Module overview*

The Surface Modeling Module contains a series of programs that enable the user to analyze prepared point data. Point data can be prepared from any combination of digitized contours, drill hole collars or downhole rock horizons, or XYZ data files.

The prepared point data can be used to create 2-D grid models of topography, or of thickness. Once the models have been created, they can be displayed in one of several different ways.

From input data comprised of drill hole collar locations and/or digitized data, MicroMODEL has three methods to create a surface model:

- Inverse Distance to a Power (IDP).
- Kriging.
- Triangulation (Triangulated Irregular Network – TIN).

Each of these modeling methods assigns an elevation or thickness value to the center of each cell that can be estimated. Unestimated cells can be remodeled using different sorting and modeling parameters or directly assigned elevations with the Grid Editor program (in Module 7).

Before the topographic or thickness data can be used for 2-D grid modeling, it must be prepared. This step aggregates all surface data, digitized and/or drill hole information to be used, into one file which is subsequently used for variogram analysis and modeling.

MicroMODEL's kriging or Inverse Distance to a Power (IDP) surface modeling techniques can be used. Prior to modeling the grid values with either kriging or IDP, the user can utilize the Variogram Analysis programs to geostatistically analyze the topographic data.

Before cell elevations are actually estimated, MicroMODEL must presort the data relevant to estimating each cell, according to user specified search radii, ellipse of anisotropy, and type of search (closest point or octant). This extra step before actual cell estimation saves modeling runtime because the user can try several modeling runs with different modeling parameters using the same presort file. If the search parameters are changed however, the prepared surface data must be presorted again.

The 2-D grids can be displayed by six methods within MicroMODEL. Each method allows the user to design the output to fit their own particular needs. The types of grid displays offered by MicroMODEL are:

- Single Digit Printer Map (Quick, not to scale, with each digit representing a range of values).
- Plan View Cell Plot (Scale map showing the numerical elevations of the topography grid or thickness grid – Can also be color-filled blocks).
- Contour Plot (Scale map with elevation or thickness isopleths at user specified intervals).
- Perspective or Isometric Three-Dimensional Fishnet Plot (visualization aid).
- Perspective or Isometric Three-Dimensional Raised Contour Plot (visualization aid).
- 3-D view in the 3-D Display Module.

As with all plot files created by MicroMODEL, the graphical display plots can be previewed on the computer screen before being sent to the plotter.

16.2.4 *Rock Modeling Module overview*

Once a satisfactory two-dimensional topographic model has been built, the user must build a three-dimensional model of the deposit geology (rock model). The purpose of the rock model is to enable the user to control how the drill hole data are used during grade modeling, as different rock types may need to be modeled independently. MicroMODEL offers five methods to build a rock model, which can be used in any combination:

- Block assignments from the sample or composited drill hole interval rock codes.
- Block assignments from digitized polygons in plan view.
- Block assignments from digitized polygons in cross section.

- Block assignments from a grade model.
- Block assignments from a wireframe model.

Rock modeling from drill hole data is the simplest way to develop a rock model. This method is generally used when detailed control during grade modeling is not required.

This method makes nearest neighbor assignments from the sampled or composited drill hole interval data to each block center. Ellipsoids of anisotropy can be used to further control the block assignments.

Plan view polygonal modeling assigns all rock model blocks that are below the topography and contained within a "rock volume" with a user specified rock code. The "rock volume" is defined by the digitized polygonal shape, a bottom elevation, and a top elevation.

Cross-sectional polygonal modeling also assigns a user specified rock code to all rock model blocks that are below the topography and contained within a "rock volume". In the case of cross-sectional rock modeling, the "rock volume" is defined by the cross-sectional shape extended a user specified distance, perpendicular to the section line.

Rock modeling from a grade model allows the user to specify grade cutoffs and assign a rock value to each interval. The user must create an interim rock model with dummy codes first, in order to create the grade model.

Generally, the cross-sectional polygon method is most frequently used, as geologic information is normally recorded in cross section. As stated earlier, the three methods can be used in combination. For example, it may be advantageous to use digitized polygons in cross section to define the geology, and then overlay a plan view polygon which limits the extent of the deposit mineralization to prevent the grade modeling programs from estimating grade into known barren zones.

The rock model grid can be displayed by six methods within MicroMODEL. Each method allows the user to design the output to fit his particular needs. The types of grid displays offered by MicroMODEL are:

- Single Digit Printer Map (Quick, not to scale, where each digit represents a range of rock codes for a given bench)
- Plan View Cell Plot (Scale map showing the rock codes and/or grade values of up to 10 grid values per block for a specified bench
- Contour Plot (Scale map with isopleths of the rock model for a specified bench)
- Perspective or Isometric Three-Dimensional Fishnet Plot (Visualization aid displaying higher rock codes as peaks for a specified bench)
- Cross Section Plot (Scale map showing the rock codes and/or grade values of up to 10 grid values per block in a cross-section through the center of a row or column. A cross sectional representation of the topography model is also available)
- Angled Cross Section Plot (Scale map showing the rock codes and/or grade values of up to 10 grid values per block in a cross-section running between any two points. Options are similar to choice 5.

As with all plot files created by MicroMODEL, the graphical display plots can be previewed on the computer screen before being sent to the printer or plotter.

16.2.5 *Drill Hole Compositing Module overview*

The programs from the Compositing Module are invoked when the user wants to composite the sample drill holes, output, statistically analyze, or manipulate the composited drill hole database. Most of the programs in this module are identical to those contained in the Data

Entry Module (Section 16.2.2). When these programs are invoked from the Compositing Module, they access the composited drill hole database rather than the sampled drill hole database.

Compositing the drill holes creates a composited database which is usually considerably smaller than the sampled database. MicroMODEL has three types of compositing:

- Drill Hole Compositing
- Mixed (Bench) Compositing
- Rock Type Compositing

Drill hole compositing prorates the sample drill hole interval assay values into composite intervals equal to the model bench height down the hole. This method does not take into account the dip of the drill hole or the composite interval relative to the bench location. See Volume II, Compositing, Calculation of Composite Values for more information.

Mixed compositing is an attempt to bench composite all drill hole intervals. Drill holes that cannot be bench composited due to shallow drill hole dip are composited downhole (drill hole compositing).

Bench compositing prorates the sample drill hole interval assay values into composite intervals such that the "from" elevation of the composite corresponds to the top (crest) elevation of the bench and the "to" elevation corresponds to the bottom (toe) elevation of the bench. See Volume II, Compositing, Calculation of Composite Values for more information.

Rock type compositing calculates composites using assay values only from within a consecutive string of invervals with identical rock codes. Composites are created in equal lengths that are as close as possible to a target length specified by the user. The user must also specify a minimum length for a composite. If a consecutive interval has a total length that is less than the minimum composite length, then no composite is created for that particular consecutive interval. Each composite is given a rock code that is the same as the code for the consecutive string of identical codes. See Volume II, Compositing, Calculation of Composite Values for more information.

Rock type compositing should be used in situations where definite discontinuities exist at rock boundaries. A classic example would be a large quartz vein, where the vein carries grade, but the surrounding host rock is barren.

Composite rock codes for both bench and drill hole composites can be extracted from either the rock model or the most prominent sample rock code within the composite.

Generally, the composite rock codes are extracted from the rock model since the rock model represents an interpretation of the deposit geology.

At any point after compositing, the composited database can be output to the printer in a report style format, allowing the user to verify the composited data. The drill hole database editor can be used to correct errors discovered in the composited drill hole database.

The remaining programs in the Compositing Module, including statistical and display programs, are identical to the programs described in Section 16.2.2, Data Entry Module Overview. Remember, the programs invoked from the Compositing Menu will access the composited drill hole database.

16.2.6 *Grade Modeling Module overview*

The Grade Modeling Module contains a series of programs which enable the user to geostatistically analyze drill hole assay data, model the assay data into three-dimensional grade models, manipulate the models, and display the models.

The grade modeling methods available are polygonal, Inverse Distance to a Power (IDP), and kriging. Grade modeling can be controlled by rock type to separate populations with different characteristics. Further control of the grade modeling can be achieved by using anisotropic data searches and anisotropically weighted modeling.

For each estimated cell within a grade model, the modeled grade block value applies to all material within the cell bounded above by the bench crest (or topography) and below by the bench toe. The unestimated blocks do not affect the statistical calculations, but are processed with 0.00 grade values during pit evaluation computations in the Pit Generation and Reserves Evaluation Module (See Section 16.2.7).

Prior to modeling grade values with either kriging or IDP, the user can utilize the Variogram Analysis programs to geostatistically analyze the drill hole assay data for the current label. Point Validation can be used to cross validate the selected modeling parameters by comparing the estimated values of drill hole intervals against the actual interval values. Manipulation of the grade model is also available to perform algebraic operations on the block grade values. For example, the user may want to convert from ounces/ton to milliounces/ton.

The grade model grids can be displayed by seven methods within MicroMODEL. Each method allows the user to design the output to fit their particular needs. The types of grid displays offered by MicroMODEL are:

- Single Digit Printer Map (Quick, not to scale, where each digit represents a range of values for a given bench)
- Plan View Cell Plot (Scale map showing the numerical values of up to 10 grid values per block for a specified bench)
- Contour Plot (Scale map with isopleths of any modeled grade grid for a specified bench)
- Perspective or Isometric Three-Dimensional Fishnet Plot (Visualization aid displaying higher values as peaks for a specified bench)
- Cross Section Plot (Scale map showing the numerical values of up to 10 grid values per block in a cross section through the center of a row or a column. A cross sectional representation of the topography is also available)
- Angled Cross Section Plot (Scale map showing the rock codes and/or grade values of up to 10 grid values per block in a cross-section running between any two points. Similar options to the previous choice)
- 3-D plot via the 3-D Display Menu.

As with all plot files created by MicroMODEL, the graphical display plots can be previewed on the computer screen before being sent to the printer or plotter.

The user can run basic statistics and cumulative frequency analyses upon the current label's grade model. Correlation analysis can be performed between any two grade models that have been created.

Once the necessary grade grids have been modeled and verified, the user can proceed to the Pit Generation and Reserves Evaluation Module. It is not necessary to model each label present in the drill hole sample or composite database. Only the grade models of interest for Pit Evaluation and Reserves Evaluation need to be modeled.

16.2.7 *Pit Generation and Reserves Evaluation Module overview*

Once the topography, rock, and grade models have been created, the user can calculate mineral resources and evaluate pit designs with the programs in this module. MicroMODEL allows the user to quickly analyze a variety of pit designs and sequences based upon user specified pit parameters. An efficient floating cone algorithm can be used to help design an open pit, based on a given set of economic and physical design constraints.

This module utilizes the previously modeled topography, rock and grade models. It analyzes material within the digitized polygons for (1) ore, low-grade, and waste volumes and tonnages, (2) run of mine grades, and (3) stripping ratios. The user has control of the pit slopes and the densities that will be used for the different rock zones.

The program then reports volume, tonnage, and grade for ore, low-grade, and waste as well as stripping ratios for each bench containing mineable material for the digitized increment. If the increment is profitable, the user can instruct MicroMODEL to "mine-out" the rock model which prevents the "mined" material from being re-counted later. The original topography, grade, and rock models are not permanently changed and the user can start again with an "unmined" model at any time.

The Open Pit Design (OPD) routines within MicroMODEL enable the user to evaluate ore zones incrementally. However, the floating cone algorithm included under the OPD Main Menu is an easier, automated way to design open pits.

Once the ultimate pit boundaries have been established, the pit model can be inverted, which makes all material outside the pit unavailable for mining. This enables the user to perform sequencing and scheduling within the ultimate pit without accidentally exceeding the current ultimate pit.

16.3 DATA ENTRY TUTORIAL

16.3.1 *Introduction*

This tutorial is based on the use of the data set Ariz_Cu. These are the same data used together with the program CSMine discussed in detail in Chapters 14 and 15. The data set consists of 40 drill holes, a total of 2017 assay values and a file containing XYZ grid points defining the topography. Rock types are not provided which considerably simplifies the example. The 3 input files

- Ariz_Cu_Assay.txt
- Ariz_Cu_Collar.txt
- Ariz_Cu_TopoXYZ.DAT

are included on the CD in the proper formats. The "New-User" should study these files and their formats.

16.3.2 *Some notes on input files*

The following are some important notes on Input Files:

1. You can create them in Excel but then save them as space delimited (*.TXT), tab delimited (*.TAB), or comma delimited (*.CSV).
2. The files read by MicroMODEL must be in plain text format, that is, readable by text editing programs such as Wordpad® or Notepad®.
3. Stick with the same format for all files. Although the Arizona Copper Drill hole files are provided in space delimited format, the most common format used is comma separated (*.CSV).

4. When creating your files, you can include header lines indicating the content of the files. Usually, the header line consists of column descriptions.
5. If you use DH as part of the Drill hole identification, there must not be a space between DH and the number. DH50 is okay, DH 50 is not okay. You may use an underscore character or dash between DH and the number. DH names are limited to 12 characters in length.
6. In creating your COLLAR file, the "Total Depth" is not required and is not read into the system.
7. You do not need to put in a rock code if there is only 1 rock type. The rock type in this case will default to 9999.
8. You do not need to add the end of file *EOF line.
9. You should have all the assay intervals filled. That is, there should be no gaps in the interval data. MicroMODEL will allow this, but will issue a warning. These gaps can be either left out, or MicroMODEL will optionally fill them in with the required missing (−999.99) value. If an assay is missing, use −999.99
10. It is very convenient to use the same names as MicroMODEL uses. These field names are DHNAME, EAST, NORTH, ELEV, AZM, DIP, FROM, TO. Do this!

It should be noted that many of the drill hole collar elevations have been adjusted up or down from their original values as part of CSM Data so that they more closely match the modeled surface topography.

16.3.3 *Getting started*

To Install MicroMODEL, insert the installation CD into the drive. From Explorer, navigate to the CD and find the file MicroMODEL. It is in the \MicroMODEL folder. Double click. You will find the installation program "Setup_MM80_Academic.exe." Double click on the icon.

You will see the Welcome Screen shown in Figure 16.1.

Click "Next".

The next screen is the License Agreement shown in Figure 16.2. Read the agreement as you scroll down it completely to the end. If you accept the terms of the agreement, check the box "I accept the agreement". Click "Next".

The next screen (Fig. 16.3) contains "Readme Information." It tells you there are six demo projects that are going to be installed on your computer in addition to the program itself. These projects will be installed in the folder "mmproj" located under "My Documents" if you have Windows XP or under "Libraries" if you have Windows 7.

Click "Next". The next screen (Fig. 16.4) is "Select Application Folder". If you wish to install MicroMODEL somewhere else, then change the destination folder.

Click "Next". The next screen (Fig. 16.5) controls the program group and desktop shortcut options. Make your choices and click "Next".

The next screen (Fig. 16.6) is the "Ready to Install" screen.

Click "Install". The Installing screen will now appear, indicating the installation progress. When the installation procedure is complete, the "Completing the MicroMODEL8x Setup

Figure 16.1. The "Welcome Screen".

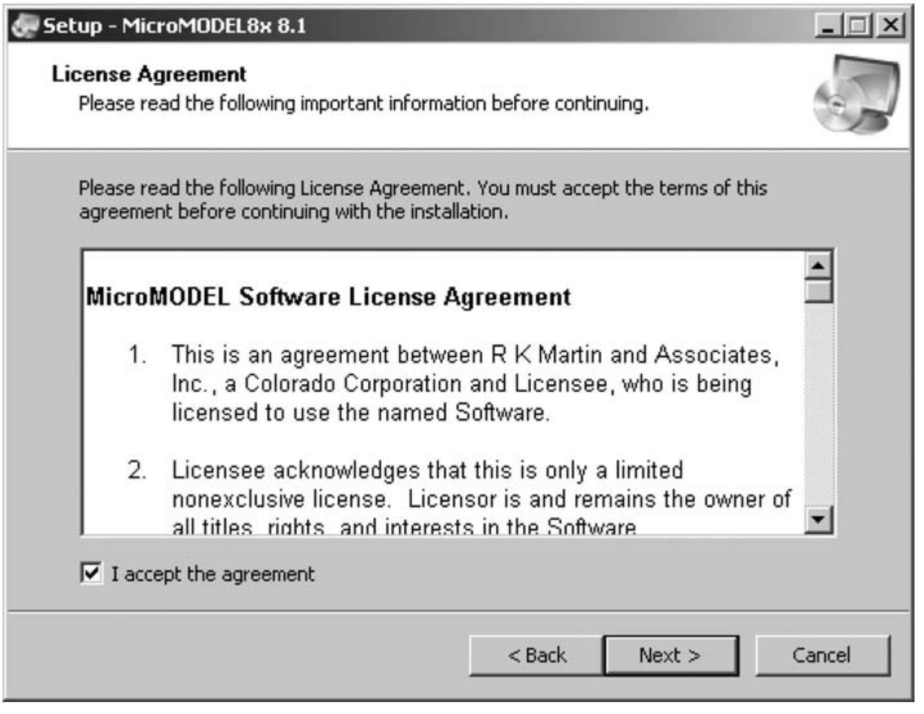


Figure 16.2. The License Agreement.

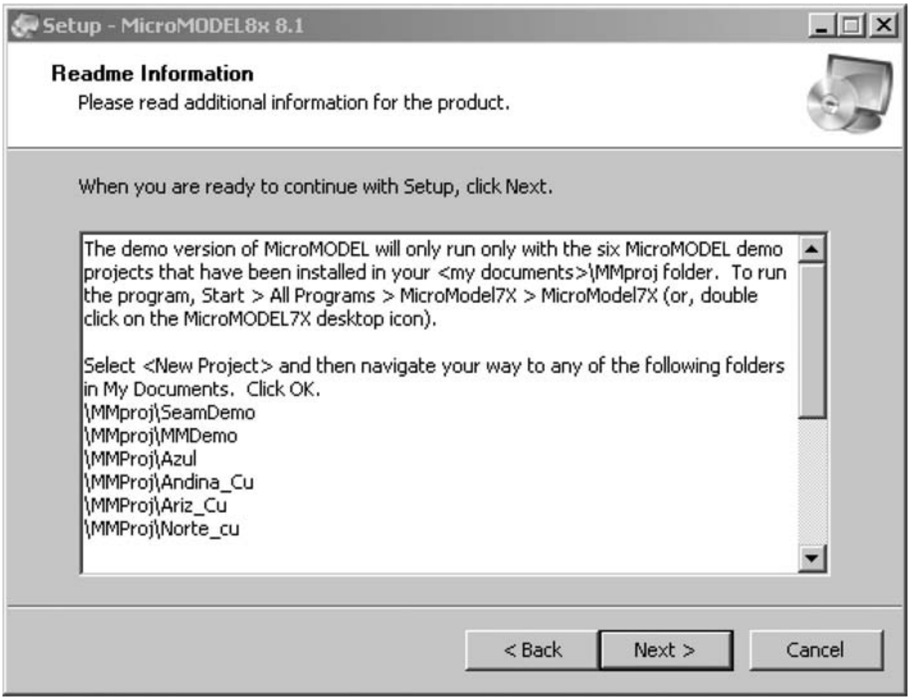


Figure 16.3. Readme Information.

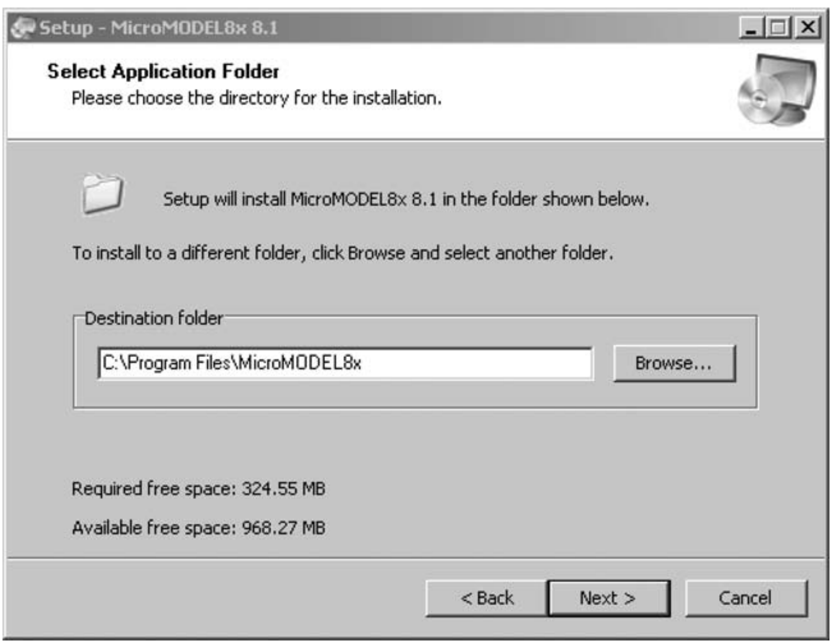


Figure 16.4. Select Application Folder.

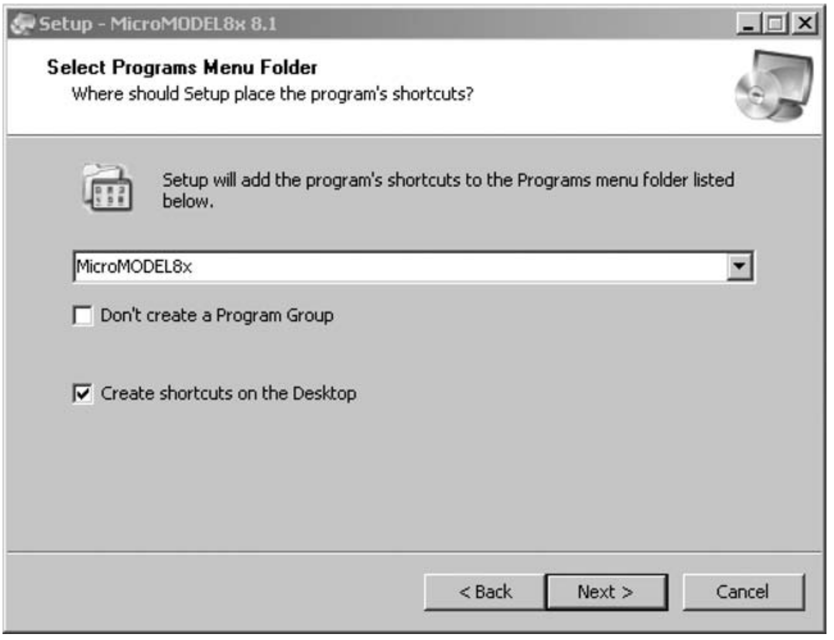


Figure 16.5. The "Select Programs Menu Folder."

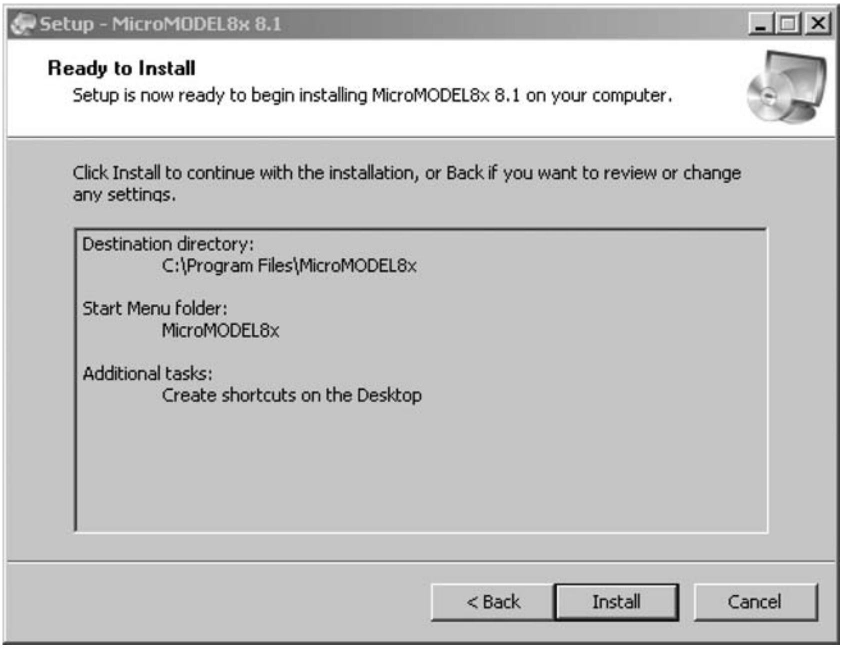


Figure 16.6. The "Ready to Install" Screen.

Figure 16.7. Setup Completed.

Wizard" screen (Fig. 16.7) will appear. If you wish to Launch MicroMODEL8x right away, leave the check box Launch MicroMODEL8x checked. Click "Finish".

16.3.4 *Starting a demo project*

To start MicroMODEL and access one of the six demo project folders, you can either double click on the MicroMODEL8x desktop icon, or start MicroMODEL from the Start menu.

From the Start Menu: Start > All Programs > MicroMODEL8x > MicroMODEL8x. When you start MicroMODEL for the first time, you will get the screen shown in Figure 16.8.

Click "OK". You will then see the "Select New Folder for MicroMODEL project" dialog screen shown in Figure 16.9.

As shown, the "MicroMODEL8X" box is illuminated. DO NOT click "OK" but

(1) If you have Windows 7 move the cursor up to "Libraries", click on this, then move to "My Documents", click on this, and then move to the folder "mmproj" and click on it.
(2) If you have Windows XP move the cursor up to "My Documents" and look for the folder "mmproj" which contains the needed files.

As you will see (Fig. 16.10), there are six demo projects located in <My Documents>\MMPROJ. In this example, we are choosing the CSM example data base for Arizona copper (ariz_cu).

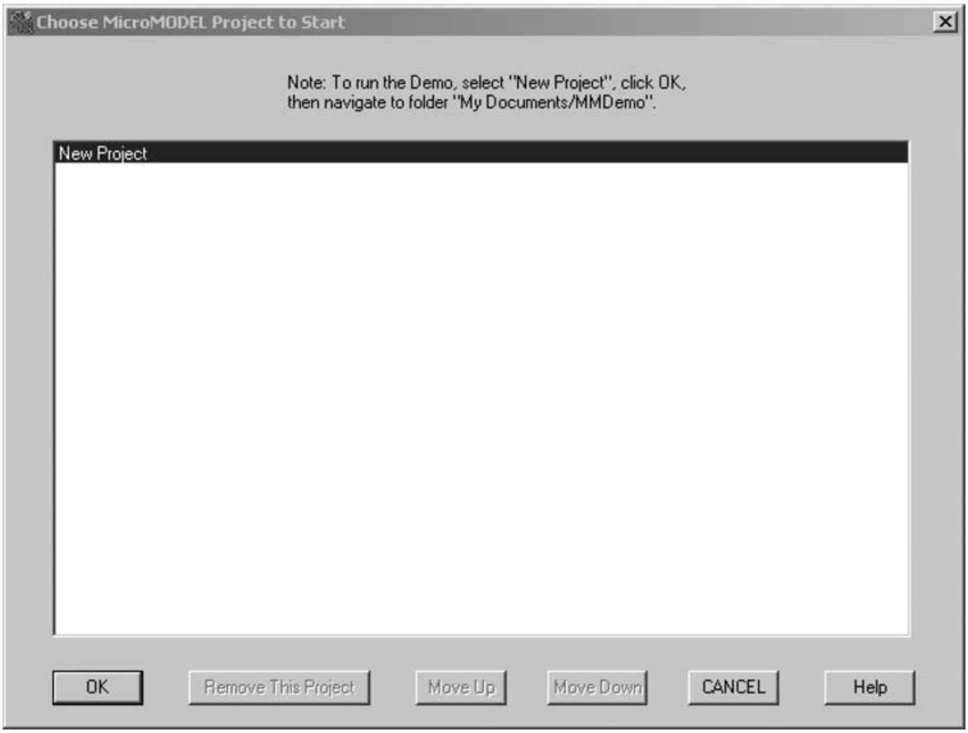


Figure 16.8. Project "Start" Screen.

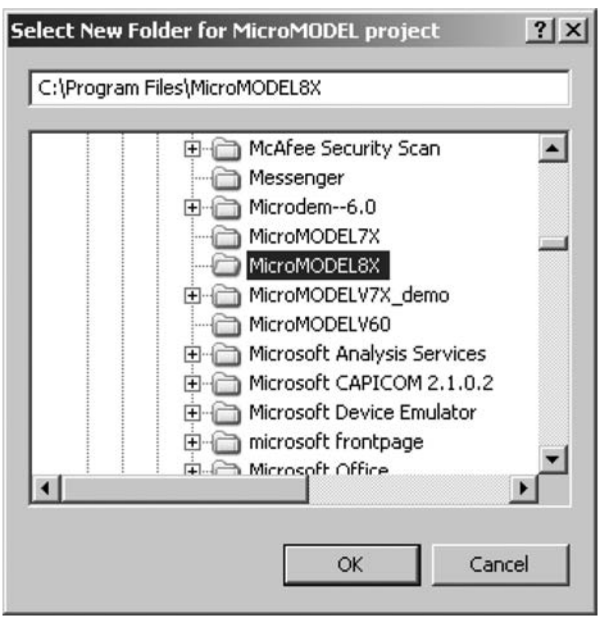


Figure 16.9. New Project Selection Screen.

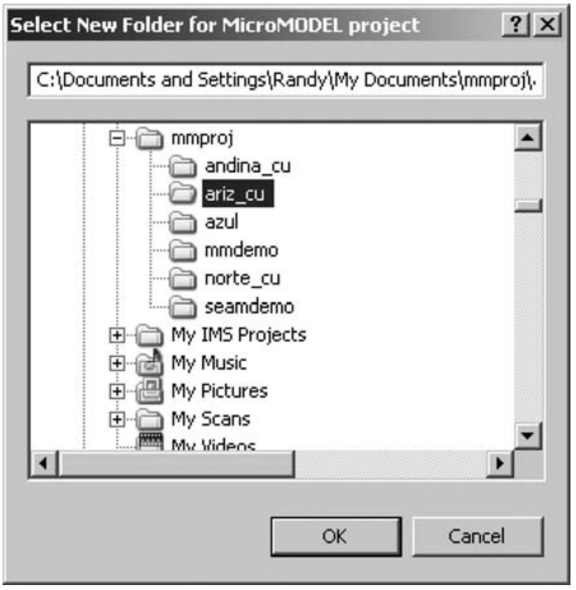


Figure 16.10. Location of the "ariz_cu" Data Set.

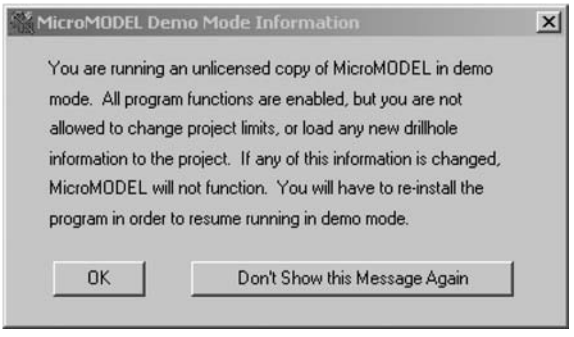


Figure 16.11. The Unlicensed Copy Warning Label.

Click "OK". At this point, you will get a dialog box (Fig. 16.11) indicating that you are using the demo version of MicroMODEL. Read the warning and then Click "Don't Show this Message Again".

As noted earlier, the MicroMODEL v8.1 Academic version allows you to change many of the parameters with the exception of:

- The model origin
- The block size
- The number of rows, columns, and levels
- The number of drill holes
- The number of sample intervals

If you should change any of these parameters, the program will issue a message that the license file is invalid and quit.

You should then see the main MicroMODEL program screen (File Manager), with 15 menu choices listed along the top of the screen (Fig. 16.12). You may also see a previously selected submenu for one of the main choices. Most likely, you will see the "Data Entry"

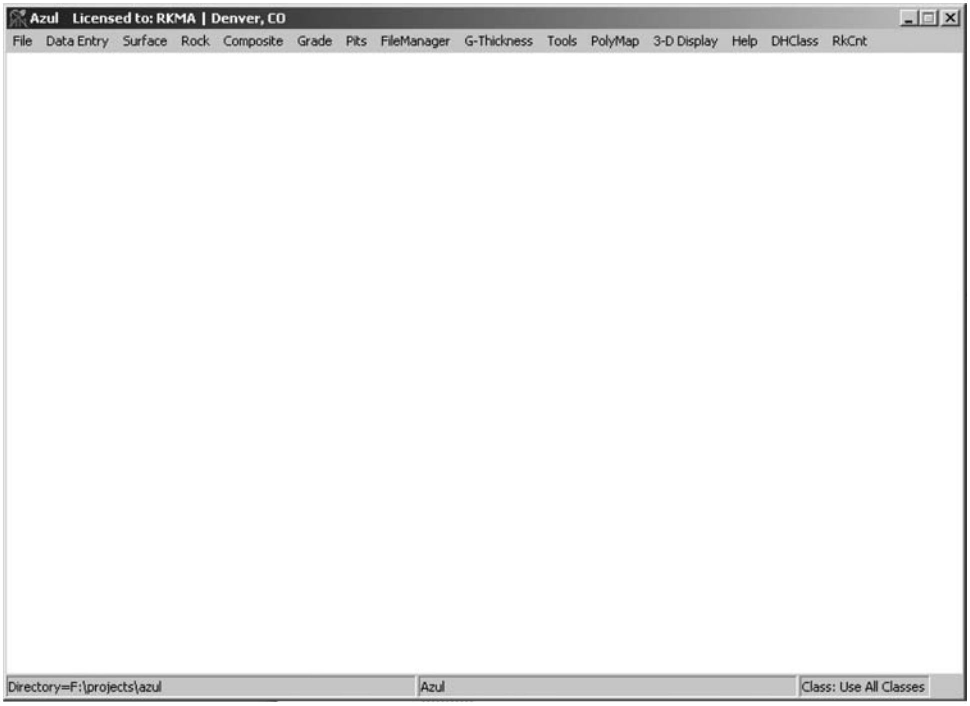


Figure 16.12. Main Menu Screen with no Submenu Selected.

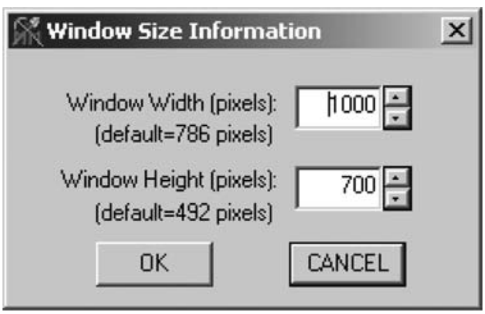
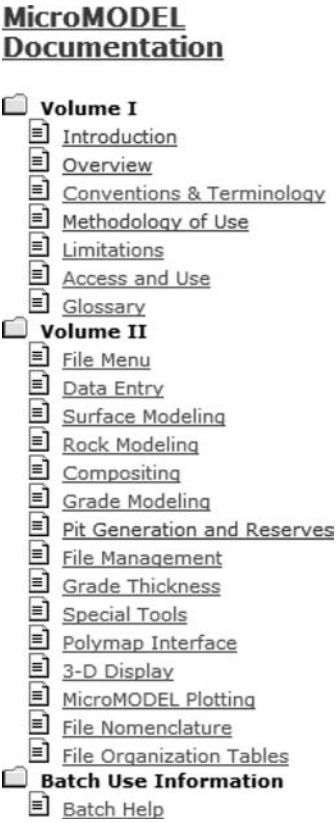


Figure 16.13. Pop-up Window for Changing the Window Size.

menu. MicroMODEL remembers what dialog you were in from previous sessions. The "Main Menu" screen with no dialog will only appear if you click "Return to Main Menu" located at the top of any of the individual modules.

16.3.5 *Some special considerations*

In most cases, you can adjust the size of the main MicroMODEL window by clicking on the "File" menu, and then choosing "Change Size of MicroMODEL Window". Use the small popup dialog shown in Figure 16.13 to shrink the height and width. Most computers will work with the default sizes shown, but you may have to leave your size "as is."


MicroMODEL Documentation

Volume I
- Introduction
- Overview
- Conventions & Terminology
- Methodology of Use
- Limitations
- Access and Use
- Glossary

Volume II
- File Menu
- Data Entry
- Surface Modeling
- Rock Modeling
- Compositing
- Grade Modeling
- Pit Generation and Reserves
- File Management
- Grade Thickness
- Special Tools
- Polymap Interface
- 3-D Display
- MicroMODEL Plotting
- File Nomenclature
- File Organization Tables

Batch Use Information
- Batch Help

Figure 16.14. Table of Contents of the MicroMODEL User Manual.

Note: When you click on the "X" in the upper right hand corner of the window, MicroMODEL will return you to the last place you were in the main menu structure. For example, if you were in "Data Entry" with choice "4 Convert Multiple Text Files and Read Into Database" selected, this is where it returns.

Windows 7 Settings – Important Information!

There are several items that need to be addressed regarding Windows 7.

- Any "Aero" themes should be avoided. It is recommended that the "Windows Classic" theme be chosen. From the Control Panel, Choose "Appearance and Personalization – Change the Theme". Then, select Windows Classic.
- Another setting which is not compatible with MicroMODEL is set in Control Panel > Ease of Access – Optimize visual display. For "Change the size of text and icons", the "Turn on Magnifier" check box should be left unchecked. Also, from this same panel, click "Change the size of text and icons". The text and other items need to be set as "Smaller – 100% (default)". If they are set at "Medium – 125%", then **many of the dialog windows in MicroMODEL will be partially obscured**.

With the main menu on the screen, by selecting the "Help" command one obtains "MicroMODEL Documentation" as shown in Figure 16.14. This is the "User Manual" referred to throughout this chapter.

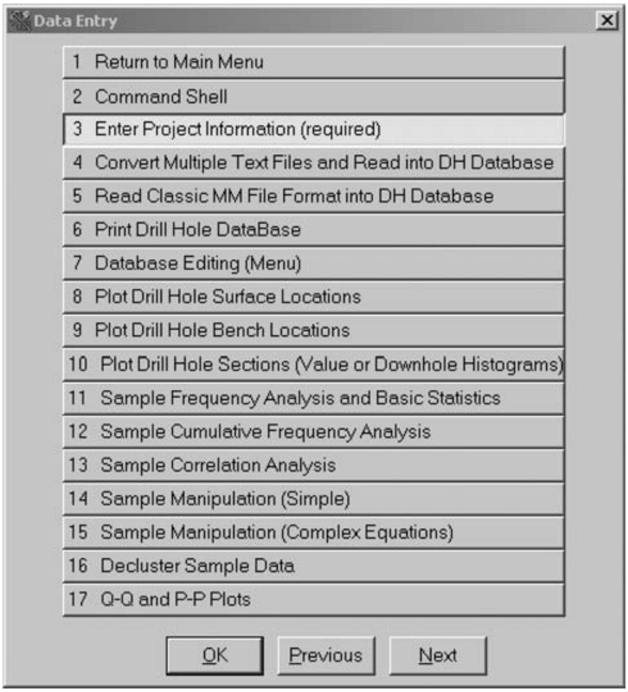


Figure 16.15. The "Data Entry" Window.

16.3.6 *Constructing the Ariz_Cu model*

It is now time to begin with the construction of the block model for the Ariz_Cu property. In the way of a review, you open MicroMODEL, then under "MicroMODEL Project to start", you select either a project you have previously accessed, or select "New Project". Click "OK". For "New Project", navigate to My Documents\mmproj\ariz_cu. click on this folder name, and then click "OK". The screen "Data Entry" will appear with the line "3 Enter Project Information (required)" highlighted (see Fig. 16.15). Click "OK".

A blank "Project Information Sheet" is shown in Figure 16.16.

You are perhaps familiar with the CSMine data set and have run the CSMine software. The CSMine data set is for a small copper orebody and consists of 40 drill holes.

At this point, it is important to point out some significant differences between the two software packages:

1. The original CSMine software included in edition 1 of this book had a maximum size of somewhat less than 10,000 blocks so the overall model size was quite small. In edition 2 and also in this edition it was expanded to 32000 blocks (maximum of 8000 blocks in plan and 255 blocks in depth). There is not the same constraint with MicroMODEL.
2. In CSMine, the "key block" for the block model is located in the left hand corner of the uppermost bench. The coordinates refer to the center of the block. For the CSMine tutorial, the key block coordinates are
   - x = 1600 ft (east)
   - y = 4550 ft (north)
   - z = 4125 ft (elevation)

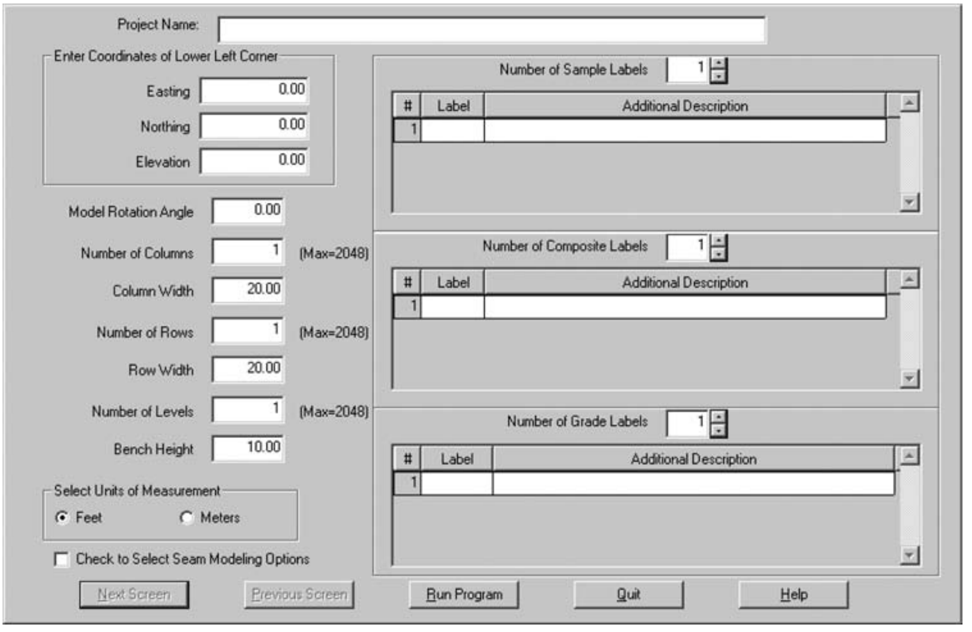


Figure 16.16. Blank Project Information Sheet.

3. For the CSMine tutorial the blocks are 100 ft × 100 ft × 50 ft high. There are 14 blocks in the x direction, 10 blocks in the y direction and 12 blocks in the vertical direction.
4. In MicroMODEL the "key block" is located at the lower left hand corner of the block model. The key coordinates refer to the coordinates at the bottom left-hand corner of the key block. In applying MicroMODEL to the CSMine data set, we will choose the key coordinates as:
   – x = 1200 ft (east)
   – y = 4200 ft (north)
   – z = 3500 ft (elevation)
5. The block size will be chosen as 50 ft × 50 ft × 25 ft. The number of blocks will be 40 blocks in the x direction (columns), 30 blocks in the y direction (rows) and there are 30 levels.

So, as you can see, there some important differences.

You are now ready to complete the "Project Information" sheet. Enter the information given above in the correct spaces. In Figure 16.17 these items have already been filled in for you.

– Easting = 1200
– Northing = 4200
– Elevation = 3500
– Rotation Angle = 0
– Number of columns = 40
– Column width = 50
– Number of rows = 30

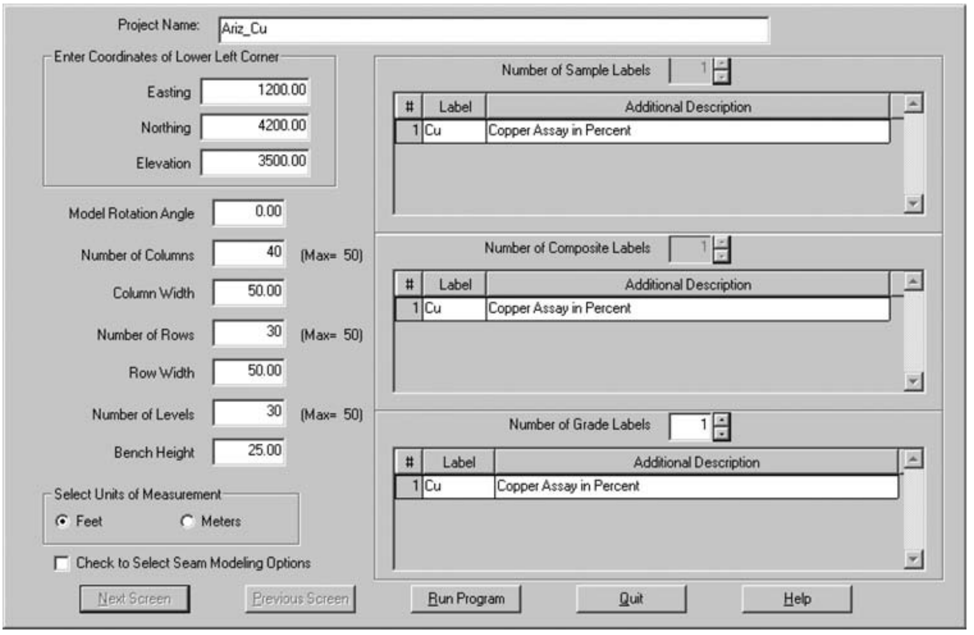


Figure 16.17. The Completed Project Information Sheet.

- Row width = 50
- Number of levels = 30
- Bench height = 25
- Units of Measurement = Feet

Since there is only one assay per interval (Cu) the "Number of "Sample Labels", "Number of Composite Labels" and "Number of Grade Labels" are all 1. Under the caption "Label" for all three categories enter the word 'Cu' and under "Additional Description" enter the words 'Copper Assay in Percent' for all three captions. The completed sheet using the information provided above is shown in Figure 16.17. Click "Run Program".

A new screen will appear "Select Print File To Display" (Fig. 16.18). The file that was just created is called "PROJPRN.PRN". Note that the extension (.PRN) may not show if you have set Windows not to display file extensions.

If you click "Open" it will show you the information you just entered on the Project Information sheet (Fig. 16.17).

If you choose not to examine the file, click the "X" to close. You are then returned to the "Data Entry" screen.

Comment 1: If you desire to use a different editor program for viewing MicroMODEL output, such as Wordpad, you can change the editor by setting the system environmental variable MMPRINTPROG to the full pathname of your editor program.

Comment 2: Now, it is probably the correct time to point out some of the other headings of the main menu. If you look at the top of the screen, you will see the headings "File, Data Entry, Surface, Rock, Composite, Grade, Pits, etc." In the "Data Entry" mode, you are entering the basic drill hole data. Under "Surface" you will be entering or creating the surface elevation file. Under "Rock" you will enter the different rock types. Under "Composite"

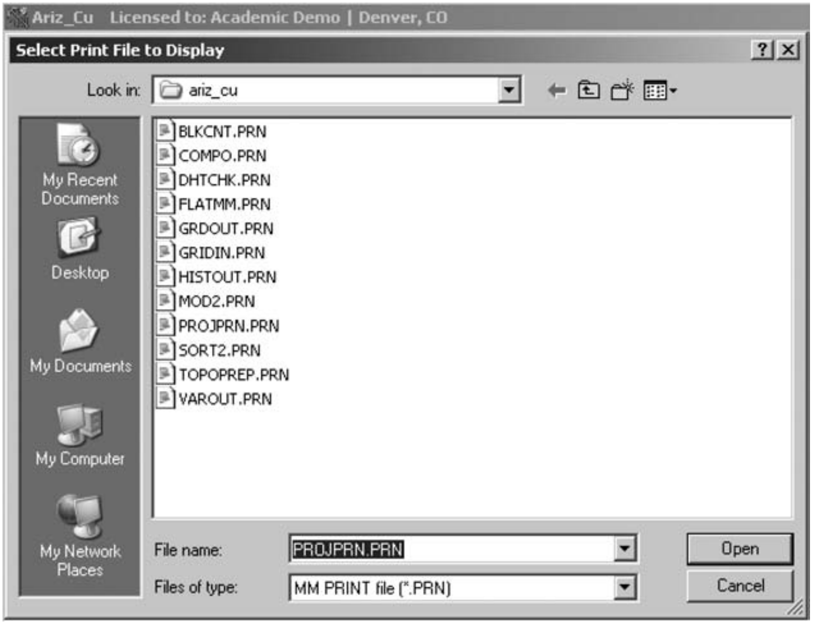


Figure 16.18. The "Select Print File to Display" Screen.

you will create the grade composites. Under "Grade" you will create the grade block model. Finally, under "Pits" you will be using the floating cone technique to obtain the pit.

On the "Data Entry" screen click on choice "4 Convert Multiple Text Files and Read into DH Database". Then click "OK". The screen shown in Figure 16.19 should appear.

There should already be a set of responses including "Convert Ariz_Cu Drill hole data files and load into MM". Highlight this choice and click "Continue". The screen shown in Figure 16.20 should appear.

This is a very important screen since it designates your input file names, sets delimiter style, and sets several other options. The chosen MicroMODEL file names are "Ariz_Cu_Collar.txt", and "Ariz_Cu_Assay.txt". These files are located in the project directory "mmproj\\ariz_cu". They can be located anywhere but, in order to simplify the installation of MicroMODEL, all work files are located in the project folder.

Note that the "Downhole Survey Information File" is not used. To change the file choice, you would simply click on the large pushbutton containing the file name, select the file, and then click "OK".

The "Delimiter Style" is already set to the appropriate values (space delimited, consecutive delimiters count as one). To change the "Delimiter Style", you would click on the "Set Delimiter Style" button below the file selection button. There is an important field in the "Set Delimiter" dialog "Data Begins on Line Number". For the two files we are using, there is one header line in each. For these files, the data begin on line number 2.

The check box "Do Not Load DH Data" is left unchecked. We DO want to load the Drill hole data.

The "If Data Already Exists" box is set to "Delete Current Data". This insures that we overwrite the current Drill hole database. If we were *appending* to the Drill hole database, we would choose "Append to Existing Data".

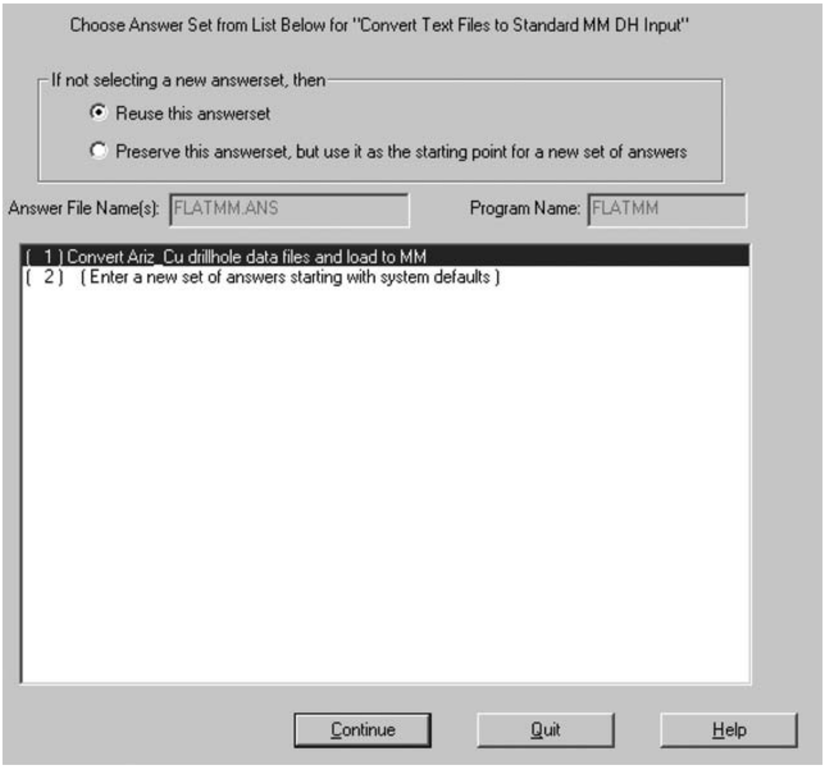


Figure 16.19. Choose Answer Set Screen.

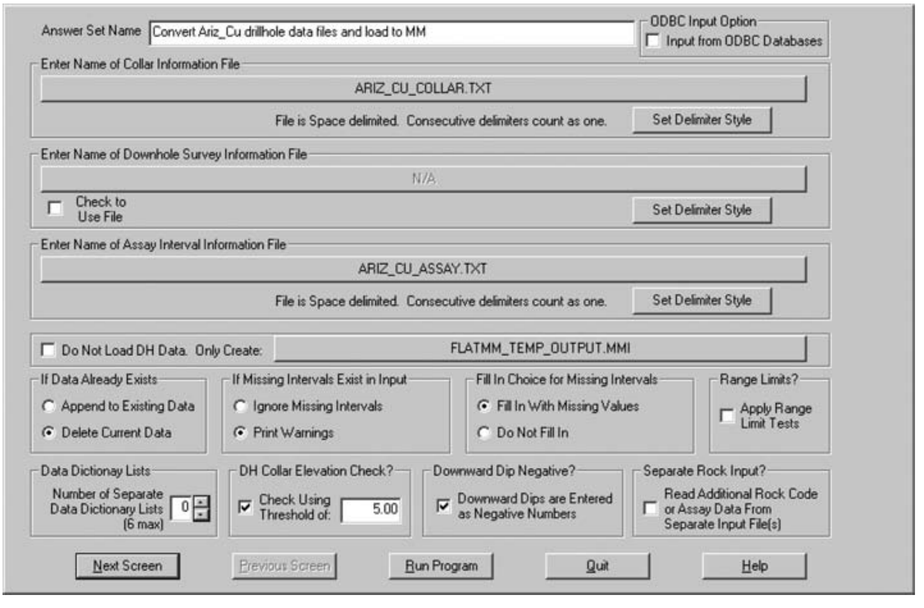


Figure 16.20. Convert Text Files – Input Screen One.

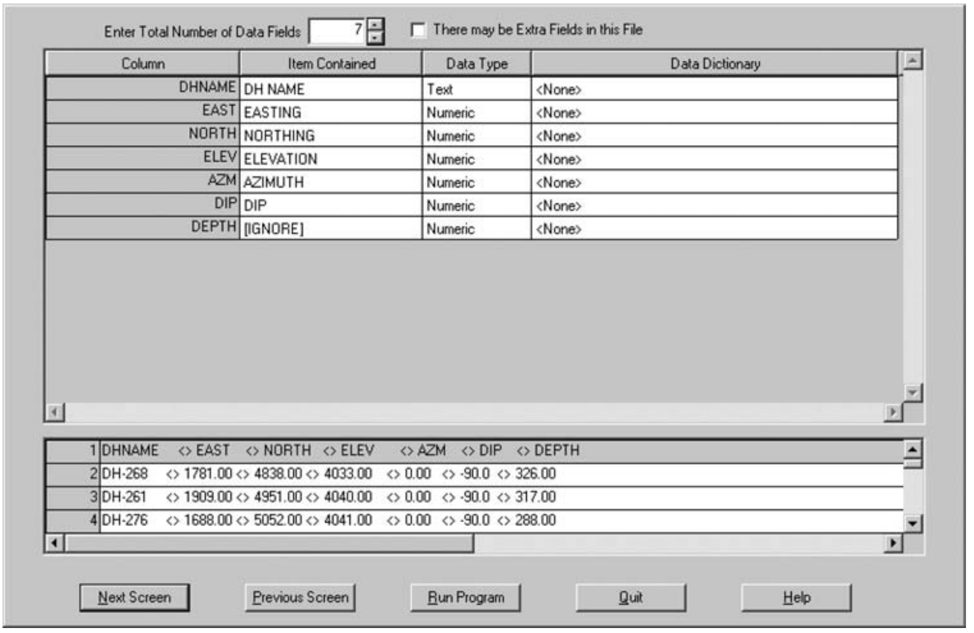


Figure 16.21. Convert Multiple Files – Collar File Column Definition.

The "If Missing Intervals Exist in Input" box is set to "Print Warnings". The "Fill In Choice for Missing Intervals" box is set to "Fill In with Missing Values". The "Range Limits" box is left unchecked. The "Data Dictionary Lists" box is set to zero since we do not have any alphanumeric fields in our data that need to be converted to integer codes.

The "DH Collar Elevation Check?" box is checked and the threshold is set to 5 feet. Although we have not as yet modeled our topo surface, one has been provided as part of the data set.

With regard to the "Downward Dip Negative?" box, your drill holes have been drilled downward. If, in your data files, you have used −90° to indicate the dip of vertical holes, then this boxed should be checked. In the sample data files, the downward vertical holes have been indicated by −90 so the box is checked.

There is no "Separate Rock Input", so this box is left unchecked.

Click "Next Screen". The screen entitled "Column, Item Contained, Data Type, Data Dictionary" as set for the Ariz_Cu_Collar.TXT appears. It is shown in Figure 16.21.

Here you must indicate the total number of data fields that you have used in creating your collar file. You have used 7 data fields. The screen shot in Figure 16.22 is a partial listing of the collar file. It is plain text and can be edited with Wordpad or Notepad.

For each column from the input file, the "Item Contained" column has been set to the appropriate MicroMODEL item name. Note that the "DEPTH" stored in the input file is being ignored. The indicated "Data Type" should be "Text" for the Drill hole name, and "Numeric" for all other columns. You should check the preview display at the bottom of this screen to be sure that the file is getting parsed correctly. Note that the top header row is greyed out because we specified that our data begins on line 2.

DHNAME	EAST	NORTH	ELEV	AZM	DIP	DEPTH
DH-268	1781.00	4838.00	4033.00	0.00	-90.0	326.00
DH-261	1909.00	4951.00	4040.00	0.00	-90.0	317.00
DH-276	1688.00	5052.00	4041.00	0.00	-90.0	288.00
DH-100	2434.00	4830.00	4007.00	0.00	-90.0	360.00
DH-102	1903.00	5041.00	4060.00	0.00	-90.0	294.00
DH-95	2005.00	4944.00	4055.00	0.00	-90.0	253.00
DH-78	2479.00	5003.00	4000.00	0.00	-90.0	186.00
DH-69	2620.00	4787.00	4066.00	0.00	-90.0	513.00
DH-105	2054.00	5112.00	4088.00	0.00	-90.0	195.00
DH-99	1933.00	4859.00	4019.00	0.00	-90.0	219.00
DH-273	1906.00	5105.00	4074.00	0.00	-90.0	348.00
DH-256	2146.00	5006.00	4062.00	0.00	-90.0	239.00

Figure 16.22. Partial Listing of Collar File.

DHNAME	FROM	TO	Cu
DH-100	0	5	-999.99
DH-100	5	10	0.04
DH-100	10	16	0.02
DH-100	16	22	0.005
DH-100	22	26	0.1
DH-100	26	32	0.12
DH-100	32	37	0.09
DH-100	37	41	0.03
DH-100	41	45	0.06
DH-100	45	49	0.1
DH-100	49	53	0.09
DH-100	53	57	0.1
DH-100	57	61	0.13
DH-100	61	64	0.35
DH-100	64	68	0.12
DH-100	68	71	0.09
DH-100	71	75	0.13
DH-100	75	78	0.1
DH-100	78	82	0.13
DH-100	82	85	0.05

Figure 16.23. Partial Listing of Assay Interval File.

Click “Next Screen”. You now repeat the procedure for the file “Ariz_Cu_Assay.TXT”. It has 4 data fields. A partial listing is shown in Figure 16.23. Note that the copper assay for the first interval has been designated as missing, using the standard MicroMODEL flag value (−999.99). As with the collar interval file, this file is plain text and can be edited with Wordpad or Notepad.

The “Data Type” entries should be “Text” for the drill hole name and “Numeric” for the other three columns. The filled in screen is shown in Figure 16.24.

Click “Next Screen”. Now click “Run Program”. Forty holes and 2017 samples should load. The resulting screen appears in Figure 16.25.

Click “OK”. The screen “Select Print File to Display” shown in Figure 16.26 will appear if everything has gone correctly.

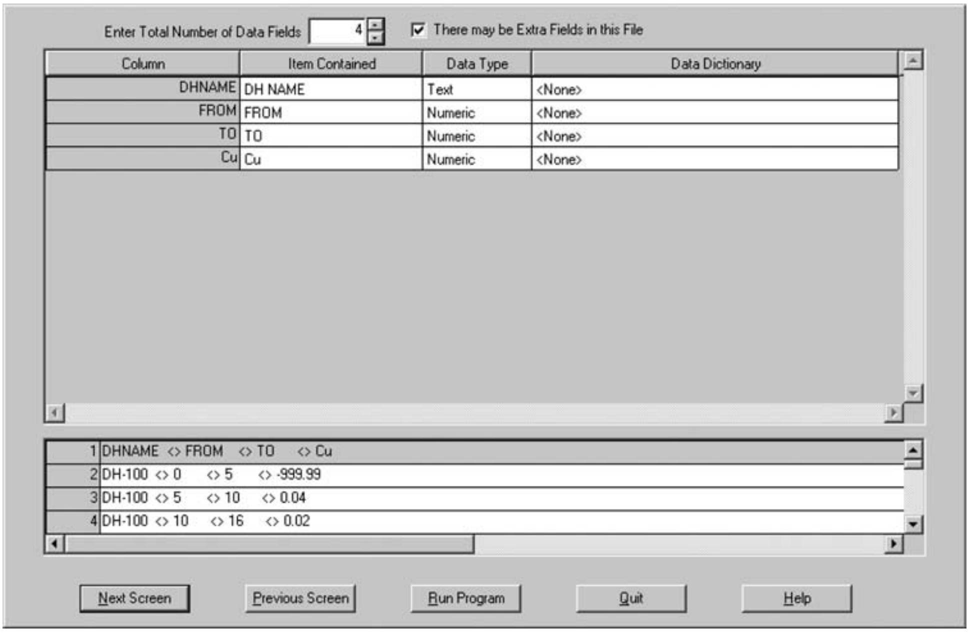


Figure 16.24. Convert Multiple Files – Assay Interval File Column Definition.

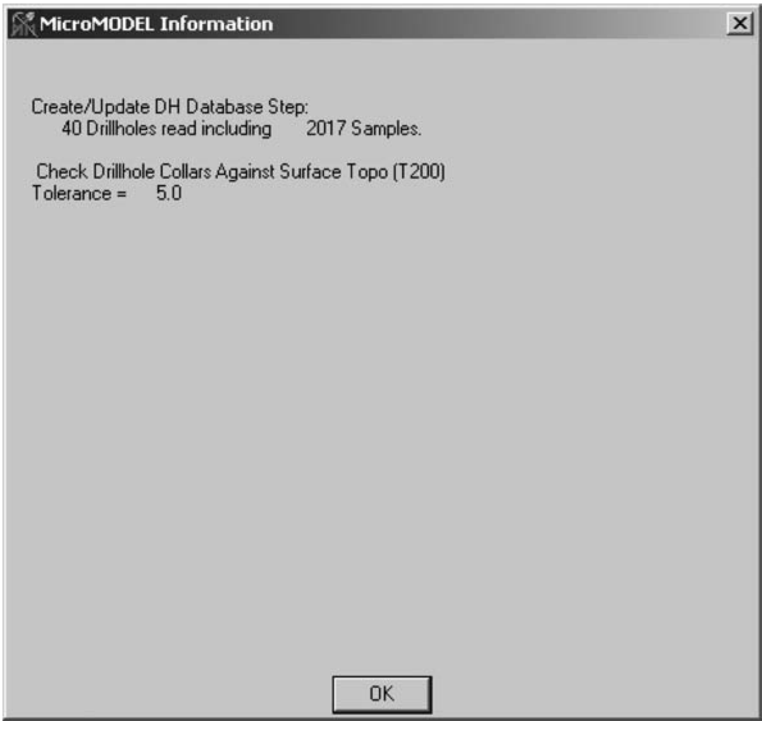


Figure 16.25. Convert Multiple Files – Summary Screen.

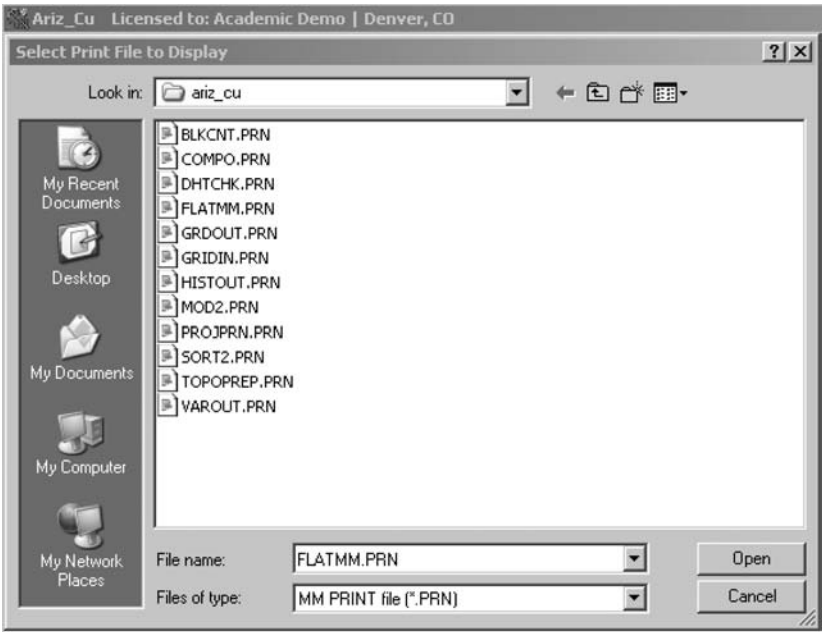


Figure 16.26. Select Print File to Display Screen.

```
*********************************************************
Reading Collar File ARIZ_CU_COLLAR.TXT
*********************************************************
Reading Assay File ARIZ_CU_ASSAY.TXT
**********************
Number of Collars Read          40
Number of Surveys Read           0
Number of Assays Read         2017
**********************
```

Figure 16.27. Convert Multiple Files Printout.

The file “FLATMM.PRN” is chosen. If you now click on “Open” you will get the screen shown in Figure 16.27.

Here it shows that there were 40 collars read, 0 surveys read and a total of 2017 assays. Click on the “X” to close the screen. You are returned to the “Data Entry” screen.

If you are interested, you can go to topic “6 Print Drill Hole Data Base” or to topic “7 Database Editing (Menu)”. However, you do not need to look at these to continue.

On the “Data Entry” screen, move down to choice “8 Plot Drill Hole Surface Locations” and click “OK”. Since an “Answer Set Name” already exists, “Plot Drill hole Collar Locations”, just click “Continue”. The screen shown in Figure 16.28 will appear.

Accepting the options shown, click “Next Screen”. The next screen (shown in Fig. 16.29) is entitled “Display Drill hole Surface Locations – Information Options”.

Accept the options shown. Click “Next Screen”. Click “Run Program”.

You are then asked to “Choose Answer Set From List Below For Title Block”. Choose the first one and click “Continue”. The screen shown in Figure 16.30 appears.

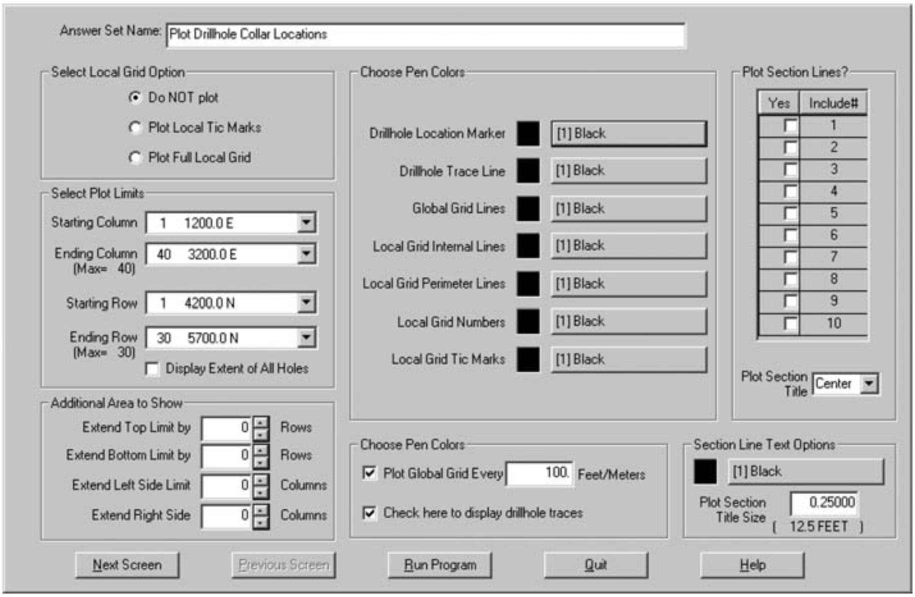


Figure 16.28. Plot Drill Hole Surface Locations – Input Screen One.

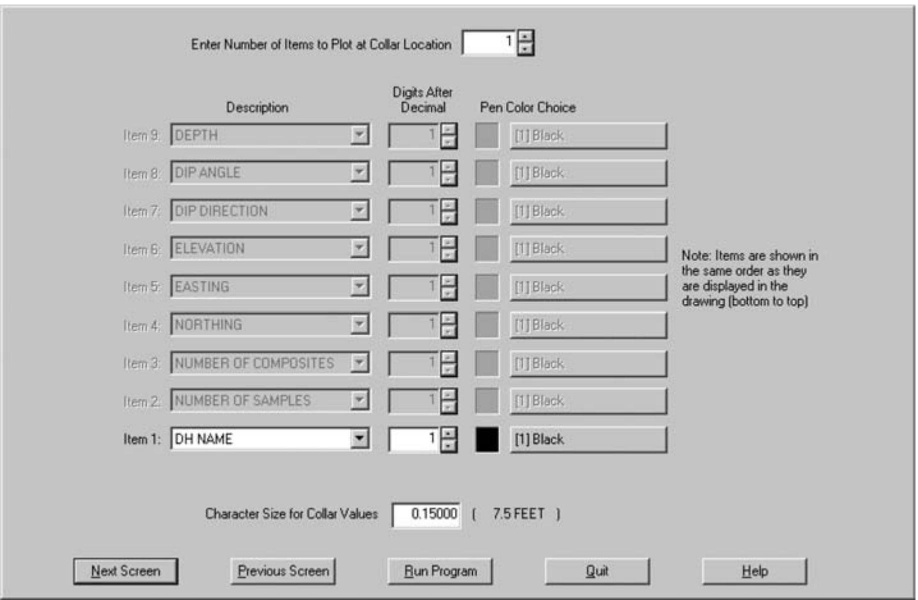


Figure 16.29. Plot Drill Hole Surface Locations – Input Screen Two.

To simplify the demonstration, we will not include the title block at this time. You do this by leaving the box "Check here to plot title block" unchecked. Click "Run Program".

You then get the screen shown in Figure 16.31 which says "Select Plot File to Display". The file "DHPLT.PLT" is highlighted. Click "Open".

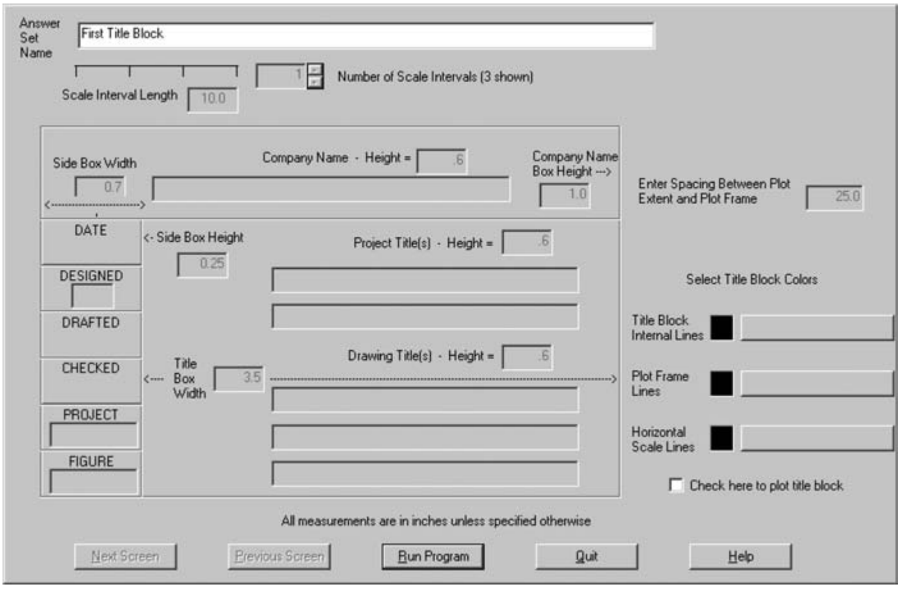


Figure 16.30. Title Block Input Screen.

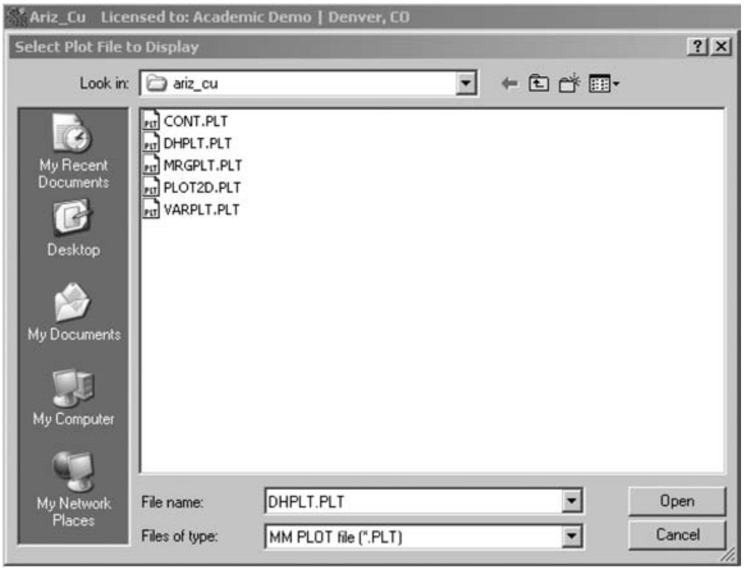


Figure 16.31. Select Plot File to Display Screen.

You will then get the screen "Select Scale for Plot Number 01". See Figure 16.32. Enter the scale as shown in here and click "OK". It should be noted that the scale which is entered here only affects the size of the title block that is displayed relative to the plot size. Since we are not using a title block, the scale is irrelevant.

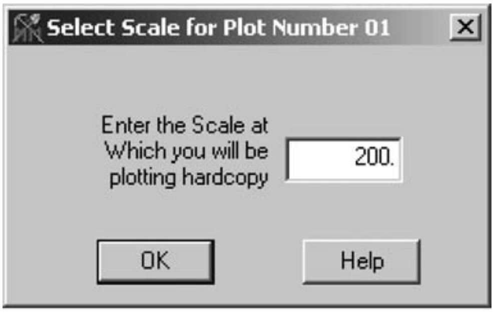


Figure 16.32. Plot Scale Input Screen.

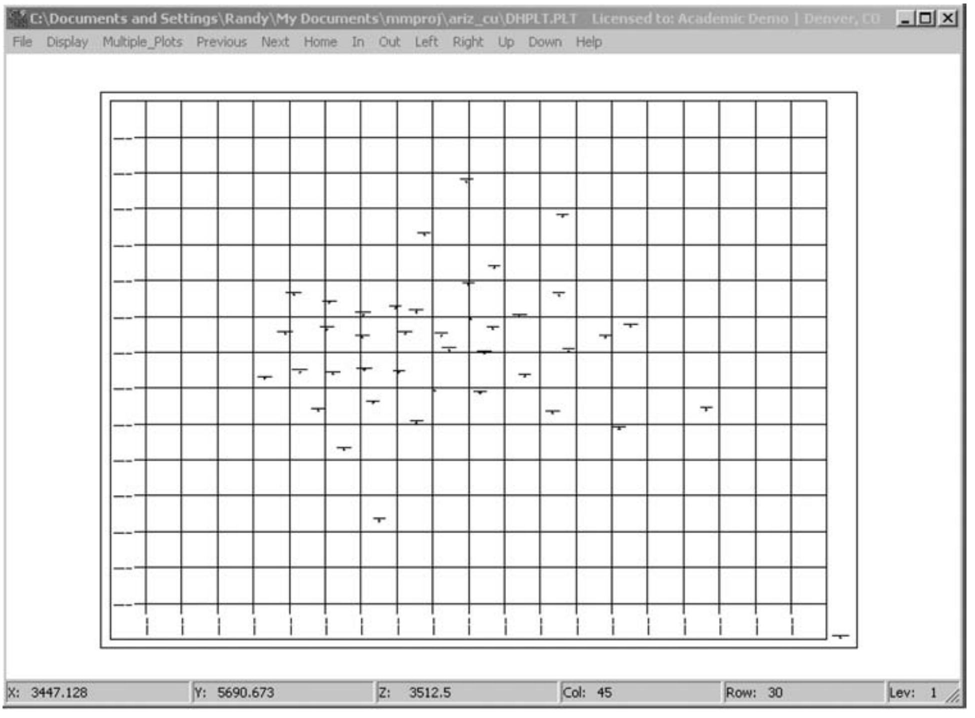


Figure 16.33. Collar Location Plan View with Text (F3) Toggled to "OFF".

You may then get the screen shown in Figure 16.33 which looks like a mistake because no text is displayed.

Press the function key F3 and the numbers will be turned on as shown in Figure 16.34. The F3 toggle is sometimes used when panning large plots which contain thousands of text characters.

You can print out this drawing by clicking on "File" at the top of the screen.

Returning to the "Data Entry" screen, you should be able to try topics "9 Plot Drill Hole Bench Locations" and "10 Plot Drill Hole Sections" by yourself.

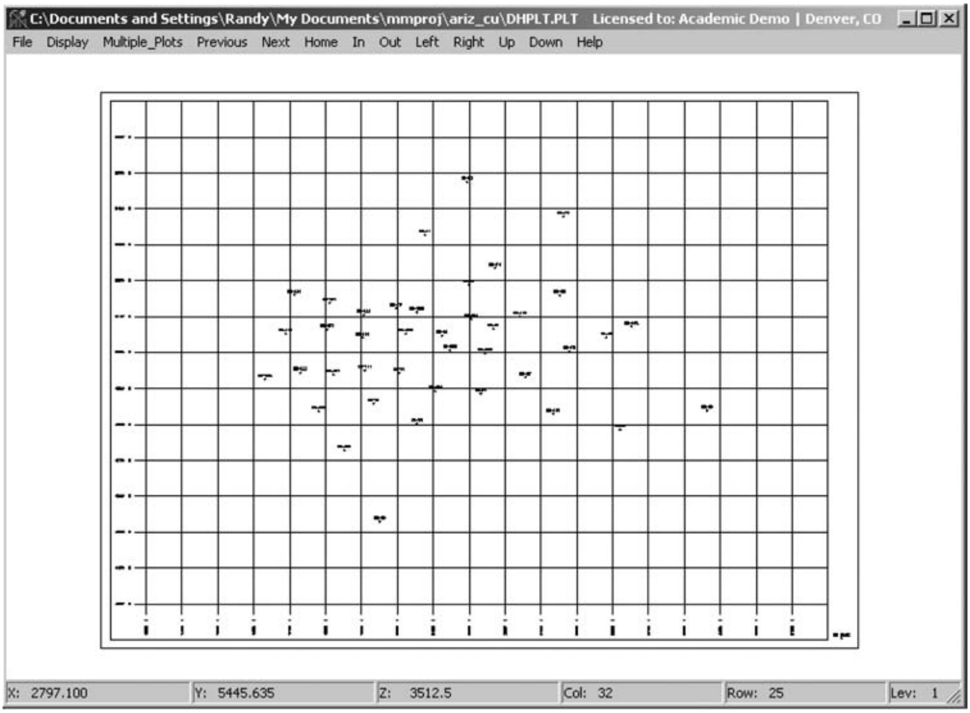


Figure 16.34. Arizona Copper Project – Plan View of Collar Locations.

This completes the "Data Entry" tutorial. You are now ready to

1. Proceed to the entry of surface elevations, compositing, constructing grade block models, etc. using the "CSMtest" data set.
2. Learn more about the different options within "Data Entry" which were not covered in this tutorial.

16.4 PIT GENERATION TUTORIAL

16.4.1 *Introduction*

In section 16.3, "Data Entry", instructions were provided regarding entering the basic drill hole information. In this section, "Pit Generation Tutorial", the remaining steps will be described. The CSMine data set will be used and the example will build on the base established in section 16.3.

16.4.2 *Surface topography*

Introduction

Before the mineral block model can be constructed, the surface topography must be generated. There are several ways that this can be done. If the computer upon which MicroMODEL has been installed is connected to a digitizing tablet, then a file containing the digitized topography can be created directly. The "Manual" accompanying the software goes through this

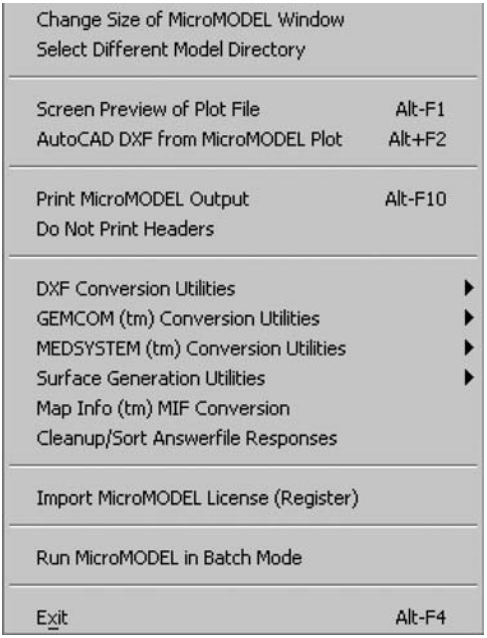


Figure 16.35. File Dropdown Menu Choices.

procedure in some detail. Here it will be assumed that the topography is not entered in this way. In this tutorial two different techniques will be demonstrated.

(1) *Importing an existing AutoCAD® topography file*
Often a topography file is available separately in the form of an AutoCAD drawing. MicroMODEL contains a utility that can change the .dxf file generated by AutoCAD into the type of file required as input by MicroMODEL. Although this procedure is described in Section 6.6 of the "Manual," some additional instructions will be supplied here. MicroMODEL should be compatible with the latest release of AutoCAD DXF files.

It is assumed that the DXF file containing topo information is stored on the computer on which the modeling is being done. In MicroModel you click on "File" and the screen shown in Figure 16.35 is displayed.

Select "DXF Conversion Utilities" and the arrow (>) will lead you to "Topo Line Conversions". The screen shown in Figure 16.36 will appear. In this case, AutoCAD polylines from layer 0 contained in the file Surface1.DXF will be converted to the standard MicroMODEL digitized topo file POLY.CNT.

(2) *Surface and thickness modelling*
For the Arizona Copper Project, we will be extracting topo values from the standard digitized topo data file (POLY.CNT) which we just converted from AutoCAD.

Surface modeling is accomplished in three steps. First, the data points are prepared. Next, the data points are sorted. Finally, the topo grid is modeled based on the sorted data points.

On the top line of the main MicroMODEL menu, one moves from the "Data Entry" module to that entitled "Surface". Clicking on "Surface" one obtains the menu entitled "Surface and Thickness Modeling" shown in Figure 16.37.

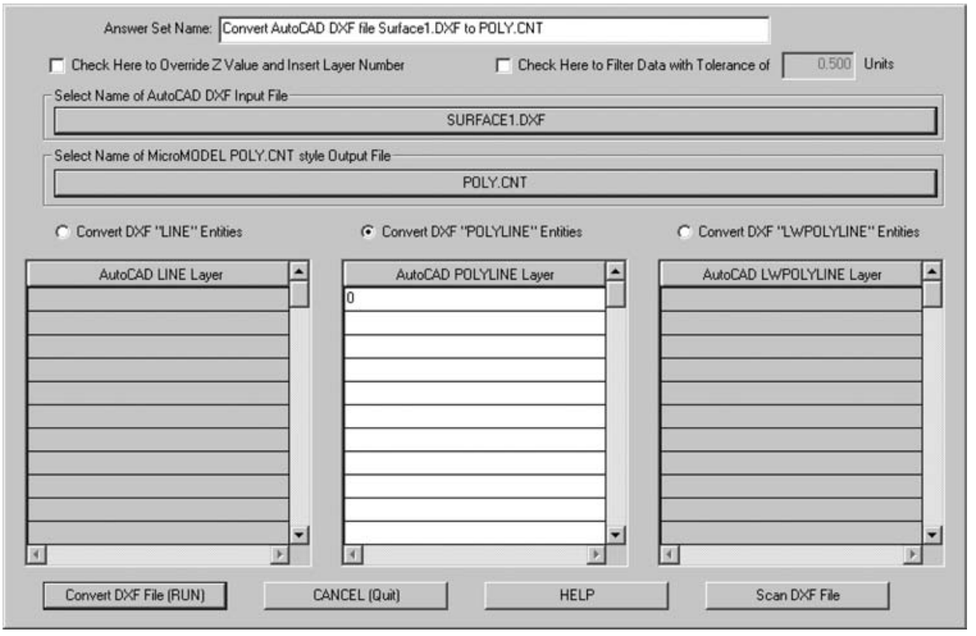


Figure 16.36. Topo Line Conversions Input Screen.

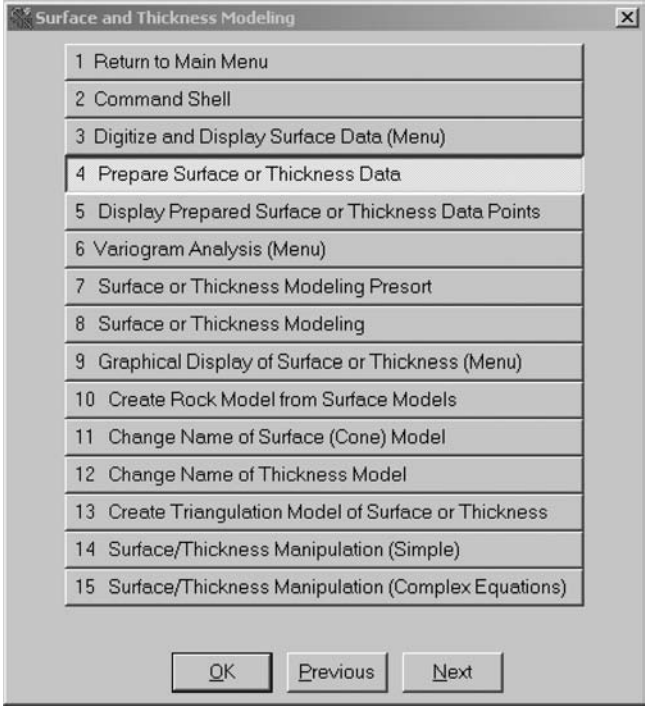


Figure 16.37. Surface and Thickness Modeling Submenu.

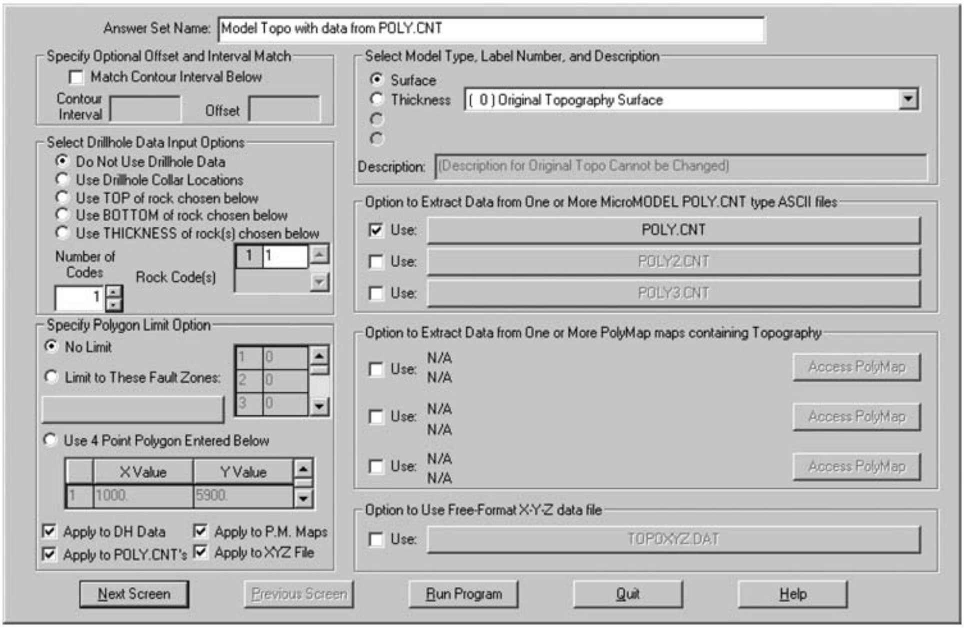


Figure 16.38. Prepare Surface Data Input Screen.

If one wants to create the digitized topography file using MicroMODEL then the cursor should be moved down to line "3 Digitize and Display Surface Data (Menu)".

If the surface topography is being entered via an AutoCAD file or being generated using drill hole collar elevations, then one proceeds to the highlighted line "4 Prepare Surface or Thickness Data". Click "OK".

The screen entitled "Select Model to Create and Data Source(s) for Topo Modeling" will have "Model Topo with data from POLY.CNT" highlighted. Type "Continue". MicroMODEL will use data from POLY.CNT to develop the surface topography file for use in the block model. No other input data are selected. The "Answer Set" is shown in Figure 16.38.

Note that we have selected "Do Not Use Drill hole Data". If you check the option "Use Drill hole Collar Elevations" they will be included in preparing the topo map. There is a feature in the software that allows you the possibility to check the "goodness" of the given hole collar positions versus the topo map so it is better to not include them at this stage. Under the heading "Specify Polygon Limit Option", check the line "No Limit". Under the heading "Select Model Type, Label Number, and Description", check "Surface". From the alternatives, select "Surface (0) Original Topography Surface". Then click "Run Program".

You now get a screen entitled "Select Print File To Display" with the file name "TOPOPREP.PRN" highlighted. Click "Open". You will see a brief printout telling you that 3329 data points were used. Click the "X" and you will be returned to the main "Surface" menu.

Select choice "5 Display Prepared Surface or Thickness Data Points". Click "OK". You will get the screen "Choose Answer Set from list below for 'Plot Surface/Thickness Locations"'. The first answer set should be selected. Click "Continue". You will now get the screen entitled "Answer Sheet: Plot Prepared Surface/Thickness Data Point Locations".

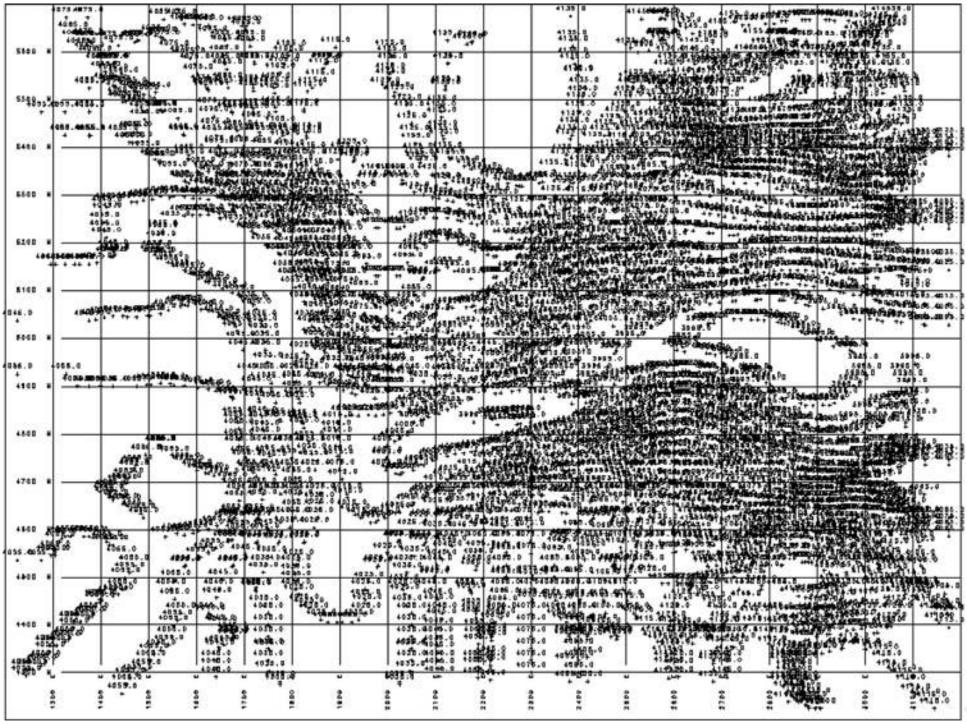


Figure 16.39. Arizona Copper – Plan View Display of Prepared Surface Data.

Click "Next Screen". Click "Next Screen". You will now get the screen entitled "Check here to Display Numeric Data Values for Surface/Thickness Shown". Change the item "Digits After Decimal" to a zero "0". Click "Next Screen". Click "Run Program". You will get the screen "Choose Answer Set – Title Block". Click "Continue". You will get the screen "Answer Sheet Name – First Title Block". Since at this point in time we are not worrying about title blocks, the item "Check here to plot title block" has been left unchecked. Click "Run Program". The next screen is entitled "Select Plot File to Display" with the file name "PLOT2D" highlighted. Click "Open". Click "OK".

You will now obtain the display shown in Figure 16.39 in which the individual digitized elevation points are shown in plan view.

It is very important to note that to be able to generate a floating cone pit, each of the 2-D surface grid locations must have an assigned elevation. Click on the "X". You are now returned to the "Surface" menu.

Click on line "7 Surface or Thickness Modeling Presort". Click "OK". The screen has the title "Choose Answer Set from the list below for Select Topography Modeling Sort Parameters". Select "Topography Modeling Presort". Click "Continue". You will get the screen "Answer set name Topography Modeling Presort". The screen is shown in Figure 16.40.

We will assume that the isotropic option applies so check the line "Check here if the data is isotropic". To assure that all of the blocks will be assigned an elevation, under the box "Enter Search Range Limits," change the range to 300 as shown in Figure 16.40.

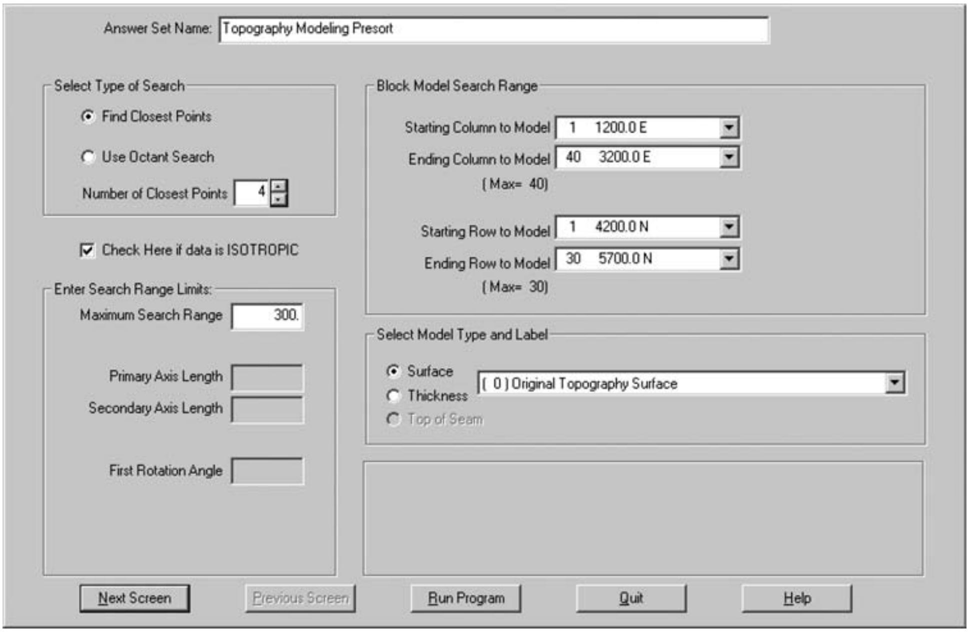


Figure 16.40. Surface Modeling Presort Input Screen.

Click “Run Program”. You will now get the screen “Select Print File To Display” with “SORT2” highlighted. Click “Open”. The screen will indicate the area over which the topography will be calculated. Click on the “X”.

You are returned to the main “Surface” menu. Click on line “8 Surface or Thickness Modeling”. Click “OK”. The screen “Choose Answer Set from the list below for ‘Select 2-D Modeling Parameters’” appears. Select “Model Surface Topography”. Click “Continue.” The screen shown in Figure 16.41 appears.

You can use either “Kriging” or the “Inverse Distance to Power” method to assign elevations to the blocks using the given data. Choose “IDP” and use the “Power” of 2.00 (this is the common inverse distance squared technique). Click “Run Program”.

The “Select Print File To Display” screen appears with “MOD2” highlighted. Click “Open”. A summary is provided of the parameters used.

By clicking on “X”, you are returned to the main “Surface” menu.

Click on line “9 Graphical Display of Surface or Thickness (Menu)”. Click “OK”. The next menu is entitled “Graphical Display of Surface/Thickness Model Sub-Menu”. Click on line “4 Plan View Cell Plot of Grid Values”. Click “OK”. From the screen “Choose Answer Set from list below” for “Plot Surface/Thickness Values”, select answer set 1. Click “Continue”. The first screen should look like Figure 16.42.

Click “Next Screen”. The screen is shown in Figure 16.43.

For “Item 1: Original Topo Surface” enter zero (0) for “Digits After Decimal”. Change the “Character Size For Cell Values” to 0.15. Click “Run Program”.

Since we are not worrying about the title blocks at this point, click “Continue”. Click “Run Program”. The “Select Plot File to Display on Screen” appears with “CELL”

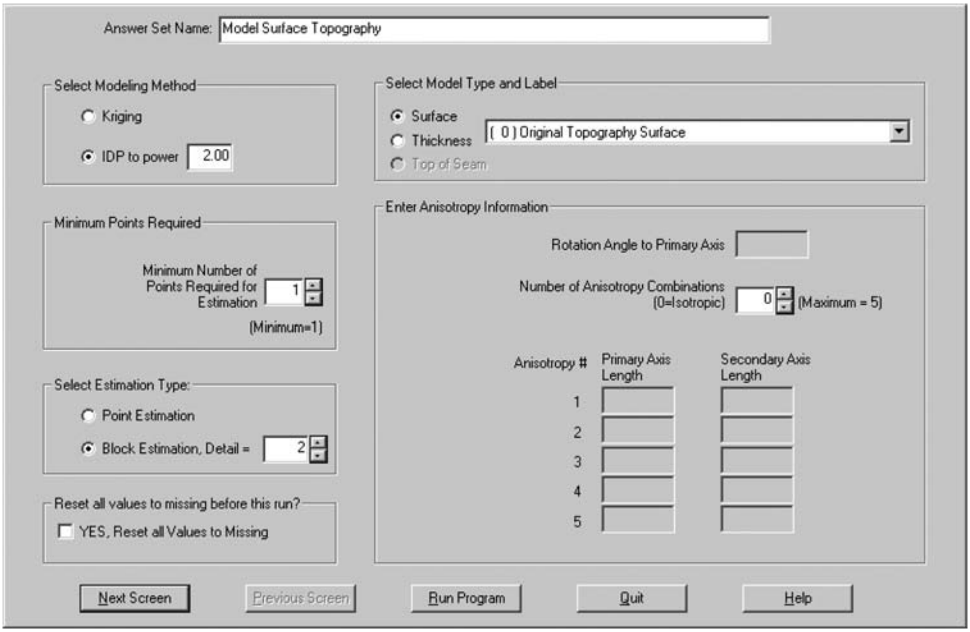


Figure 16.41. Surface Modeling Input Screen.

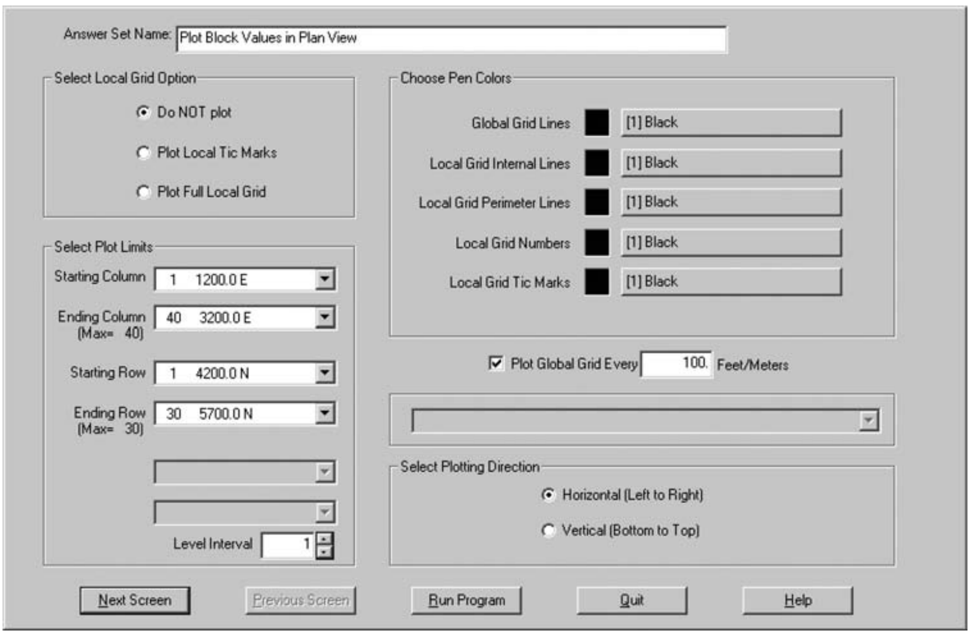


Figure 16.42. Plan View Cell Plot of Grid Values – Input Screen One.

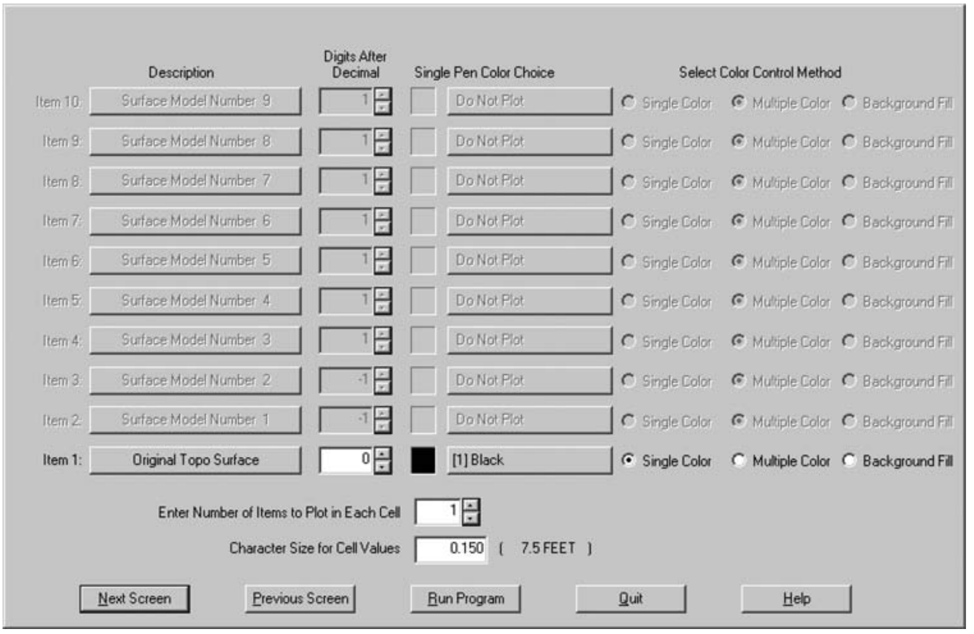


Figure 16.43. Plan View Cell Plot of Grid Values – Input Screen Two.

highlighted. Click "Open" and click "OK". You will now get a plan view map showing the elevations assigned to each block. This is shown in Figure 16.44.

Click the "X" which will return you to the sub-menu "Graphical Display of Surface/ Thickness Model". Select choice "5 Contour Grid Values". Click "OK". Click "Continue". Click "Run Program". Continue until you get the "Select Plot File to Display on Screen" display with "CONT.PLT" highlighted. Click "OK". You will then get a contour plot of the surface like the one shown in Figure 16.45.

This should be compared with the original digitized AutoCAD topography map to make sure that no mistakes have been made. Click the "X". You are returned to the sub-menu "Graphical Display".

16.4.3 *Rock modeling*

On the main menu screen, you should select the module "Rock". In this Academic Version of the program, there is only 1 rock type permitted. Select choice 5 "Create/Update Rock Model From Plan Polygons". Click "OK". You will get a screen shown in Figure 16.46 entitled "Answer Set Name 'Create/Update Rock Model From Plan Polygons". Under "Initialization Options" click "Yes, Initialize with Background Code, But DO NOT Apply Digitized Shapes". The "Background Rock Code = 1". You can ignore the "Digitized Data File Name" as it is not being used.

Click "Run Program". You may get a MicroMODEL Error "Warning: Rock Polygon File Does Not Exist". Click "OK".

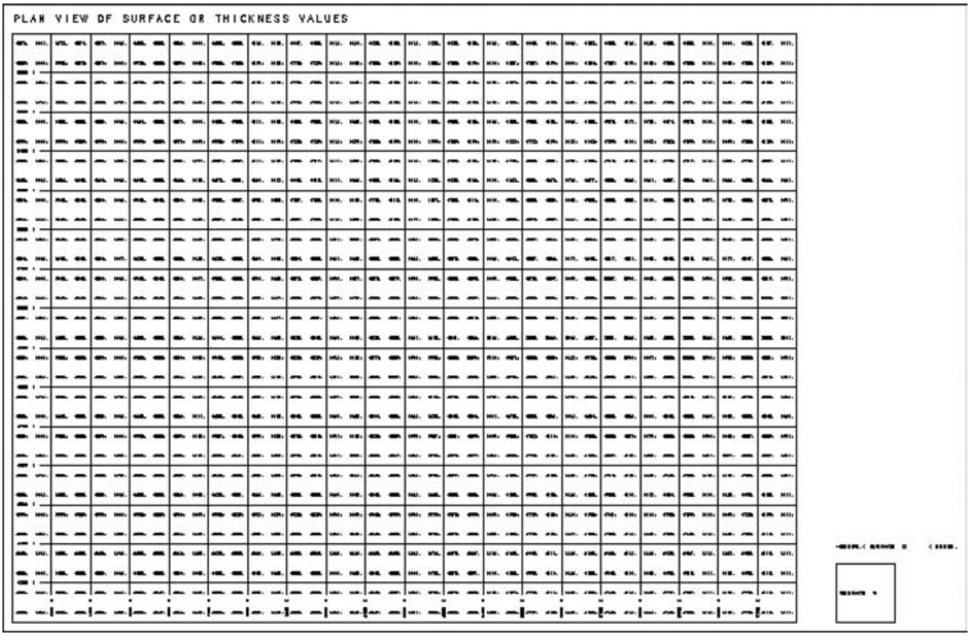


Figure 16.44. Arizona Copper – Plan View Plot of Surface Grid.

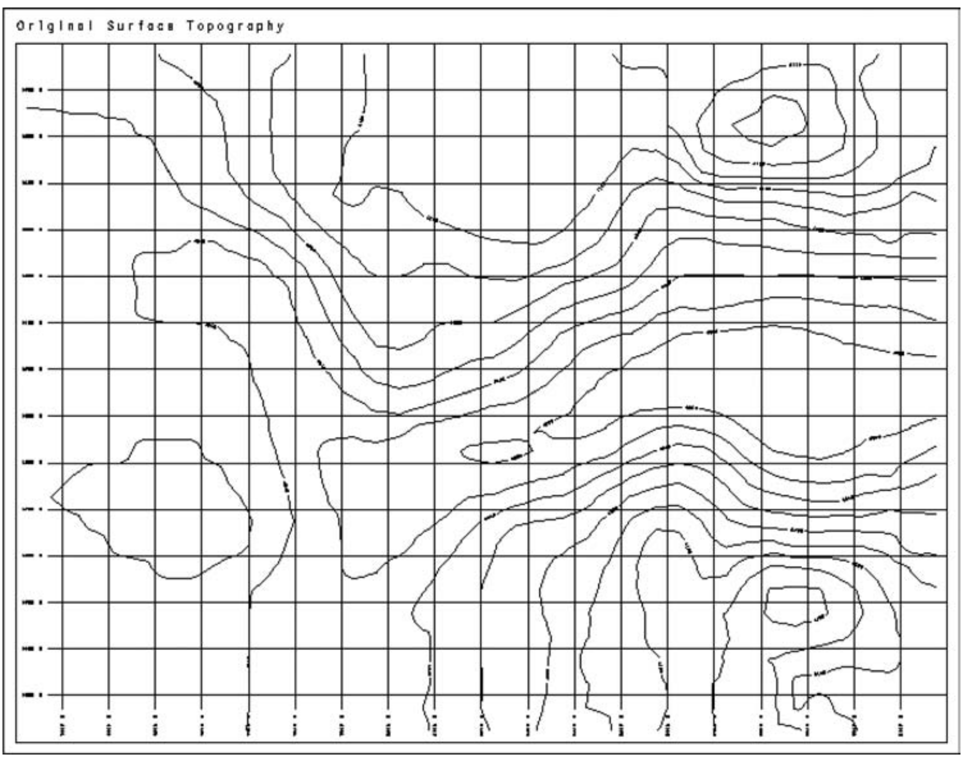


Figure 16.45. Arizona Copper – Contour Plot of Surface Grid.

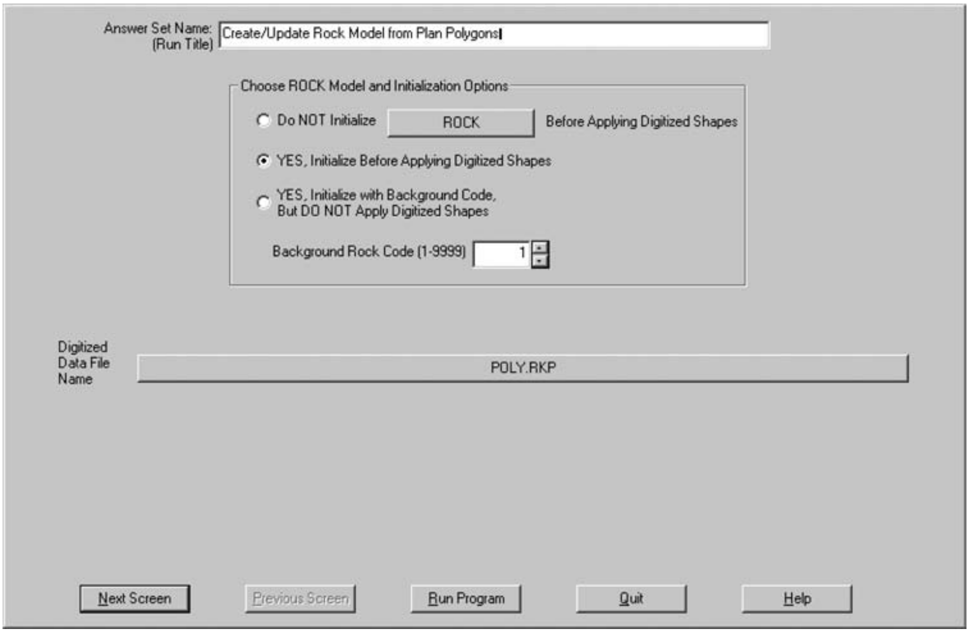


Figure 16.46. Initialize Rock Model Input Screen.

Go to choice 8 "Graphical Display of Rock Model (Menu)". Click "OK". Go to "7 Plot Cross Section of Block Grades". Click "OK". You will get the screen "Answer Set Name: Plot Cross Section of Block Grades" as shown in Figure 16.47.

Leave all items as is. Click "Next Screen".

The screen name is "Plot Cell Values in Section". For "Item 1" click on "Description" and in the popup dialog, change model to "(0) ROCK". The "Model Type" does not matter. Click "OK". Click "Next Screen" three times. Click "Run Program". Click "Continue". You do not need to make the "Title Block". Click "Run Program". The progress bar will stay busy as we are generating a set of cross section plots, one for each row in the model. Select Plot File to Display "XSECT.PLT". Click "Open". Click "OK". An hourglass will appear while the plot file is being read. You will get a cross section plot at row 1 showing a rock code of 1 below the surface and "0" for the air above.

Click on "Next" located along the top edge of your screen, or press the plus key of your numeric key pad, to see the other rows. The topography profile is shown in each section. Click "X".

16.4.4 *Compositing*

On the main menu screen, you should select the module "Composite". The Compositing Dialog will be shown. Click on line "3 Calculation of Composite Values". Click "OK". Choose answer set 1 and continue. The screen "Choose Compositing Parameters" will appear as in Figure 16.48. We are generating bench composites. Click "Run Program".

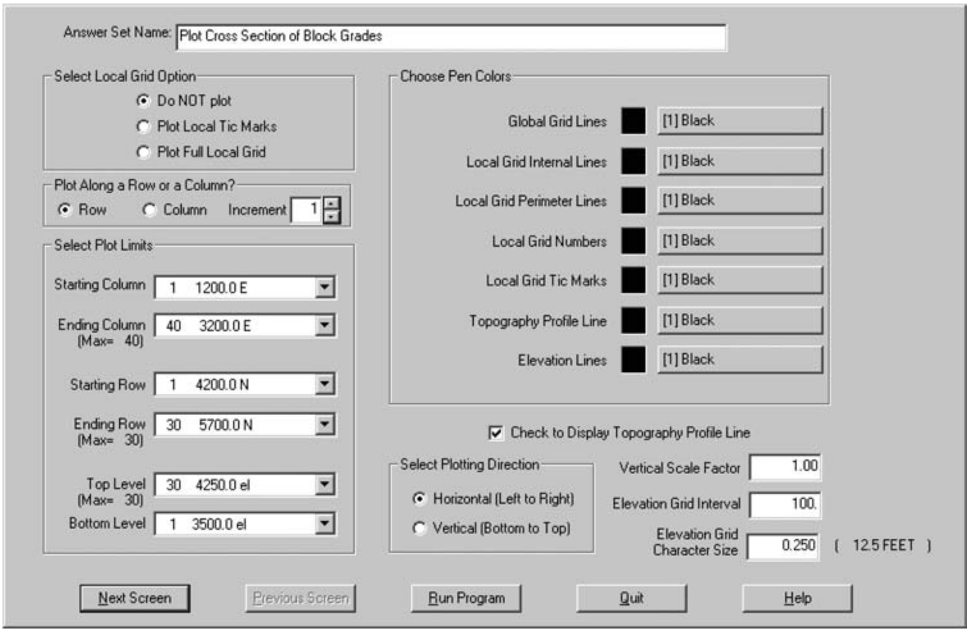


Figure 16.47. Plot Cell Values in Section – Input Screen One.

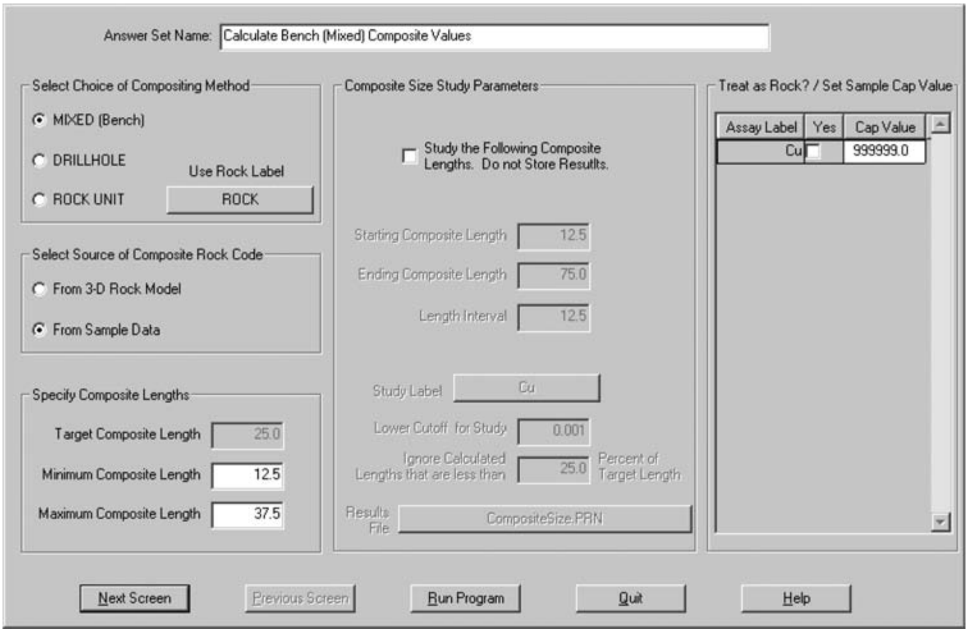


Figure 16.48. Calculation of Composite Values – Input Screen One.

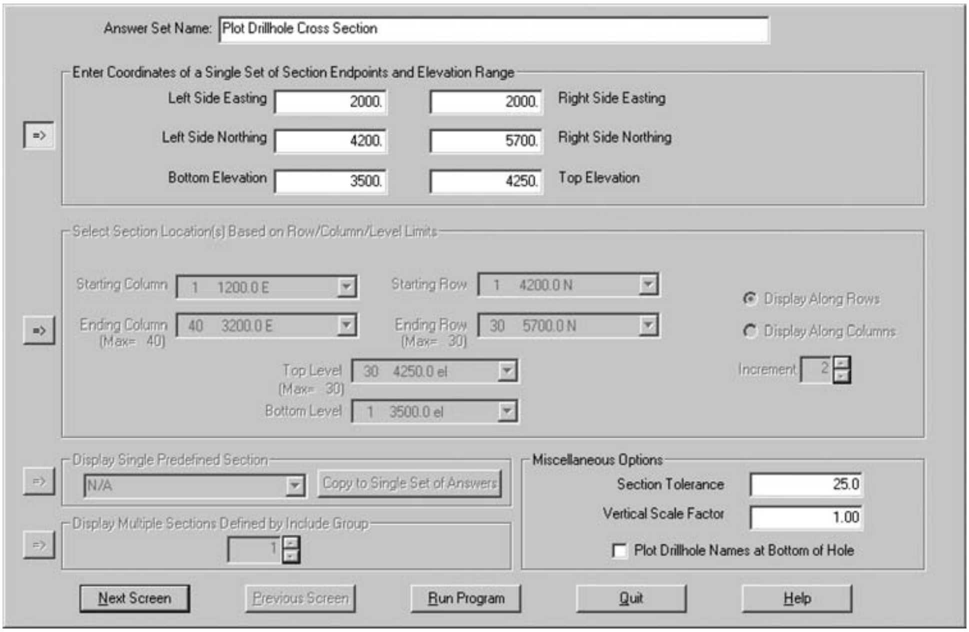


Figure 16.49. Plot Drill Hole Sections – Input Screen One.

You should then return to the "Compositing" menu. You can then select choice "10 Plot Drill Hole Sections (Value or Downhole Histograms)" to view the composites which have been calculated. The screen is as shown in Figure 16.49.

Along the left hand side of the screen, you will see that there appears to be four arrows (=>). Click the upper left radio button (=>) that points to the Single Set of Coordinates Group Box. As an example, to examine the composites on the 2000E section, change the coordinate values under "Plot Drill hole Cross Section" to

Left Side Easting = 2000
Right Side Easting = 2000
Left Side Northing = 4200
Right Side Northing = 5700

Change the Section Tolerance to 25.0. Click "Next Screen". You will get the screen shown in Figure 16.50.

Under the "Select Item to Plot" option, select "Plot Label Value". Choose "Cu". Under the "Select Type of Value to Plot" option, select "Plot Numerical Values". Under the "Select Type of Data" option, select "Plot Composited Values". Under the "Select Pen Control Method" option, select "Multiple Pen Colors". Finally, set the "Number of Digits After the Decimal" to "2". Click "Next Screen". This screen as shown in Figure 16.51 is used to select color ranges.

Display copper below 0.1 percent in green, below 0.2 percent in yellow, below 0.3 percent in orange, and anything greater than 0.3 percent in red. While still on the color range selection screen, click "Save Cutoff Setup to File", use the default name GRADE.CUT, and click the "Save" button. This file will be used later when displaying 3-D copper block grades in plan view. Click "Run Program".

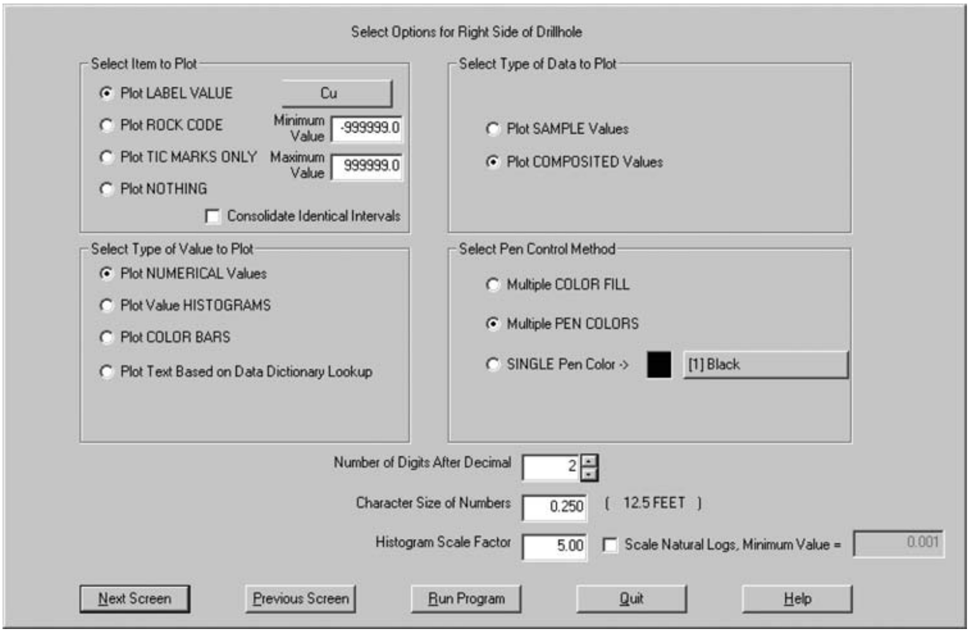


Figure 16.50. Plot Drill Hole Sections – Input Screen Two.

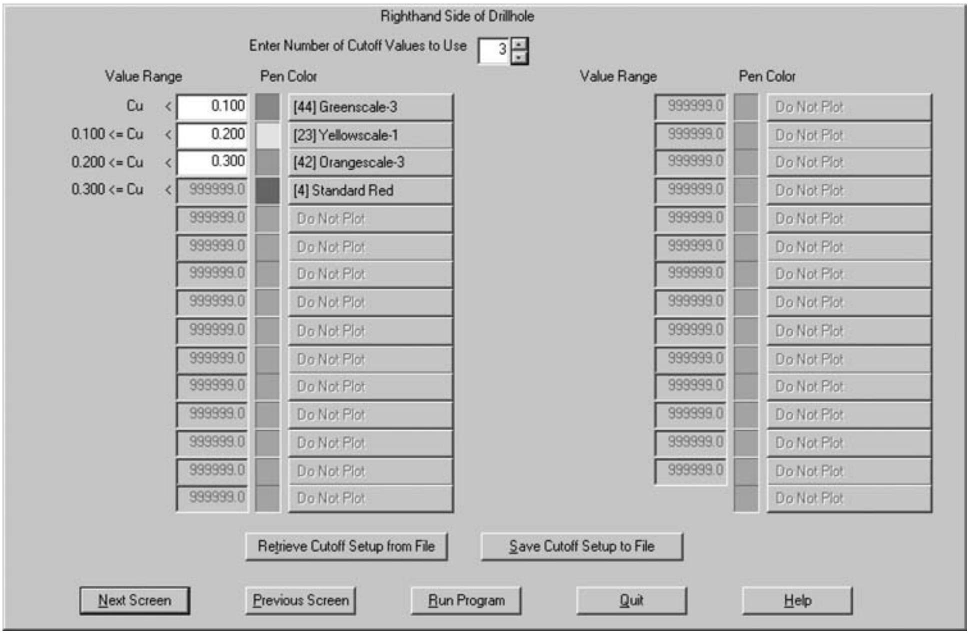


Figure 16.51. Plot Drill Hole Sections – Color Range Input Screen.

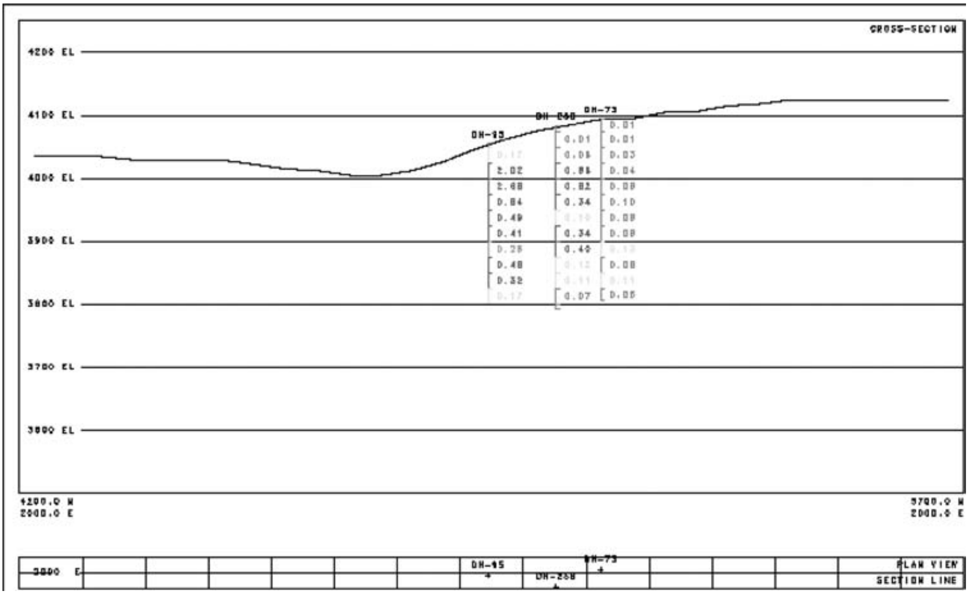


Figure 16.52. Arizona Copper – Drill Hole Section Plot.

Cycle through the title block screens as before. Click “Run Program”. The “Select Plot File to Display on Screen” will come up with the file “SECT.PLT” highlighted. Click “Open”. Click “OK”. You will get the display shown in Figure 16.52.

16.4.5 *Grade modeling*

Return to the main menu screen, and select the module “Grade”. Go to line “8 Grade Modeling Presort”. Click “OK”. Click “Continue”. The screen shown in Figure 16.53 will appear.

Under the heading “Select Type of Input Data and Label” set to “Composite Cu”. Use the 4 closest points. Check the ISOTROPIC box and change Maximum Search Range to 300. Under the headings “Rock Code Input Selection” and “Rock Codes to Interpolate” elect to “Use ALL Rock Codes” and to “Interpolate ALL Codes”.

Click “Next Screen”. You will get the screen shown in Figure 16.54.

Click “Next Screen”. Click “Run Program”. You will get the screen “Select Print File to Display” with the file “SORT3.PRN” highlighted. Click “Open”. It provides a summary of the Pre-Sort process. Click “X”.

You are returned to the “Grade” menu. Select choice “9 Grade Modeling”. Click “OK”. You will notice that the “Choose Answer Set from the list below for 3-D grade modeling” choice is “Model grade values with Kriging”. Although this is incorrect, accept it and click “Continue”. You will get the screen shown in Figure 16.55.

Change the “Answer Set Name” to “Model grade values with IDP2”. Set “Source of Presorted Data” to Composite Cu. Set the “Modeling Method” to IDP to power and set the power to 2.0. The “Output Label” is Cu. Set the other values as shown in Figure 16.55.

Click “Next Screen” and be sure your responses are set as in Figure 16.56.

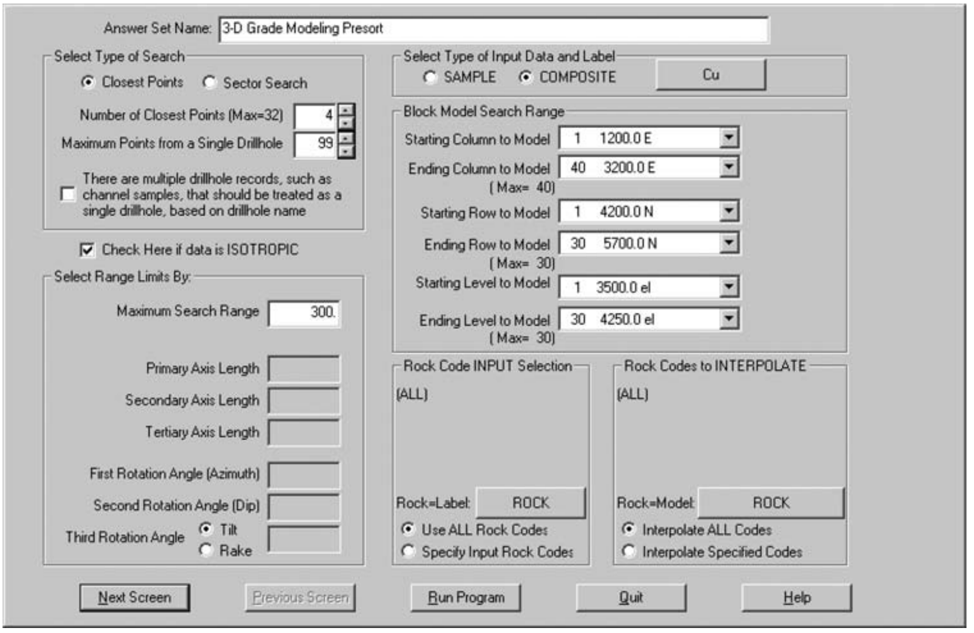


Figure 16.53. Grade Modeling Presort – Input Screen One.

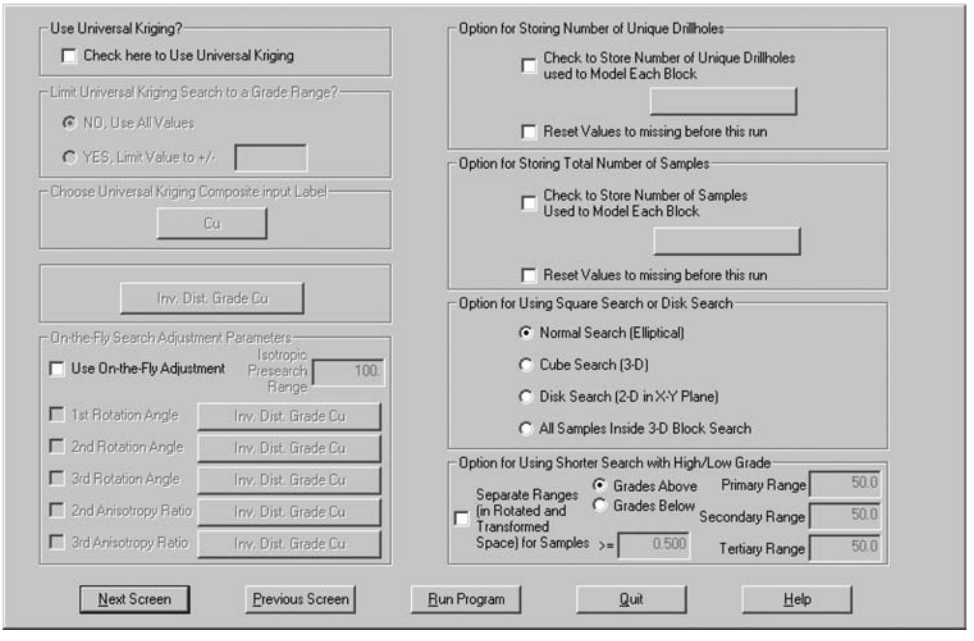


Figure 16.54. Grade Modeling Presort – Input Screen Two.

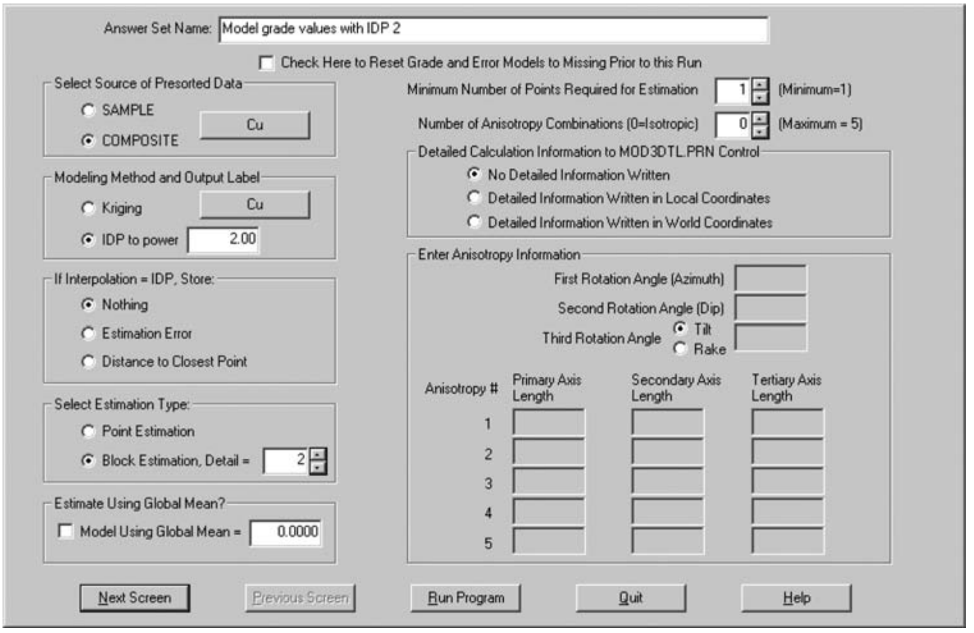


Figure 16.55. Grade Modeling – Input Screen One.

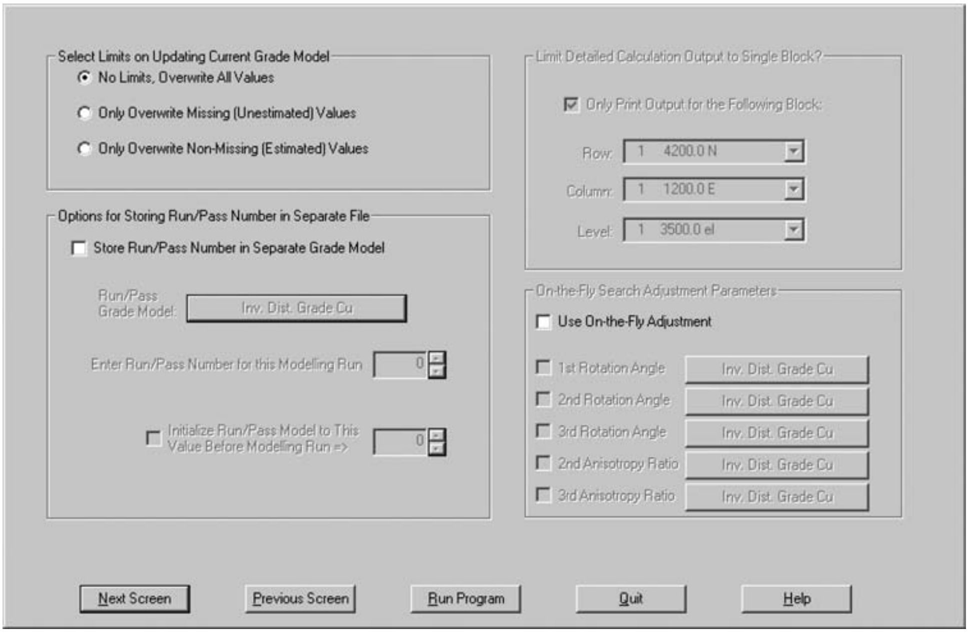


Figure 16.56. Grade Modeling – Input Screen Two.

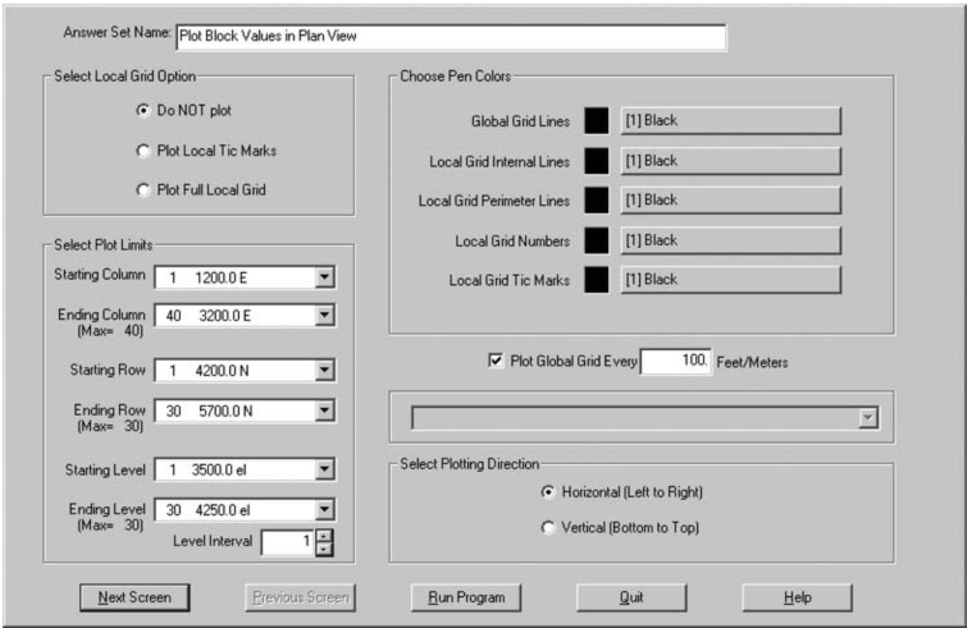


Figure 16.57. Plan View Cell Plot of Grid Values – Input Screen One.

Click "Run Program." You will get the screen "Select Print File to Display" with the file "MOD3.PRN" highlighted. Click "Open". Some statistics are provided.

Return to the "Grade" menu and click on the line "13 Graphical Display of Grade Model (Menu)". Click "OK". Click on the line "4 Plan View Cell Plot of Grid Values". Click "OK". Click "Continue". You will get the screen shown in Figure 16.57.

Under the heading "Select Plot Limits", change "Ending Level" to 30 so you can see all of the benches. Click "Next Screen". For "Item 1: Inv. Dist. Grade Cu" change to "2" Digits After Decimal. Set the "Color Control Method" to "Multiple Color". Click "Next Screen" twice.

If you saved the coloring scheme for copper grades when you were generating the composite section plots, you can click "Retrieve Cutoff Setup from File", select GRADE.CUT, and click Open. If not, be sure to set the grade cutoff ranges and colors as in the screen shown in Figure 16.58.

Click "Run Program". Click "Continue". Click "Run Program". You will get the screen "Select Plot File to Display on Screen" with the file "CELL.PLT" highlighted. Click "Open". Click "OK". You will now see level 1, toe elevation 3500 feet, showing the Cu grades assigned to each cell (block). To move up in the levels click the "Next" key at the top of the screen. An example level display is shown in Figure 16.59.

16.4.6 *Pit creation*

In order to design a pit using the floating cone analysis program, the minimum requirement is to have a surface grid, a rock model, and a grade model. We have generated all three in the

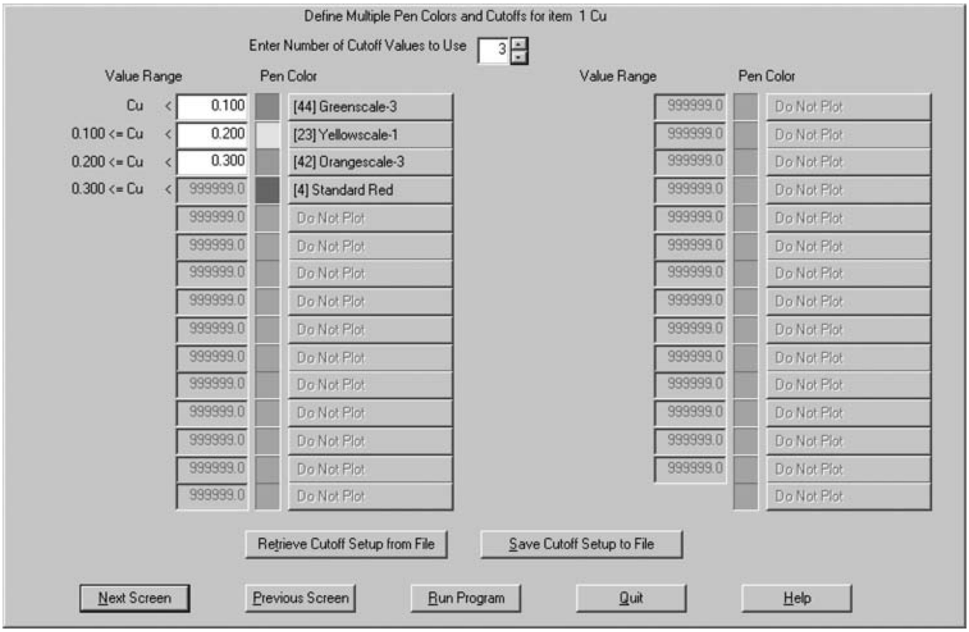


Figure 16.58. Plan View Cell Plot of Grid Values – Color Range Input Screen.

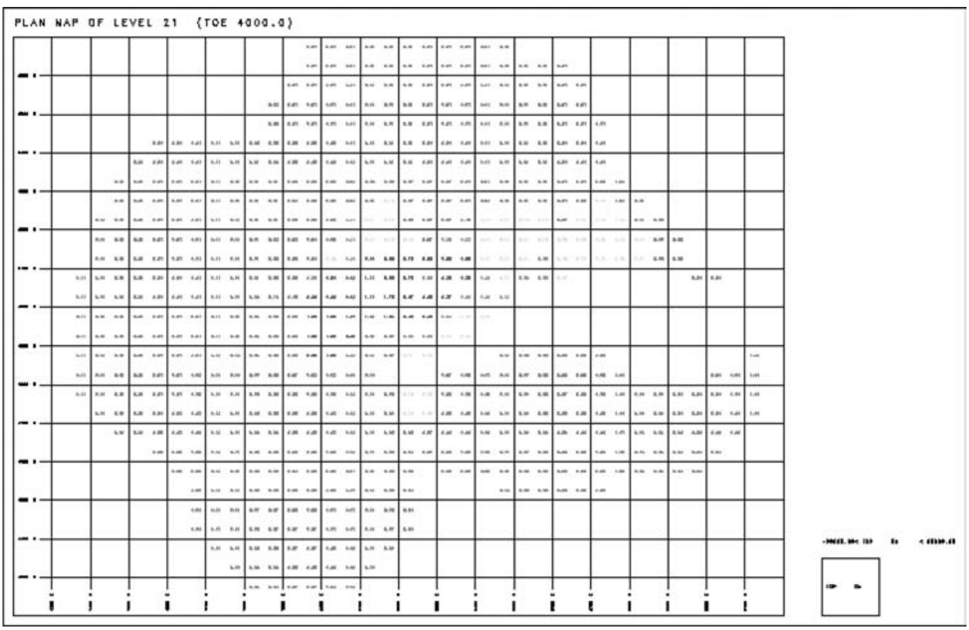


Figure 16.59. Arizona Copper – Plan View Cell Plot of Grid Values.

Table 16.2. Ariz_Cu project: Floating cone design parameters.

Parameter	Unit	Base Case
METAL PRICE		
Copper Price	US$/lb Cu	$ 3.00
Sulfide Flotation	–	–
Copper Recovery	%	80.00%
Process Cost	$ per ton ore	$ 8.00
Mining Cost		
Open Pit Mining	$ per ton	$ 2.50
Calculated Breakeven Cutoff	%	0.22
Slope Stability		
Overall Final Pit Slope Angle	Degrees	45

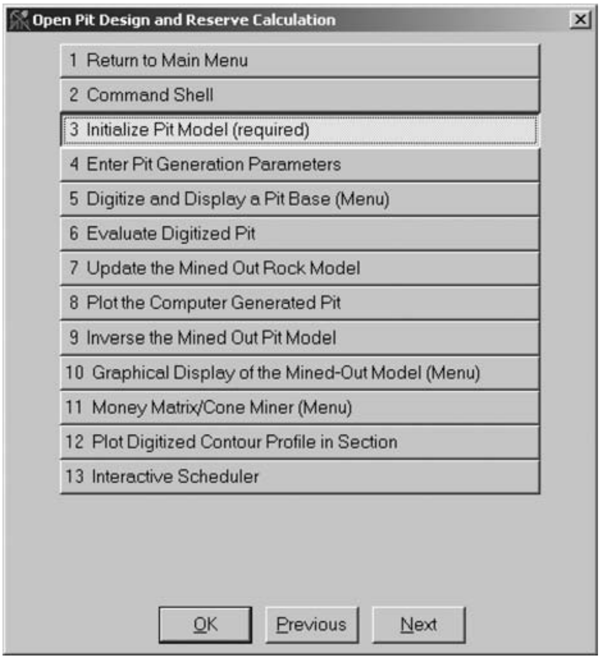


Figure 16.60. Open Pit Design and Reserve Calculation Submenu.

previous steps. We also need some economic parameters. Table 16.2 reflects some typical costs associated with a copper flotation operation in the western USA. We will use these numbers in our cone analysis.

Navigate to the "Pits" module. The full title is "Open Pit Design and Reserve Calculation" and the selections are shown in Figure 16.60.

Select choice "3 Initialize Pit Model (required)". Click "OK". Click "Continue". The screen "Choose Rock Tabulation Method" appears. Select "Automatic Rock Code Tabulation". Click "Run Program". The screen "Select Print File to Display" appears. The file

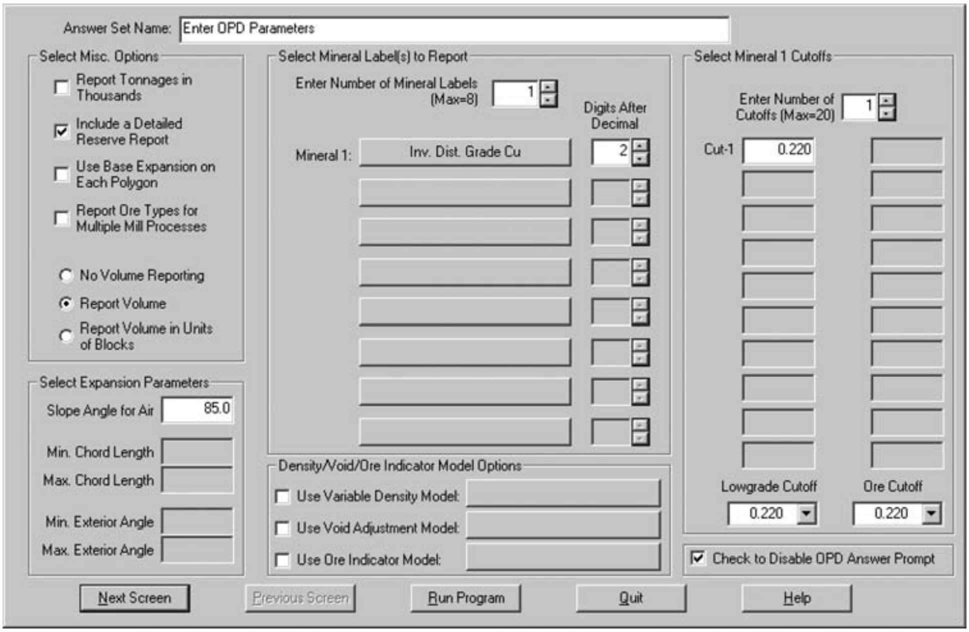


Figure 16.61. Enter Pit Generation Parameters – Input Screen One.

"INTRCK.PRN" is highlighted. Click "Open". If everything went correctly, you will get the message "Program Completed Normally. Codes Found in Model:1".

In the "Pits" module click on line "4 Enter Pit Generation Parameters". Click "OK". Click "Continue". You will get the screen entitled "Choose OPD Parameters, Models and Cutoffs" shown in Figure 16.61.

Change the input "Mineral 1: Inv. Dist. Grade Cu" to 2 "Digits After Decimal". Set the "Number of Cutoffs" to 1, and "Cut-1" to 0.220, which is our breakeven copper cutoff. Be sure the remaining settings match those in Figure 16.61 including the check in the box "Check to Disable OPD Answer Prompt".

Click "Next Screen". It is entitled "Enter Rock Density Information". The rock density is 12.5 ft^3/ton. This is okay. Click "Next Screen". It is entitled "Enter Rock Slopes in Angle From Horizontal". It is indicated as 45 degrees. Leave this alone. Click "Next Screen". Click "Run Program".

Before we generate an economic model called a Money Matrix, we must first create a slot for it in our grade modeling labels. Currently, there is only the label for copper, Cu.

Return to the "Data Entry" menu and choose "3 Enter Project Information". Click "OK". Change the "Number of Grade Labels" to 2. On line number two, under "Label" enter "$3_Cu" and under "Additional Description" enter "Money Matrix for $3.00 copper" as shown in Figure 16.62.

Then click "Run Program".

Return to the "Pits" menu item and select choice "11 Money Matrix/Cone Miner (Menu)". Click "OK". You will get the "Cone Mining Sub-Menu. Select choice "3 Create Money Matrix". Click "OK". You will get the menu "Choose Money Matrix Calculation Options"

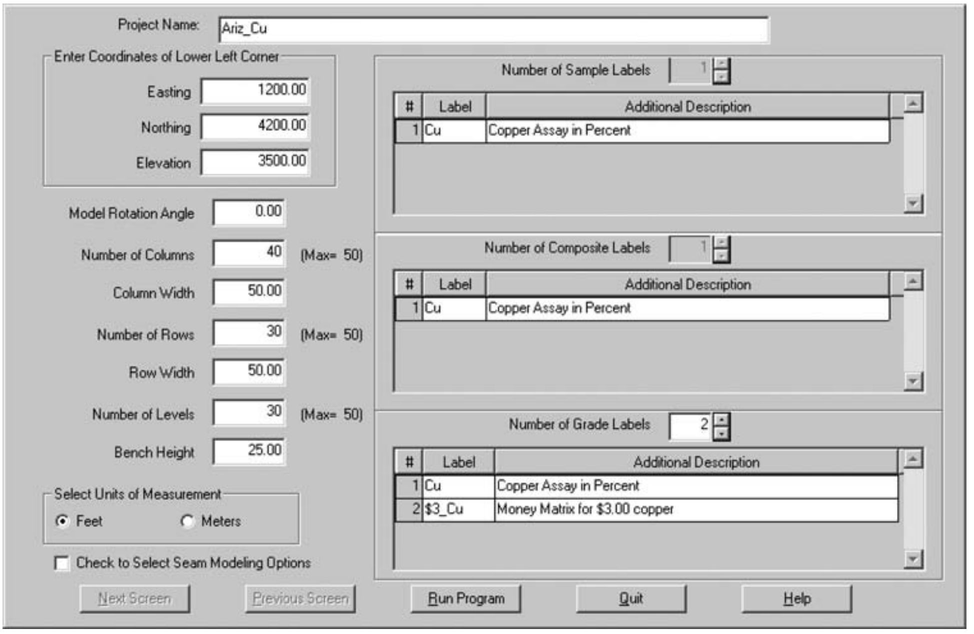


Figure 16.62. Enter Project Information – Add New Grade Label for Money Matrix.

dialog. You are to "Create Money Matrix for Floating Cone Analysis". The screen is shown in Figure 16.63.

You must now fill in the costs and price to be used in the calculation. Use the following values:

– mining cost = $2.50/ton
– haulage cost = $0.0/ton
– milling cost = $8.00/ton
– recovery factor = 80.0 and constant tail 0.0
– value for all rocks = $60/percent Cu/ton
– SRF (smelting, refining, freight) costs = $0/percent/ton

Click "Run Program". The screen will show "Select Print File to Display" with "MONEY.PRN" highlighted. Click "Open". A summary of the price and cost values to be used is presented. The money matrix is simply a 3-D representation of the net profit (or loss) that we would get if we mined a particular block of material in our 3-D model. Blocks that have copper grade below the specified cutoff of 0.22% are treated as waste, while blocks greater than or equal to 0.22% Cu are treated as ore.

Return to the "Cone Mining Sub-Menu". Select choice "4 Floating Cone Pit Design". Click "OK". Click "Continue". Set the values in the dialog to the following choices:

– minimum net value for mining = $1.00
– minimum mining radius = 25
– minimum level to mine = 1 3500.0 el
– number of scouring runs = 5

Your screen should match Figure 16.64.

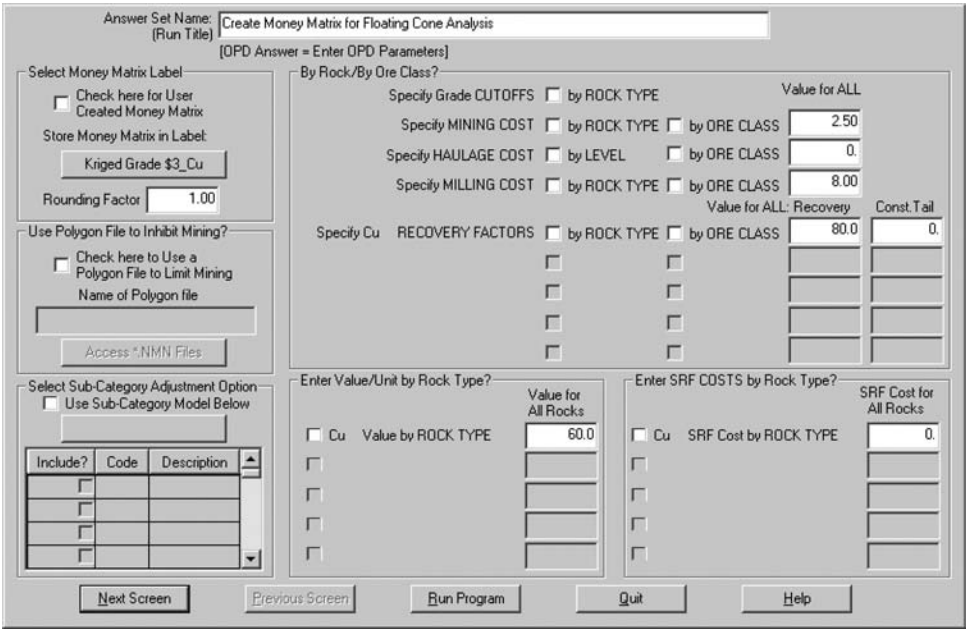


Figure 16.63. Create Money Matrix – Input Screen One.

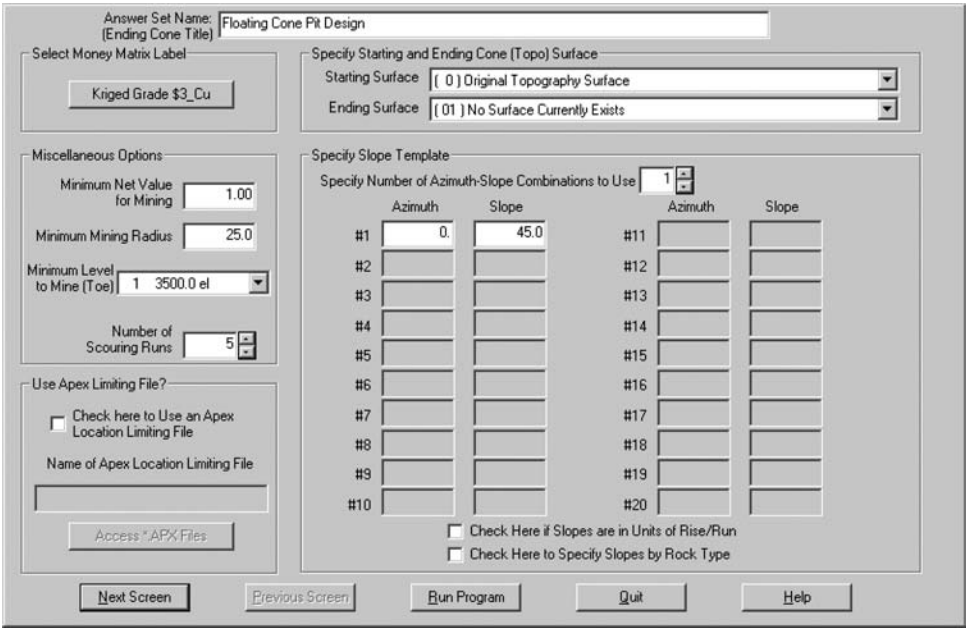


Figure 16.64. Floating Cone Pit Design – Input Screen One.

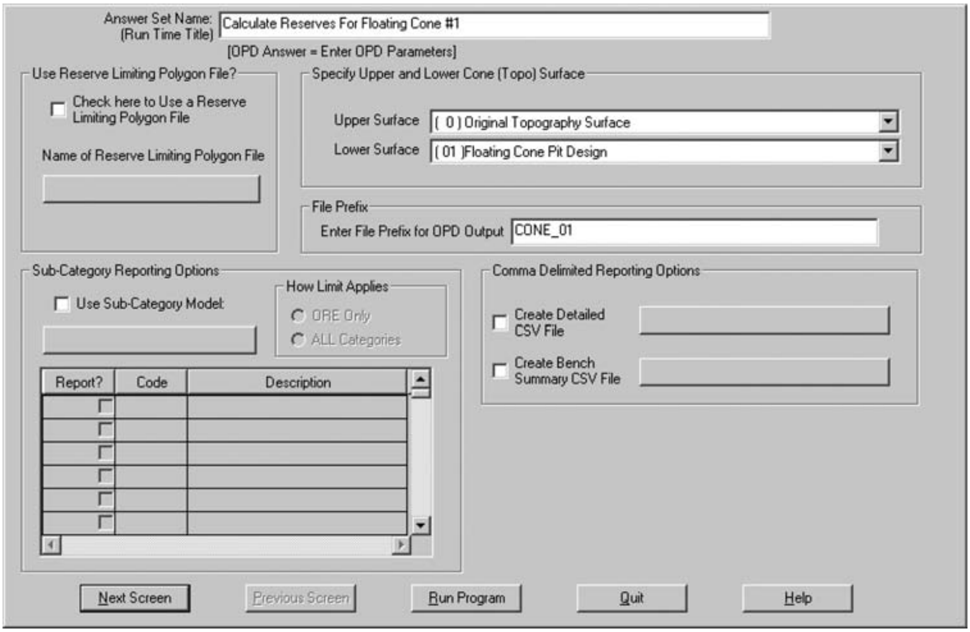


Figure 16.65. Calculate Cone Reserves – Input Screen One.

Click “Next Screen”. Click “Run Program”. The screen “Select Print File to Display” with “CONE.PRN” highlighted will appear. Click “Open”. A summary of the cone miner geometry and instructions will appear.

Return to the “Cone Mining Sub-Menu”. Select choice “5 Calculate Cone Reserves”. Click “OK”. Click “Continue” if it is requesting “Answer Set Name”. The screen shown in Figure 16.65 will appear. You will “Calculate the Reserves for Floating Cone #1”. Use the settings shown.

Click “Run Program”. The screen entitled “Select Print File to Display” with the file “OPDABR” highlighted will appear. Click “Open”. You will get a detailed summary of the reserves in the pit by level. The cone “mines” just over 1.4 million tons of ore at an average grade of 0.58% Cu at a stripping ratio of 0.80 Waste:Ore.

Return to the “Cone Mining Sub-Menu” and click on the line “7 Contour Plot of Cone Pit”. Click “OK”. Click “Continue”. Click “Next Screen”. Click “Next Screen”. Click “Run Program”. Click “Continue”. Click “Run Program”. The screen “Select Plot File to Display on Screen” will appear with the file “CONT” highlighted. Click “Open”. Click “OK”. You will get the contour map shown in Figure 16.66.

Return to the “Cone Mining Sub-Menu” and click on line “13 Plot Cone Profiles in Section”. Click “OK”. Click “Continue”. You will get Figure 16.67.

Under the heading “Enter Coordinates of Section Endpoints and Elevation Range” enter the coordinates required to plot section 2000E. Use a section tolerance of 25 ft.

Click “Next Screen”. You will get a screen entitled “Display Cone Pit Sections – Miscellaneous Plot Options” shown in Figure 16.68.

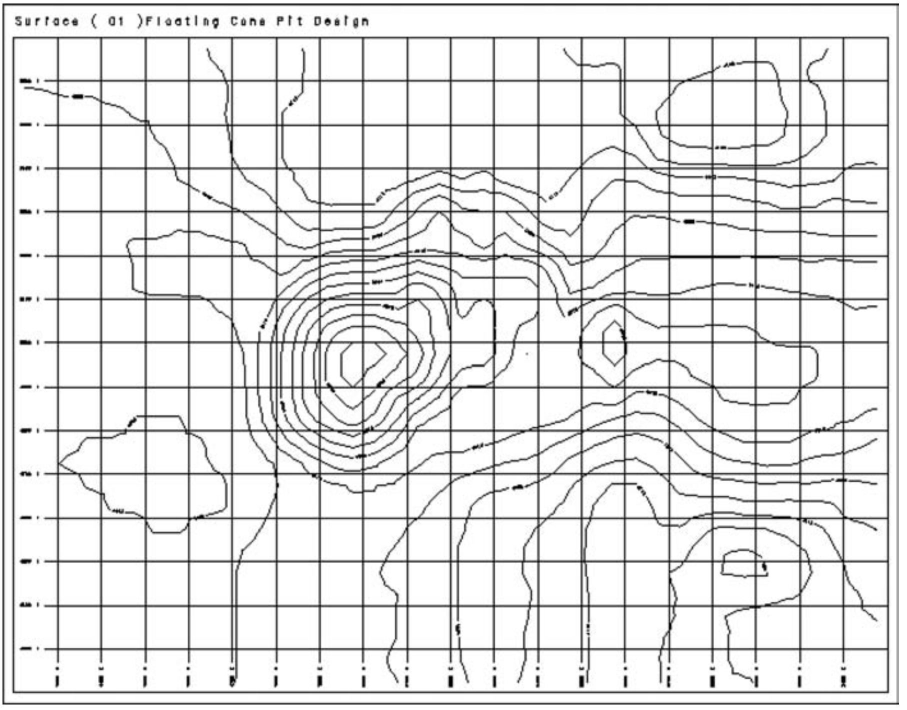


Figure 16.66. Arizona Copper – Plan View of Floating Cone Contours.

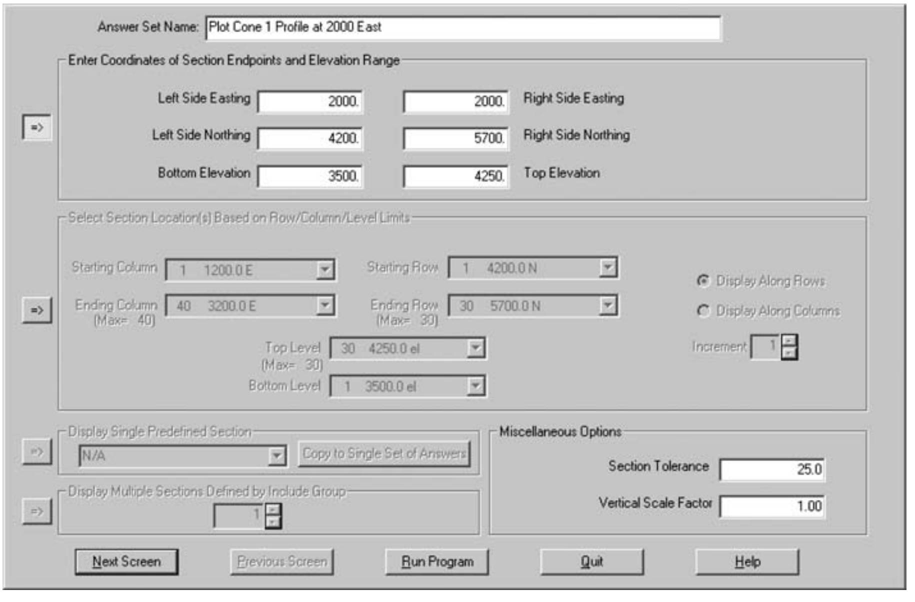


Figure 16.67. Plot Cone Profiles in Section – Input Screen One.

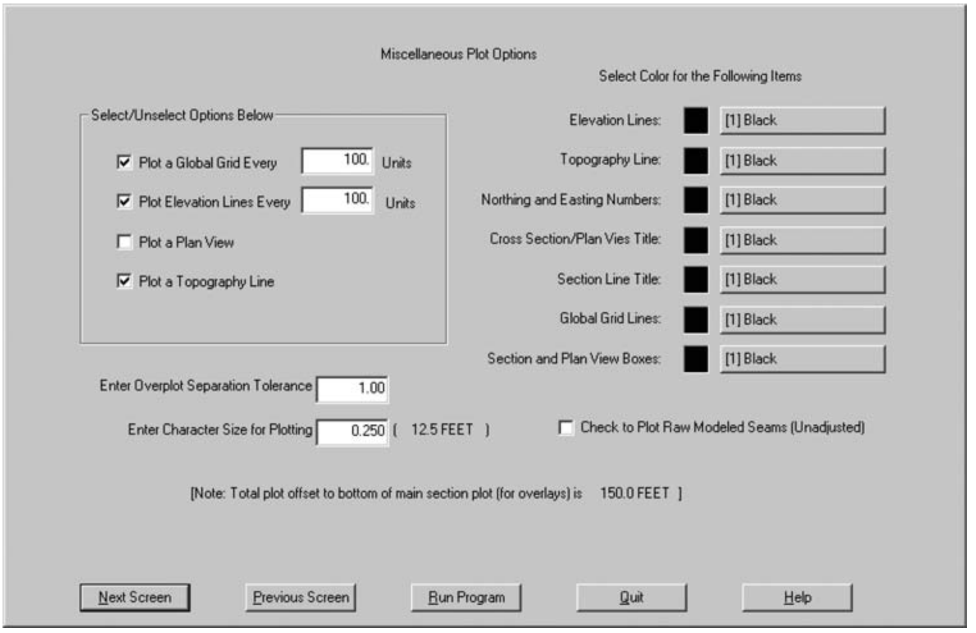


Figure 16.68. Plot Cone Profiles in Section – Input Screen Two.

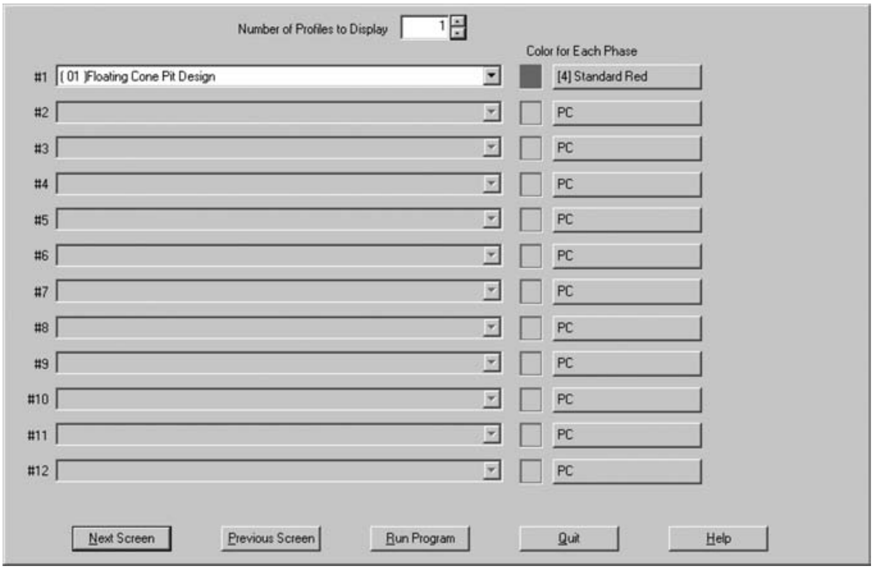


Figure 16.69. Plot Cone Profiles in Section – Input Screen Three.

Under “Select/Unselect Options Below”, check “Plot a Global Grid Every 100 Units”, “Plot Elevation Lines Every 100 Units”, and “Plot a Topography Line”.

Click “Next Screen”. The screen is entitled “Display Cone Pit Sections – Select Surfaces for Profile Plotting” shown in Figure 16.69 will appear.

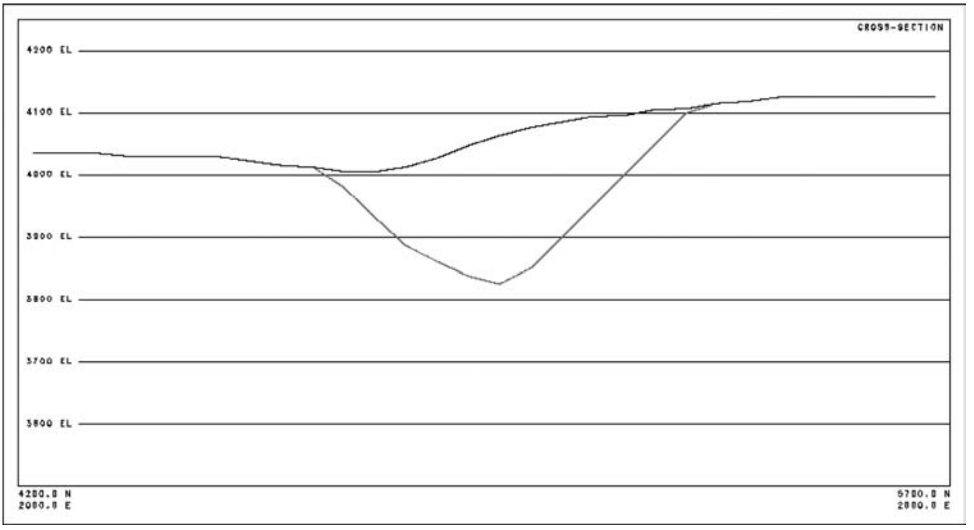


Figure 16.70. Arizona Copper – Section Plot of Cone Pit.

The "Number of Profiles to Display" should be set to 1. Choose "(01) Floating Cone Pit Design" and Color number "[4] Standard Red".

Click "Next Screen". Click "Run Program". Click "Continue". Since we are not worrying about the title block, click "Run Program". You will get the screen "Select Plot File to Display". With the file name "CONSCT.PLT" highlighted. Click "Open". Click "OK". You will get the cross-section shown in Figure 16.70.

16.4.7 *File manager*

The menu for the "File Manager" module is shown in Figure 16.71. We will use one of the File Manager choices to combine our cone profile with our composite Drill hole section plot.

Select choice "6 Create a Combined Plot". Click "OK". Click "Continue" if the "Choose Answer Set" appears.

In the first dialog shown in Figure 16.72, enter the responses shown.

Click "Run Program". You will get the screen "Select Plot File to Display". With the file name "MRGPLT.PLT" highlighted. Click "Open". Click "OK". You will get the combined cross-section plot shown in Figure 16.73.

As can be seen, the cone miner is going after the higher grade ore (red), as expected.

16.4.8 *Happy times*

Congratulations are in order!! You have succeeded in going from the input data to a final pit. But you have probably not fully understood either the process or how the different parameters have been chosen. Now begins the interesting part of the learning journey – understanding how to get from here to there on your own. You are encouraged to use the Ariz_Cu tutorial data set once again beginning with the "New Project" step. Consult the "User Manual" and also this "Tutorial" to help you progress along the path. When you

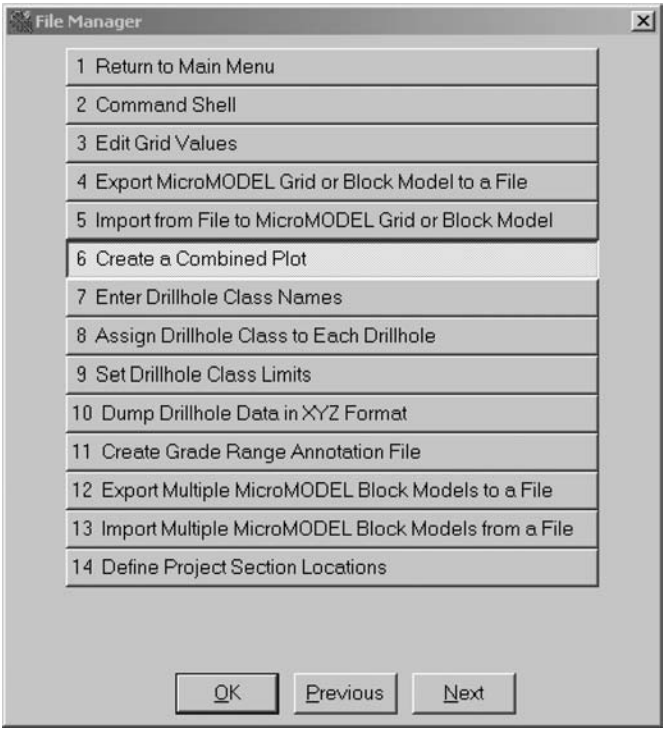


Figure 16.71. File Manager Submenu.

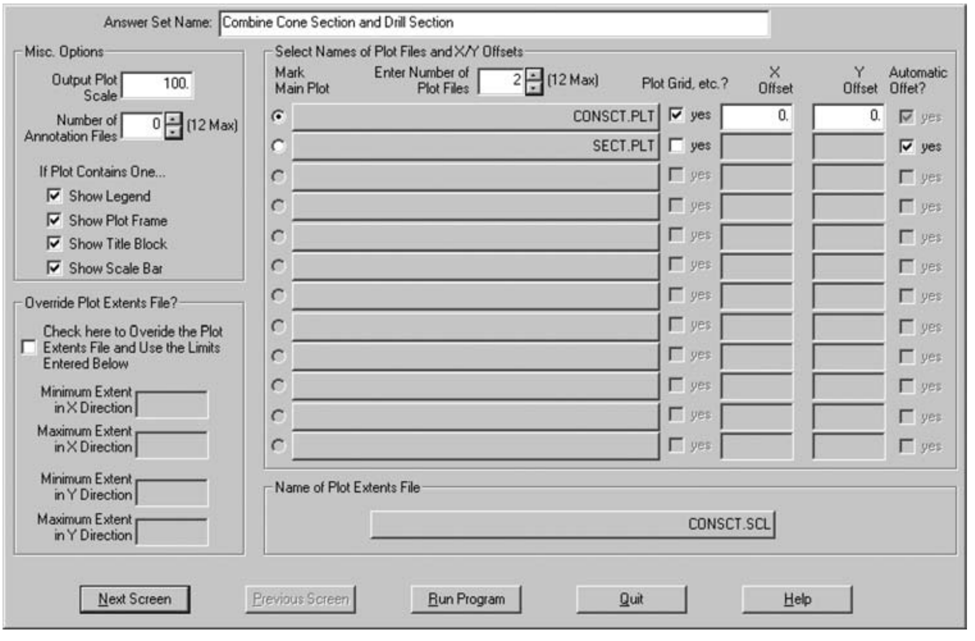


Figure 16.72. Create a Combined Plot – Input Screen One.

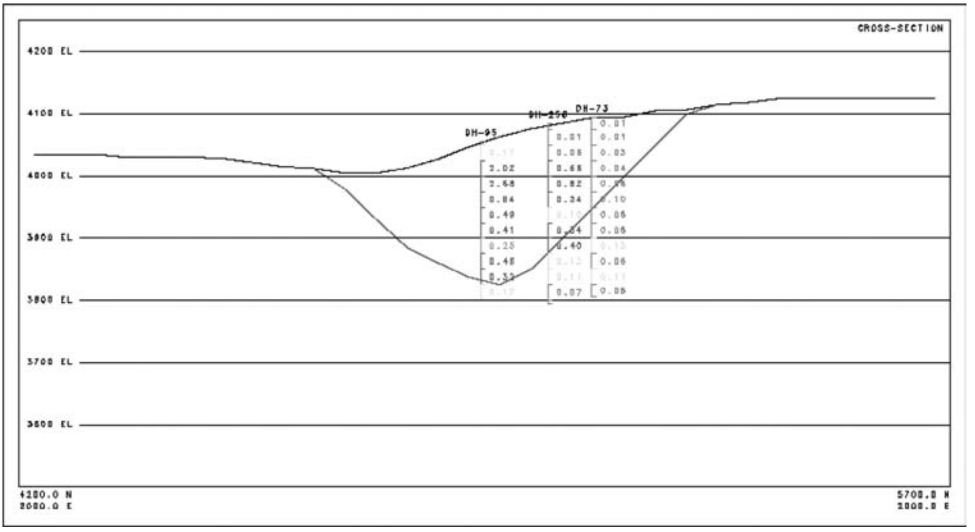


Figure 16.73. Arizona Copper – Combined Plot of Cone and Drill Holes in Section.

have mastered this, you are ready to take the plunge and try one of the other five data sets included with MicroMODEL.

16.5 OTHER DATA SETS – CONTINUATION

The MicroMODEL installation program creates six different project folders that contain data sets that can be analyzed using the Academic Demo license. As indicated earlier, the demo license will work for these projects as long as:

- The model origin
- The block size
- The number of rows, columns, and levels
- The number of drill holes
- The number of sample intervals

are not changed. The Ariz_Cu (Arizona Copper) project has already been discussed in detail. A description of the property is included in Chapter 17, section 17.2.

The remaining projects are as follows:

- Andina_Cu: An open pit copper deposit from central Chile. This property continues to be mined. There are 76 Drill holes, and 2099 assay intervals in the drill hole data file. Drill data have already been loaded into MicroMODEL, and a topo surface grid (T200) has been modeled. A description of the property is included in Chapter 17, section 17.8.
- Azul: An open pit gold deposit from central Chile. This project has already been mined. Drill hole data have already been loaded. There are 489 Drill holes and 33,878 assay intervals for Au ppm. A topo surface has been modeled, as well as several different types of grade model (nearest neighbor, inverse distance, and kriging).
- MMdemo: An open pit gold prospect in Nevada which has yet to be mined. Drill hole data have been loaded. There are 58 Drill holes and 2,397 assay intervals for Au in

Table 16.3. Demo project model parameters.

Project	Origin East	Origin North	Origin Elev	Ncol	Nrow	Nlev	Column Size	Row Size	Level Size	Number DH	Number Samples
MMDemo	3500	4300	2800	80	68	66	25	25	15	58	2397
Seam Demo	0	0	5900	57	53	30	100	100	10	17	51
Azul	69300	52140	3900	115	83	70	20	20	10	489	33878
Andina	4000	4300	2500	130	170	70	10	10	10	76	2099
Ariz_Cu	1200	4200	3500	40	30	30	50	50	25	40	2017
Norte_Cu	600	0	2000	210	160	103	10	10	10	144	22869

troy oz per ton. A topo surface has been created. Digitized ore zones, grade models, money matrix files, and floating cone pit designs are also included. A description of the property is included in Chapter 17, section 17.7.

- Norte_Cu: An open pit copper deposit from northern Chile. This project continues to be mined. Drill hole data have been loaded. There are 144 Drill holes, containing 22,869 sample intervals for total copper and sulfide copper. A topo surface grid has been modeled. A description of the property is included in Chapter 17, section 17.9.
- SeamDemo: A thermal coal deposit in New Mexico. This is the only demo project which is setup as a seam model rather than a 3-D model. There are 17 boreholes containing 51 seam assays for BTU and percent sulfur. Topo surface, seam surface, and quality models have all been created. There is also a money matrix model and floating cone design already in place.

The constraints listed in Table 16.3 apply to these projects.

Good Luck!

CHAPTER 17

Orebody case examples

17.1 INTRODUCTION

In most mining engineering curriculums, there is a capstone course called 'Senior Design.' During their academic careers, the students have taken a number of courses on individual topics in mining engineering such as surface mining, rock mechanics, materials handling, drilling and blasting, mine finance, mineral economics, etc. In this capstone course the students will be drawing upon this background and complete a near pre-feasibility level evaluation of some mining property. Beginning with the location information, geology and drill hole data for a given prospect, the student calculates the resource base, estimates the costs and recoveries, selects a price, determines the reserves, selects a production rate and a corresponding equipment fleet, develops a mine plan, calculates the economic indicators and writes a comprehensive engineering report. Unfortunately, it is often difficult for the students to procure the necessary drill hole data for carrying out the project, and hence quite a lot of time is spent simply spinning wheels. In the way of a response, in this chapter, eight drill hole data sets have been included on the distribution disk:

1. The Arizona Copper property
2. The Minnesota Natural Iron property
3. The Utah Iron property
4. The Minnesota Taconite property
5. The Kennecott Barneys Canyon Gold property
6. The Newmont Gold property
7. The Codelco Andina Copper property
8. The Codelco Norte Copper property

All of the data sets are of a size that can be run using the CSMine software. The Arizona Copper property data have been used in the CSMine tutorial.

In this section, some actual background information has been included for the different properties as well as some complementing information taken from various other operations. In some cases, the data have been adjusted to facilitate analysis. No attempt has been made to provide complete information since the students are expected to be able to do some work on their own.

One of the objectives of the senior thesis is to get the students familiar with the mining literature. Keeping up-to-date on mining developments around the world is important to a modern mining enterprise. In many cases, commodity prices are established on a worldwide basis and an individual mining operation competes by being cost competitive. This generally means using the highest level of suitable technology. The developments in mining technology, the arrival of new mines, the demise of old mines, operating tips, general mining news, new regulations, etc. have been and are the subject of various types of mining publications. Unfortunately, the indexing of mining articles is still poorly done due to the fact that mining is a relatively very small field. One often has to work very hard to find pertinent articles on a particular subject. The Internet is a popular way of finding and distributing information today. Some journals are now available on the Internet. However, older journals/publications containing much valuable information are not. Listed below are few of the journals that might be consulted by the student during the process of collecting information:

- Coal Age
- Canadian Mining Journal
- CSM Quarterly
- CIM Bulletin
- Australasian IMM Transactions
- Engineering and Mining Journal (E/MJ)
- Australian Mining Journal
- AusIMM Bulletin
- Pit and Quarry
- Transactions IMM
- Mining Congress Journal
- Mining Engineering (SME)
- Mining Engineer (UK)
- Int. Journal of Surface Mining and Reclamation
- Int. Journal of Mining Engineering
- South African Institute of Mining and Metallurgy (SAIMM) Journal
- Northern Miner
- Gluckauf
- World Mining Equipment
- Soviet Mining Science
- International Journal of Rock Mechanics and Mining Science
- Fragblast Journal
- Rock Mechanics
- Skillings Mining Review
- Explosives Engineer
- World Mining Equipment
- Mining Journal (London)
- Mining and Scientific Press

Some important books and publications are:

- Peele's Mining Engineers' Handbook
- USBM Information Circulars
- USBM Report of Investigations
- USBM Bulletins
- CANMET Publications

- USGS Publications
- State Geological Survey Publications (Utah Geological Survey, for example)
- Proceedings, U.S. Rock Mechanics Symposia
- Proceedings, International Rock Mechanics Symposia
- Proceedings, International Symposia on Computer Applications in Mining (APCOM)
- Transactions AIME
- USBM Index to Publications
- USGS Index to Publications
- KWIC Rock Mechanics Index
- Engineering Index
- Proceedings, U.S. Mine Health and Safety Institute
- Various books published by the SME, CIM and SAIMM.

At this stage in their careers, the student engineer should have already started a collection of important mining articles especially those containing technical and cost information.

By including these data sets and brief property descriptions, it is expected that both mining students and the professors of those students will have an enhanced senior design experience and this will be demonstrated in the final results.

17.2 THE ARIZONA COPPER PROPERTY

17.2.1 *Introduction*

The Arizona Copper property is a small copper prospect located within the Globe-Miami District of Arizona (Figure 17.1). The drill hole information included on the disk and the topography map have been graciously provided by a large U.S. copper mining company. The true location of this prospect is not of importance and the authors have moved it to the vicinity of the old Copper Cities Mine in order to provide some realistic geological information from published sources (Hardwick & Stover, 1960; Peterson, 1954). In this fictitious account, the names of the structural features have been retained because they reflect something of the early Western culture. However, the geologic maps provided are rotated from the true positions to fit more-or-less the given property topography. The major purpose of the data sets contained on the distribution disk is to provide the user of this book with the chance to explore the principles presented. A detailed description of the ArizCu data set is given in this tutorial. Descriptions of the additional data sets are provided in text files on the distribution disk.

17.2.2 *Historical background*

The Globe-Miami District was prospected and mapped in the early 1900's. The first serious attempt to explore the Arizona Copper property was begun in about 1920. A shaft was sunk and a number of drill holes bored. Although disseminated copper minerals were found, the grade was too low to be considered as ore. Further exploration work was conducted in 1930. With the projected copper price of $2.50 lb and the technological advances made over the years, the Arizona Copper prospect is once again of interest.

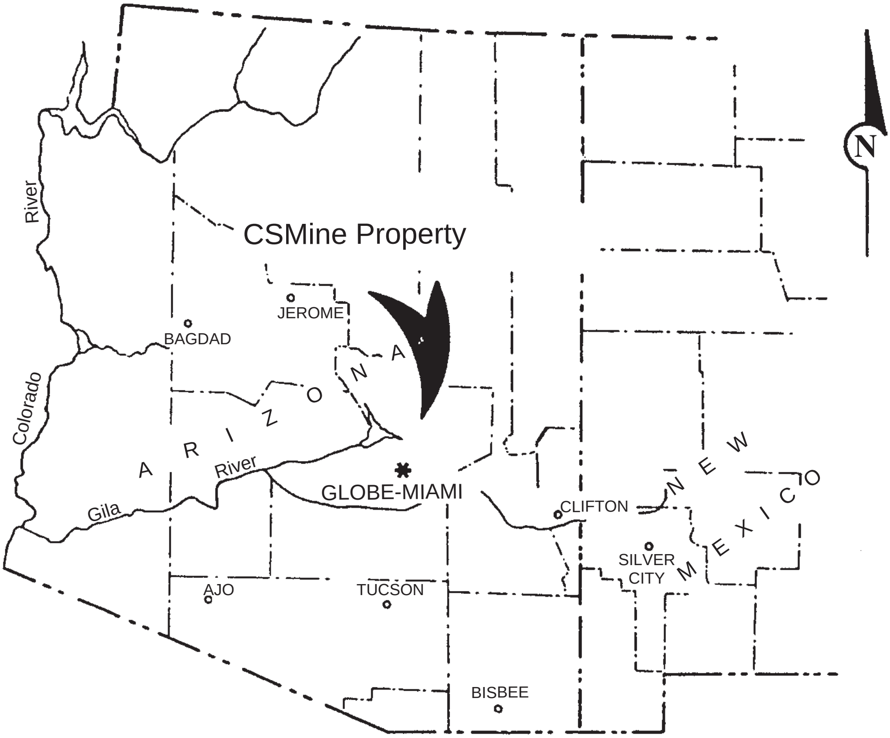


Figure 17.1. Location of the Arizona Copper property.

17.2.3 *Property topography*

The property is located approximately 3.5 miles due north of Globe, Arizona, off of State Highway 77. The topography map for the property is included as Figure 17.2 through Figure 17.5. The readers are encouraged to copy and enlarge these pages to the desired scale and then assemble them to create the overall map.

17.2.4 *Geologic description*

The property is located on the south flank of Sleeping Beauty Mountain (Figure 17.6). The Lost Gulch quartz monzonite is a relatively elevated northeast trending block that is bounded on three sides by faults. On the east side, the boundary is made by the east-dipping Miami fault, on the northeast by the Ben Hur fault, and on the northwest by the Sleeping Beauty fault.

Along the south side the quartz monzonite is in normal intrusive contact with Pinal schist and the various dioritic intrusive rocks of the Precambrian igneous complex.

The future ore zone is located within an area bounded approximately by the Drummond, Coronado and Sleeping Beauty faults.

The mine area straddles the upper reaches of Tinhorn Wash, a tributary of Pinal Creek, two miles to the east. The topography and drainage pattern is governed to a large extent by

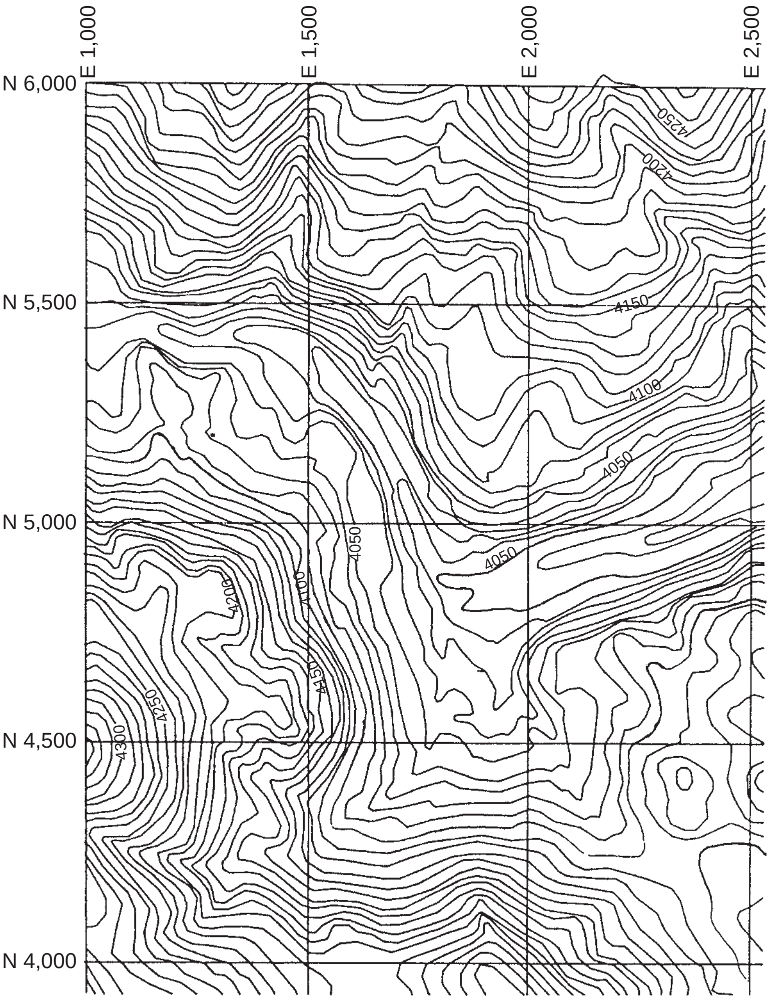


Figure 17.2. Northwest corner of the Arizona Copper topography map.

the relatively resistant outcrops of granite. The maximum relief is about 300 feet. Elevations range from 4250 feet on the top of the highest hill in the northwestern part of the mine area to 3950 feet in the bed of Tinhorn Wash where it crosses the southeastern limits of the mine.

17.2.5 *Mineralization*

The mineralogy of the Arizona Copper deposit is fairly simple and is in most respects like that of the other disseminated deposits of the district.

The orebody is covered by a weathered outcrop of quartz monzonite and granite porphyry, having the characteristic of leached capping. The leached capping ranges in thickness from 20 to 115 feet and has a copper content of generally less than 0.1 percent.

The thickest part of the orebody and that with the highest grade is located in the quartz monzonite. The thinnest is located in granite porphyry.

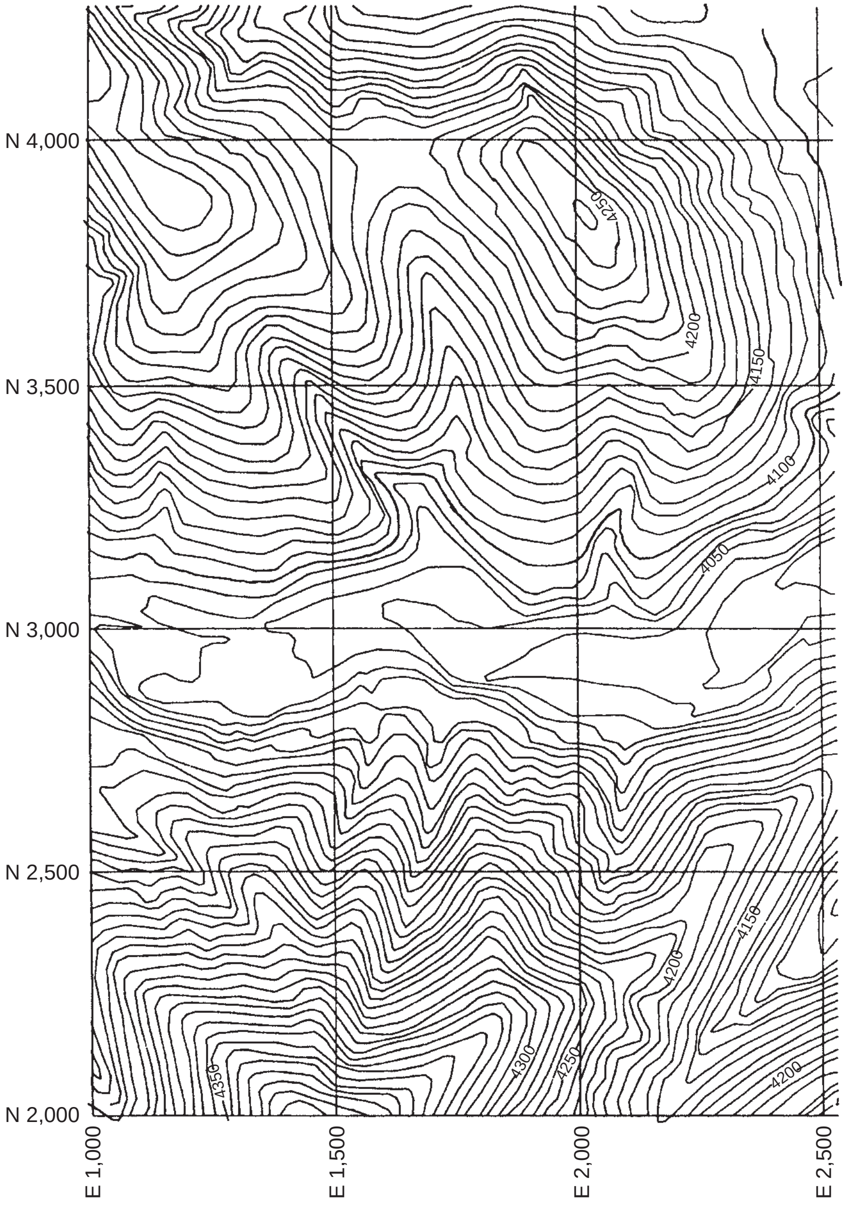


Figure 17.3. Southwest corner of the Arizona Copper topography map.

The principal hypogene minerals are quartz, pyrite, chalcopyrite, and molybdenite. Chalcocite and covellite are the only supergene sulfide minerals in the deposit. Malachite, azurite and turquoise account for most of the acid soluble copper in the orebody, as well as for the small amount of copper that remains in the leached capping. The molybdenum content of the rock within the limits of the orebody has been estimated at about 0.004 percent. The

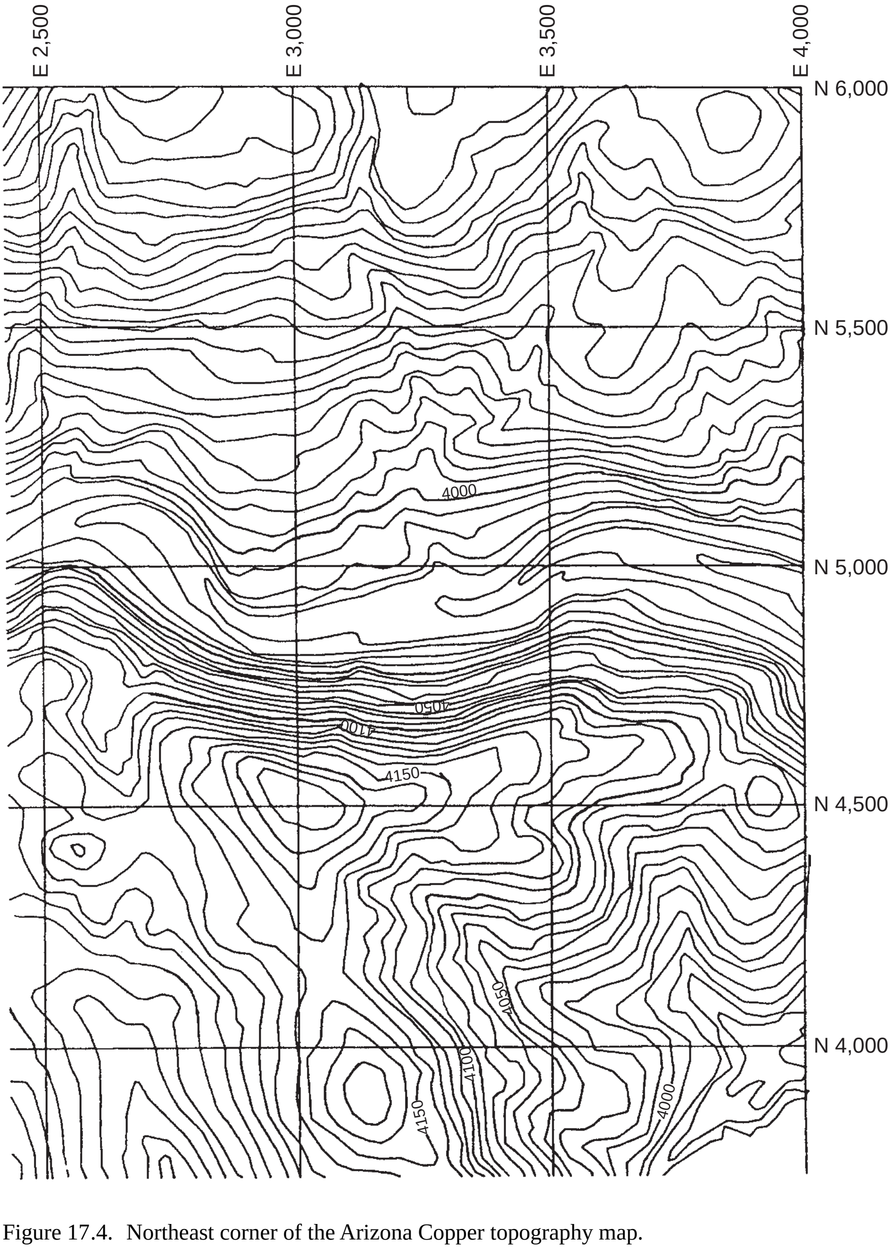


Figure 17.4. Northeast corner of the Arizona Copper topography map.

mineralized quartz monzonite is intricately dissected by joints, fractures, and minor faults, some older and some younger than the mineralization.

The pre-mineral fractures are now occupied by quartz-pyrite and chalcopyrite veinlets. In many places, it is difficult to find a hand-sized fragment of rock that is not bounded on one or more sides by walls of veinlets. Post-mineral fractures and minor faults are abundant. In the deposit, there is no clear evidence of lateral zoning. There is however strong evidence

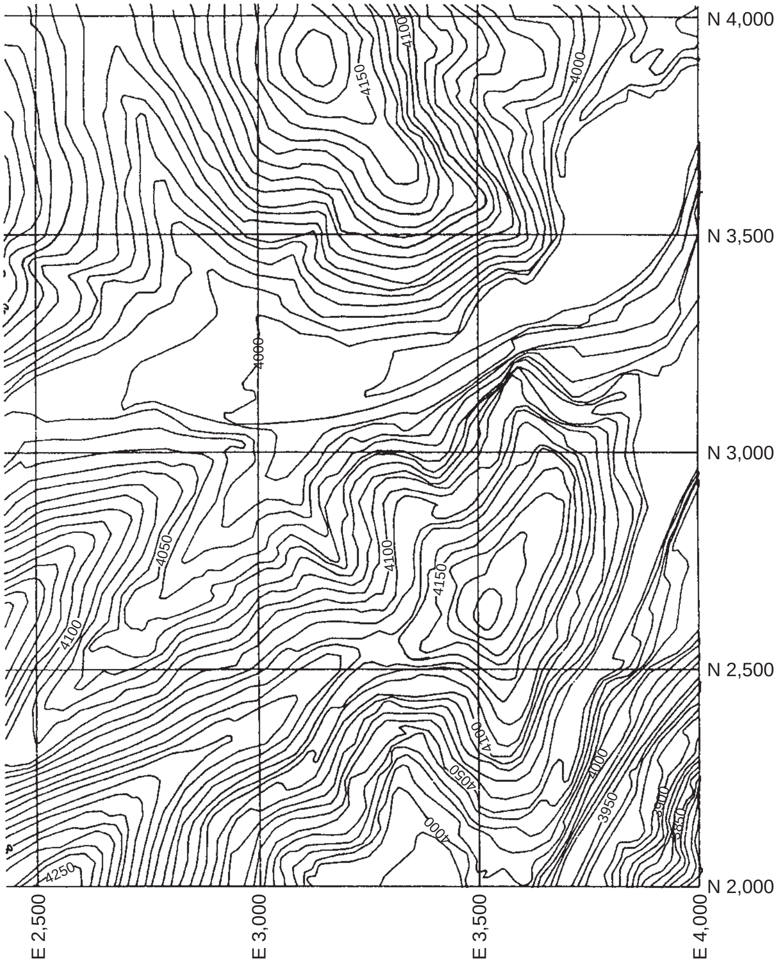


Figure 17.5. Southeast corner of the Arizona Copper topography map.

of vertical zoning. Practically all the exploratory drill holes show a definite decrease in the copper content of the rock with increasing depth below the bottom of the leached capping. The drill logs record surprisingly few irregularities in the assay patterns that can clearly be attributed to structural features. There has been little recognizable deep leaching or oxidation along faults cut by drill holes.

17.2.6 *Drill hole data*

There have been 40 diamond drill holes (DDH) of various depths drilled and logged on this property.

The basic drill hole data are of the form illustrated in Table 17.1 for DDH 49. Table 17.2 is an explanation of the different portions of this actual record. The simplified form of this

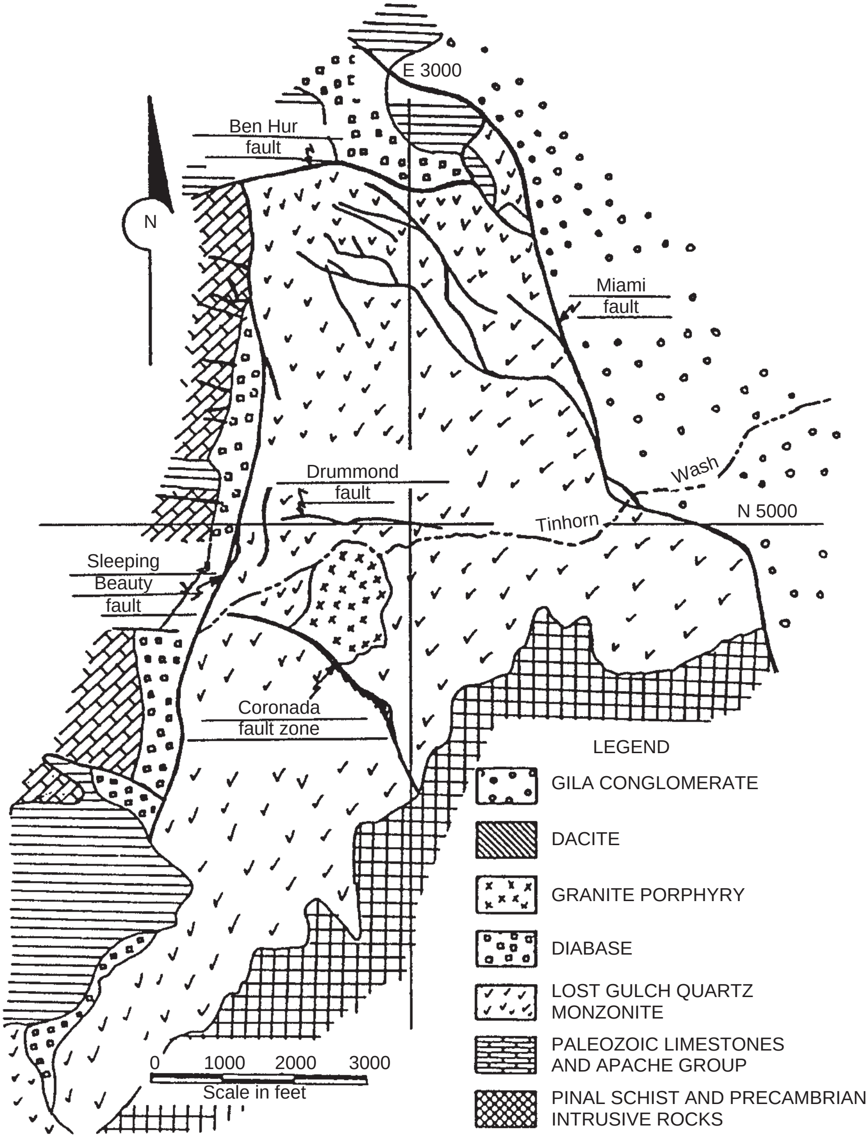


Figure 17.6. Geologic map of the Arizona Copper property.

same record used in the CSMine program is shown in Table 17.3. As can be seen only the total copper column has been retained for use in the evaluation.

17.2.7 *Mining considerations*

Rock cuts excavated just east of the Coronado fault show a prominent system of closely spaced fractures and veinlets that strike N 15° W to N 25° E and dip steeply NE. There are

Table 17.1. Typical computerized form of a drill hole log (hole DDH 49).

8	49	5042	2582	3984	5042	0	0	−90	0	145	145
C		68		49		2582	3984			−90	
8	49	0	8	−0.990	−0.990	−0.99043	099	0	0	0	
8	49	8	10	0.120	0.100	−0.99043	0 4	0	0	0	
8	49	10	13	0.350	0.280	−0.99043	0 4	8	0	0	
8	49	13	18	0.330	0.280	−0.99043	6299	8	0	0	
8	49	18	23	0.390	0.250	−0.99043	6299	0	0	0	
8	49	23	28	0.520	0.350	−0.99043	099	0	0	0	
8	49	28	33	0.270	0.130	−0.99043	0 4	0	0	0	
8	49	33	38	0.460	0.280	−0.99043	0 4	8	0	0	
8	49	38	43	0.160	0.090	−0.99043	099	8	0	0	
8	49	43	48	0.100	−0.990	−0.99043	099	0	0	0	
8	49	48	52	0.180	0.090	−0.99043	0 4	0	0	0	
8	49	52	57	0.240	0.100	−0.99043	0 4	8	0	0	
8	49	57	62	0.220	0.070	−0.99043	099	0	0	0	
8	49	62	67	0.180	0.040	−0.99043	6099	0	0	0	
8	49	67	70	0.050	−0.990	−0.99043	099	0	0	0	
8	49	70	75	0.040	−0.990	−0.99043	033	0	0	0	
8	49	75	80	0.050	−0.990	−0.99043	099	0	0	0	
8	49	80	85	0.030	−0.990	−0.99043	099	0	0	0	
8	49	85	90	0.100	−0.990	−0.99043	6299	0	0	0	
8	49	90	99	0.040	0.020	−0.99043	6299	0	0	0	
8	49	99	104	0.030	0.030	−0.99043	099	0	0	0	
8	49	104	109	0.040	0.005	−0.99043	6099	0	0	0	
8	49	109	114	0.020	0.005	−0.99043	099	0	0	0	
8	49	114	119	0.020	0.020	−0.99043	099	0	0	0	
8	49	119	122	0.020	0.005	−0.99043	099	0	0	0	
8	49	122	126	0.020	0.005	−0.99043	099	0	0	0	
8	49	126	131	0.020	0.005	−0.99043	099	0	0	0	
8	49	131	136	0.020	0.005	−0.99043	099	0	0	0	
8	49	136	141	0.020	0.005	−0.99043	099	0	0	0	
8	49	141	145	0.090	0.020	−0.99043	099	0	0	0	

also many minor faults of similar strike and dip. These are considered representative of the overall pit area.

The initial geomechanic studies have suggested that an overall final slope angle of 45° can be maintained. This, incidentally, is the same slope angle successfully used at a neighboring property. The pit limits will be determined by assay boundaries rather than geological features. It is a semi-arid landscape, with about 20 inches of rainfall a year. This is relatively uniformly distributed over the months of July through March (Table 17.4). Violent thunderstorms occur in the late summer months July and August and provisions must be made in the design for this.

17.3 THE MINNESOTA NATURAL IRON PROPERTY

17.3.1 *Introduction*

The Minnesota Natural Iron property is located in the North range of the Cuyuna iron-ore district of Minnesota. The background information regarding the property has, for the most

Table 17.2. Description of the items included in Table 17.1.

Collar record (line 1)

8	49	5042	2582	3984	0 0	−90 0	145
Series No.	Hole No.	Northing	Easting	Elevation	Bearing	Dip	Total depth

Assay record heading (line 2)

C	68	49	5042	2582	3984	−90	145
Comment	Series No.	Hole No.	Northing	Easting	Elevation	Dip	Total depth

Assay records

8	49	0	8	−0.990	−0.990	−0.99043	099 0 0 0
8	49	8	10	0.120	0.100	−0.99043	0 4 8 0 0
8	49	10	13	0.350	0.280	−0.99043	0 4 8 0 0
8	49	13	18	0.330	0.280	−0.99043	6299 0 0 0
8	49	18	23	0.390	0.250	−0.99043	6299 0 0 0
8	49	23	28	0.520	0.350	−0.99043	099 0 0 0
Series No.	Hole No.	From depth	To depth	Total copper	Acid soluble copper	% Moly	Mineral data

−0.990 = No assay.

part been extracted from the publication ‘Geology and Ore Deposits of Cuyuna North Range, Minnesota’ by Robert Schmidt of the U.S. Geological Survey (Schmidt, 1963). The drillhole data, the hole location map, and the initial mine design have been generously provided by the Minnesota Department of Natural Resources.

As can be seen in Figure 17.7, this district lies near the geographic center of Minnesota. The mining property is located in Section 10, Township 46 N, Range 29 W of Crow Wing County. By way of explanation, each Section is divided into four quadrants and each quadrant is further divided into four sectors. The Minnesota Natural Iron deposit lies in the SW and SE sectors (the south half) of the NW quadrant and the NW and NE sectors (the north half) of the SW quadrant of Section 10. Hence, the nomenclature used to describe this property is S½ NW¼-N½ SW¼-10-46-29. Property ownership is sometimes separated into ‘surface rights’ and ‘mineral rights,’ either or both of which are owned or leased by the iron-mining companies, and there are many complex arrangements of leases and successive subleases. The many lakes in the district further complicate the ownership of mineral rights.

Table 17.3. Simplified form of the log used in CSMine (hole DDH 49).

0	1 0						
0	% Copper						
1	Dh 49	2582.00	5042.00	3984.00	0.00	−90.0	145.00
3	Dh 49	0.00	8.00	−0.99			
3	Dh 49	8.00	10.00	0.12			
3	Dh 49	10.00	13.00	0.35			
3	Dh 49	13.00	18.00	0.33			
3	Dh 49	18.00	23.00	0.39			
3	Dh 49	23.00	28.00	0.52			
3	Dh 49	28.00	33.00	0.27			
3	Dh 49	33.00	38.00	0.46			
3	Dh 49	38.00	43.00	0.16			
3	Dh 49	43.00	48.00	0.10			
3	Dh 49	48.00	52.00	0.18			
3	Dh 49	52.00	57.00	0.24			
3	Dh 49	57.00	62.00	0.22			
3	Dh 49	62.00	67.00	0.18			
3	Dh 49	67.00	70.00	0.05			
3	Dh 49	70.00	75.00	0.04			
3	Dh 49	75.00	80.00	0.05			
3	Dh 49	80.00	85.00	0.03			
3	Dh 49	85.00	90.00	0.10			
3	Dh 49	90.00	99.00	0.04			
3	Dh 49	99.00	104.00	0.03			
3	Dh 49	104.00	109.00	0.04			
3	Dh 49	109.00	114.00	0.02			
3	Dh 49	114.00	119.00	0.02			
3	Dh 49	119.00	122.00	0.02			
3	Dh 49	122.00	126.00	0.02			
3	Dh 49	126.00	131.00	0.02			
3	Dh 49	131.00	136.00	0.02			
3	Dh 49	136.00	141.00	0.02			
3	Dh 49	141.00	145.00	0.09			

Sub-surface proprietorship is bounded by planes extending vertically downward from the property lines (Schmidt, 1963).

Overall, the Cuyuna district is about 68 miles long. It extends from Randall in Morrison County northeastward to a point 11 miles east of the small community of Hassman in Aitkin County. Traditionally the district is divided into three general subdistricts or ranges: the South range, the North range, and Emily district. The Cuyuna South range is long, extending the full length of the district, but narrow. The width ranges from less than 1 mile to about 7 miles. The North range is about 12 miles long and 5 miles wide. It lies near the villages of Crosby and Ironton in Crow Wing County (Schmidt, 1963).

17.3.2 *Access*

The North range is served by good highways and railroads capable of handling the heavy freight load imposed by the shipping of the Cuyuna iron ores. The main highway, U.S. 210,

Table 17.4. Climatic data for Miami, Arizona; as observed over the reference period 1961–1990 (latitude: 33°24′ N, longitude: 11°53′ W, elevation: 3560 ft). (National Oceanic and Atmospheric Administration, 1992.)

Month	Temperature			Extreme temp		Precipitation			Heat days	Cool days
	Mean	Max	Min	Max	Min	Norm >0.01	Mean days >1.0	Mean snow/ice		
January	44.2	56.2	32.1	(1)	(1)	1.85	(1)	(1)	645	0
February	47.8	60.6	34.9	(1)	(1)	1.68	(1)	(1)	482	0
March	52.3	65.2	39.4	(1)	(1)	1.98	(1)	(1)	402	8
April	60.2	74.0	46.3	(1)	(1)	0.59	(1)	(1)	192	48
May	69.1	83.3	54.7	(1)	(1)	0.38	(1)	(1)	26	153
June	78.7	93.7	63.7	(1)	(1)	0.27	(1)	(1)	0	411
July	83.3	96.5	70.0	(1)	(1)	2.41	(1)	(1)	0	567
August	80.7	93.7	67.7	(1)	(1)	2.98	(1)	(1)	0	487
September	75.1	88.3	61.9	(1)	(1)	1.79	(1)	(1)	0	303
October	64.7	78.4	50.9	(1)	(1)	1.53	(1)	(1)	113	104
November	52.8	65.7	39.8	(1)	(1)	1.61	(1)	(1)	366	0
December	44.8	56.6	33.0	(1)	(1)	2.63	(1)	(1)	626	0
Full year	62.8	76.0	49.5	(1)	(1)	19.70	(1)	(1)	2852	2081

Various columns:
- 01 Normal monthly mean temperature (°F).
- 02 Normal monthly maximum temperature (°F).
- 03 Normal monthly minimum temperature (°F).
- 04 Extreme maximum temperature for month (°F).
  (1) = Data missing.
- 05 Extreme minimum temperature for month (°F).
  (1) = Data missing.
- 06 Normal monthly precipitation (inches).
- 07 Mean number days with precipitation of 0.01^{11} or more.
  (1) = Data missing.
- 08 Mean number days with snow/ice pellets of 1.0^{11} or more.
  (1) = Data missing.
- 09 Normal monthly heating degree days (base 65°F).
- 10 Normal monthly cooling degree days (base 65°F).

All temperatures are in degrees Fahrenheit. Data shown in columns 1 to 3, 6, 9 and 10 are based on average monthly observations for the standard normal reference period 1961–1990. Data shown in the other columns are means based on different (mostly longer than 1961–1990) averaging periods.

is a modern road extending generally east-west. From Crosby it leads 15 miles southwest to Brainerd, a city of 12,353. Eastward it passes through Deerwood and to Aitkin where it connects with U.S. 169, the main road linking this area with Duluth and to the Mesabi district. State Highway 6 extends northward from Crosby and southward from Deerwood. The ore from the district is transported by rail to Duluth for further shipment to eastern markets (Schmidt, 1963).

17.3.3 *Climatic conditions*

The average temperature conditions at Brainerd, Minnesota which is approximately 20 miles from the Minnesota Natural Iron property are provided in Table 17.5.

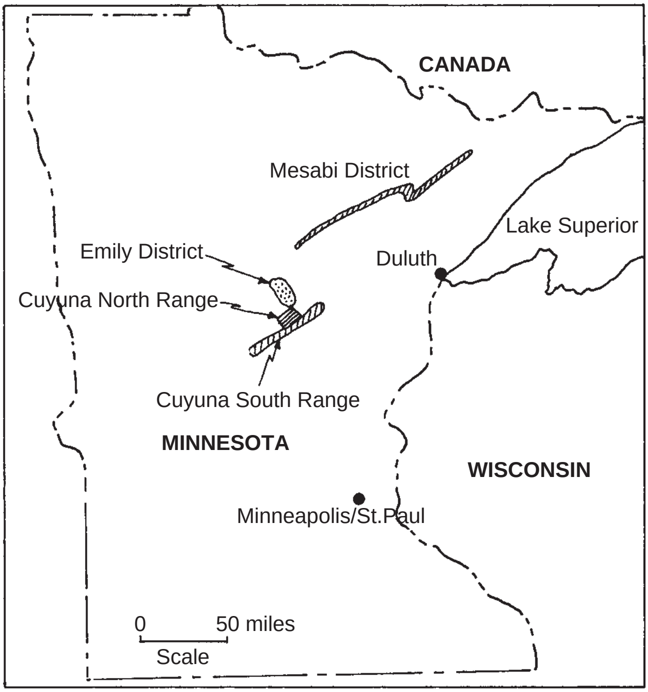


Figure 17.7. Map of Minnesota showing the location of the Cuyuna and Mesabi mining districts. After Schmidt (1963).

Table 17.5. Normal climate around Brainerd, Minnesota.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave. temp (deg. F)	5.5	12.5	25.6	41.7	55.5	64	68.6	66.1	55.6	43.8	27.6	12.6
Ave. high temp (deg. F)	18.5	26	37.5	54	68.6	76.5	81	78.8	68.8	56.2	37.4	23.2
Ave. low temp (deg. F)	−7.5	−1	13.6	29.3	42.3	51.4	56.2	53.4	42.4	31.4	17.7	1.9

http://mcc.sws.uiuc.edu/climate_midwest/historical/temp/mn/210939_tsum.html

17.3.4 *Historical background*

The Cuyuna district, its iron-formation hidden by glacial drift, was the last major iron-ore district discovered in the Lake Superior region. In 1859, land surveyors laying out section lines in the northern part of T. 45 N., R. 29 W., noted deflection of the magnetic compass and wrote that ‘there was very great local attraction caused by Bog Iron in the low marshes’ (Himrod, 1940).

The first merchantable ore was discovered on the South range in the SEl/4NE1/4Sec. 20, T. 46 N., R. 28 W., in May 1904 (Himrod, 1940). The drilling had been conducted by a partnership including Cuyler Adams, R. C. Jamieson, W. D. Edson, and others. Drilling was started on the North range in July 1904 in sec. 30, T. 47 N., R. 28 W (Himrod, 1940).

According to Himrod (1940), the name of the district was suggested by Mrs. Cuyler Adams from the first syllable of Cuyler and the remainder formed by the name of Adams’ dog, Una, who accompanied him on his explorations.

Throughout the years, the demand for the manganiferous iron ores produced on the range was cyclic. From the time when ore was discovered in 1904 until mining ceased in 1984, more than 106 million tons of ore were mined and shipped from the Cuyuna range. Now, however, with the currently high prices for iron ores, interest in the district has once again increased and the Minnesota Natural Iron property is being re-evaluated.

17.3.5 *Topography*

The Cuyuna district, unlike the other iron-bearing districts in the Lake Superior region, is a generally flat area covered almost entirely by a thick mantle of Pleistocene glacial drift. The topography of the district is dominated by features developed by the retreating continental glacial ice. It is a poorly drained morainic surface consisting of irregular hills and many lakes and swamps. Part of the area is a relatively level outwash plain. Natural elevations in the North range are from 1172 to l390 feet above sea level, but most of the bigger hills have less than 100 feet of relief. With regard to the mines in the district, it is important to note that the elevations are expressed relative to the elevation of Lake Superior which is 602 feet (Schmidt, 1963).

The area south of Crosby and Ironton is farmland and second-growth forest. Farming is less important to the north of these villages and much of the land is included within State forests. Timber products are an important source of income. Many of the lakes are surrounded by summer homes and cottages and services for tourists form a thriving industry in the area (Schmidt, 1963).

17.3.6 *General geologic setting*

The strata of the Cuyuna district are a tightly folded sequence of slightly metamorphosed Precambrian rocks consisting of argillite, slate, phylite, and iron-formations, and lesser amounts of coarser clastic rocks and volcanic tuff and flows. The trend of the Cuyuna folds is about N. 65°E. and parallels the axis of the Lake Superior synclinorium. The North range is comprised of three stratigraphic units of Precambrian age: the Trommald, Rabbit Lake and Mahnomen formations. These are assumed to rest unconformably on a surface of Algoman granite and various other older rocks. The lower unit of the three, the Mahnomen formation, consists of at least 2000 feet of light-colored argillite or slate and quartz siltstone and includes local quartzite lenses and layers near the top. The middle unit, the Trommald formation, is almost entirely an iron-rich chemical sediment known as iron-formation. It contains only local thin layers of clastic sediments. An iron-oxide-rich thick-bedded facies of this formation is confined to part of the North range. It overlaps an iron silicate-rich and iron carbonate-rich thin-bedded facies which occurs in part of the North range and probably all of the South range. The Trommald formation generally ranges in thickness from 45 to more than 500 feet. The uppermost unit, the Rabbit Lake formation, is argillite and slate, partly carbonaceous and ferruginous. Lenses of lean cherty iron-formation are distributed throughout. Local basaltic flows or sills and tuffs occur near the base. The Rabbit Lake formation is at least 2000 feet thick.

The sedimentary rocks in this synclinonum have been tightly folded, probably by forces acting from the southeast, and subjected to metamorphism that increased gradually toward the south or southeast. In the North range the strata were intruded by diorite or gabbro dikes during or after folding. These intrusive rocks were metamorphosed with the sedimentary rocks (Schmidt, 1963).

The bedrock throughout the area is covered by Pleistocene glacial drift ranging in depth from 15 to more than 200 feet.

Except for a few small ore bodies within iron-formation lenses in the Rabbit Lake formation, the iron ores have developed within the Trommald formation. The ores are residues, having formed by leaching of the original minerals of the iron-formation (quartz and one or more iron oxides, silicates, or carbonates). The ores, which were formed by the removal of carbon dioxide, silica, and magnesia, are porous and vuggy. They were probably formed in two stages. The first stage involved ground waters that possibly were partly hydrothermal which affected the iron-formation to great depths. In the second stage, the alteration was shallower and related to ordinary surface weathering. The time of the formation of the Cuyuna ores has not been established (Schmidt, 1963).

The kind and distribution of the ores differ considerably according to the areas within the district and according to the facies of the Trommald formation that is present. The most important ore bodies have formed along the southeast edge of the North range in the thin-bedded facies. Other important deposits have formed in the thick-bedded facies in the North range. A smaller number of minable ore bodies occur in the thin-bedded facies on the northwest side of the North range, in the South range, and in the iron-formation lenses of the Rabbit Lake formation (Schmidt, 1963).

17.3.7 *Mine-specific geology*

The Feigh mine neighbors the Minnesota Natural Iron property and therefore provides an excellent source of mine-specific geologic information. The material in this section describes the geology of the Feigh in which some of the most complex geology seen in the district is exposed.The general structure is believed to be an anticlinal fold with the axis overturned to the northwest. The fold axis appears to have been broken by a thrust fault of major proportions. The Trommald formation is exposed in a band as much as 700 feet wide that strikes about N. 65° E. and in general dips steeply to the southeast. Part of the iron-formation is highly distorted (Schmidt, 1963).

The Rabbit Lake formation is exposed on the northwest side of the Feigh mine. It is black argillite and slate dipping 70°–80° SE. The wallrock on the southeast side of the mine is also considered to be Rabbit Lake formation. It is a bedded chlorite schist, contains 1-inch thick layers of gray-green chert, and dips 40°–60° SE. At the northeast end of the mine, a large lens of schist strikes parallel to the length of the pit and dips steeply southeastward. Although it is tentatively considered an intrusive along the faulted crest of the large anticline, it may also be schist from the metamorphosed Rabbit Lake formation faulted into its present place. From drilling, the lens is known to increase in width for a considerable distance below the bottom of the mine.

The presence of the Mahnomen formation has also been disclosed within the heart of the anticline through the drilling of one deep hole.

There is abundant shearing and brecciation that may be evidence of faulting in a zone that lies nearly midway between the sides and parallels the length of the pit. The zone is aimed roughly with the lens of schist at the northeast end of the mine and with a severely folded belt in the South Hillcrest mine to the southwest. In the northeast end of the mine there are numerous thin layers or lenses of chlorite schist conformably inter-layered with the iron-formation. It is not clear whether these were argillaceous lenses in the original sediment, fault slivers of Rabbit Lake formation, or intensely sheared intrusives. Near the top of the Trommald formation at the southwest corner of the mine there are argillaceous lenses, a few feet thick, containing abundant tourmaline needles (Schmidt, 1963).

The iron-formation is largely oxidized, but there is abundant unoxidized material in the south-central part of the mine. Oxidation has yielded both red-brown and brown iron ore, and the mine has produced some wash ore. There are local occurrences of manganiferous iron ore (Schmidt, 1963).

17.3.8 *An initial hand design*

In 1914, when the diamond drilling was performed, an initial pit was designed and reserve calculations made (Minnesota Department of Mineral Resources, 1999). A plan view of the property showing the drill hole locations, surface features and the pit outline is shown in Figure 17.8. The dashed line represents the surface intercept and the solid line indicates top of ore. A summary of the results is given below:

Approximate depth of pit: 190 ft
Length on surface: 1600 ft
Maximum width on surface: 640 ft
Minimum width on surface: 420 ft
Depth of surface: 90 ft
Yards of surface: 2,275,800
Length, top of ore: 1400 ft
Maximum width, top of ore: 500 ft
Minimum width, top of ore: 240 ft
Average depth in ore: 93 ft
Yards in ore: 1,575,100
Ratio surface to ore (yards): 3 : 2 (approx)
Tonnage factor: 15 $ft^3 = 1$ lt (long ton)
Tons of ore in pit (all assays included): 2,835,200

- Percent iron: 51.31
- Percent phosphorus: 0.192

Tons of ore in pit (50% Fe and over): 1,567,100

- Percent iron: 57.09
- Percent phosphorus: 0.196

17.3.9 *Economic basis*

The ore being mined is hematite and the degree of concentration required depends upon the ore grade. If the grade is greater than 60% Fe, the ore is considered direct shipping and no processing is required other than crushing. For ore of lower grade, a concentration process involving a wash plant is used. Depending upon the grade, this may be followed by a heavy media plant employing ferrosilicon. For the wash plant, the feed averages 49% Fe, the concentrate runs 57% and the recovery is 70%. For the heavy media plant, the feed averages 44% Fe, the concentrate runs 50% Fe, and the recovery is 55%. An economic analysis of a similar property (Faust, 1954) was made in 1954 based upon the use of the

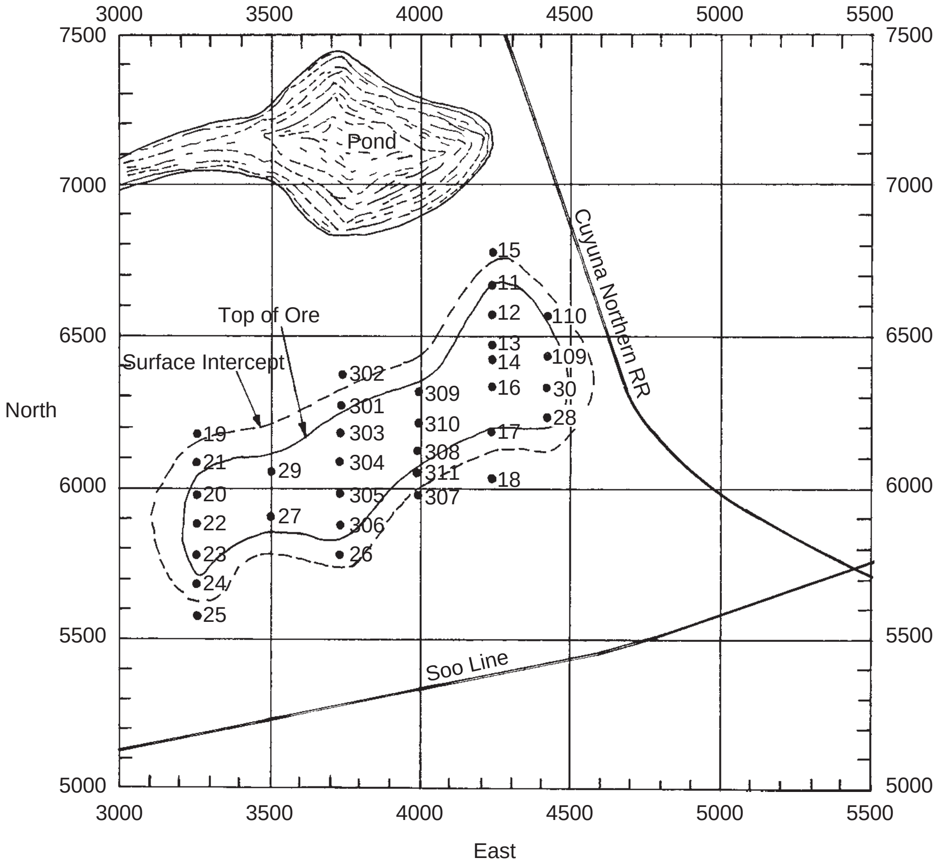


Figure 17.8. Plan map of the Minnesota Natural Iron deposit showing the location of the drill holes. Minnesota Division of Natural Resources (1999).

equipment available at that time and the prevailing labor costs. The following costs were obtained:

Overburden stripping: \$0.35/yd^3
Mining: \$0.48/lt
Crushing: \$0.18/lt
Concentrating: \$0.53/lt
G&A cost: \$0.46/lt

Although quite old and incomplete, these costs may be of some use to the interested student. For the initial designs, the slope angle in overburden is assumed to be 45°, while in ore it is 63°.

The Algoma mine is a neighbor to the Minnesota Natural Iron property. Although costs have not been provided, the following information from the 1965–1966 period (Pfleider and Weaton, 1968) may be of value.

The ownership cost was 11% of the total operating cost. The production rate, at that point, in time was 500,000 lt/year of ore and 3,600,000 lt of waste.

Table 17.6. Productivity as a function of material handled. Pfleider and Weaton (1968).

Material	Productivity (lt/manshift)
Ore	33
Overburden/glacial till	200
Waste rock	110
Total tons handled	60

Table 17.7. Mine operating cost (for ore only) broken down by type of operation. Expressed as a percent (Pfleider and Weaton, 1968).

Operation	Percent
Drilling	7
Blasting	6
Loading	12
Hauling	37
General	38

Table 17.8. Mine operating cost (for ore only) broken down by type of cost. Expressed as a percent (Pfleider and Weaton, 1968).

Type of cost	Percent
Labor	57
Materials, expenses and power	43

17.4 THE UTAH IRON PROPERTY

17.4.1 *Background*

The Utah Iron property is located in the famous Iron Springs iron mining district of southwest Utah. The background information regarding the property has, for the most part, been extracted from the senior thesis ‘Technical and Economic Evaluation of the Jackrabbit Prospect, Iron Springs District, Utah’ by Clifford Krall of the Department of Mining Engineering, University of Utah (Krall, 2004). The drillhole data have been provided by the Geneva Steel Company (Jones, 2001).

As shown in Figure 17.9, the district is located approximately 20 miles southwest of Cedar City, Utah. Three different mining areas, Iron Mountain, Granite Mountain, and Three Peaks, are found within the district. The Utah Iron property lies at the base of the southwest side of Iron Mountain in the midst of what was previously extensive mining activity. The old workings include the Blowout, Burke, Blackhawk, Pinto, and Duncan Pits.

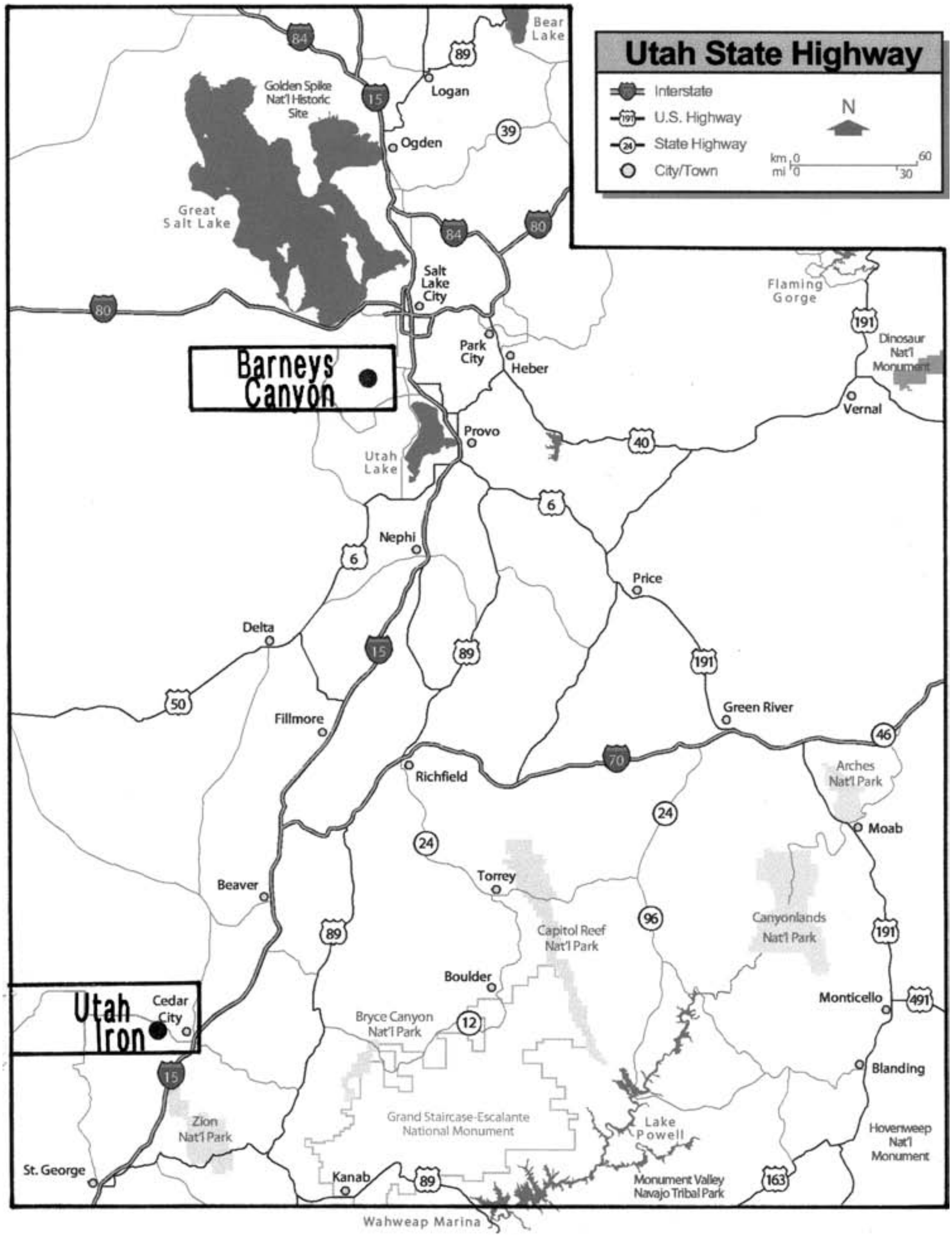


Figure 17.9. Map of Utah showing the location of the Utah Iron and Kennecott Barneys Canyon properties.

17.4.2 *Mining history of the district*

The iron ore deposits in the Iron Springs District were discovered by Mormon pioneers in 1849 (Larson, 1963). With an abundance of iron ore and accessible sources of coal, the pioneers believed that a steel-making industry could be started in Utah. The towns of Cedar

City and Parowan were established on this basis. The first furnace began operation in Cedar City in 1852 but was shut down in 1855. It produced only 25 tons of pig iron. The second furnace began operation in 1868 in Little Pinto, Utah, which is now the deserted ghost town of Old Ironton. It shut down after 6 years of production that produced 400 tons of pig iron (Mackin, 1968).

Over the period from 1923 through 1965, there were three primary phases of iron ore production. During the first phase, which extended from 1923 through 1942, the Columbia Iron Mining Company mined about 4 million long tons of iron ore. Most of the ore was shipped to the company's plant in Ironton. The second phase began towards the end of World War II in 1943 and continued until 1961. Iron ore production totaled nearly 60 million long tons over this period. The Columbia Iron Mining Company and the Colorado Fuel and Iron Corporation (C.F.&I.) were the major producers. Columbia shipped its ore to the newly built Geneva Steel Plant in Provo, Utah. This facility which was started by the federal government was later acquired by U.S. Steel. C.F.&I. shipped its ore to Pueblo, Colorado together with ore from its mine near Sunrise, Wyoming. Utah Construction Company mined a small amount of iron ore and shipped it to the Kaiser Plant in Fontana, California. The third phase took place from 1962 through 1965 during which time nearly 9 million long tons were produced. During this phase, production in the Iron Springs District steadily decreased because the Geneva Plant began to buy its ore from a new mine in Lander, Wyoming. The total production over the entire period from 1923 through 1965 was about 72 million long tons with a value of $340 million (Mackin, 1968). Iron ore production continued at a minimal level through the 1970s and 1980s, but eventually ceased. The high cost of rail transport, the lack of a proven local iron ore supply, and a plummeting U.S. steel market caused the Geneva Steel Plant to shut down and file Chapter 17 Bankruptcy in the late 1990s.

17.4.3 *Property topography and surface vegetation*

The peak elevation of Iron Mountain is about 7830 feet. The Antelope Range lies to the north of Iron Mountain and extends northward about 10 miles. The Harmony Mountains lie to the south of Iron Mountain. The Escalante Desert, which is relatively flat with elevations ranging from 5150 to 5600 feet, bounds the Antelope Range.

The two types of surface vegetation in the Iron Mountain area are forested lands and desert lands. Most of the land occupied by Iron Mountain is forested with the covering consisting of shorter trees and shrubs. The lands surrounding Iron Mountain are largely desert with limited vegetation.

17.4.4 *Climate*

The state of Utah is divided into four main climate zones. These are steppe, desert, humid continental-hot summer, and undifferentiated highlands. The mining area is near the border of the steppe zone and the zone of undifferentiated highlands. A steppe zone is one that occurs between desert regions and mountain regions and the typical average annual precipitation is between 8 and 14 inches. Most steppeland areas in Utah experience winters with average temperatures of less than 32°F., but those in southern Utah usually experience warmer winters (Pope and Brough, 1996).

Table 17.9. Average temperatures at Cedar City, Utah.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave. temp (deg. F)	29.5	34.2	40	47.7	56.4	66.5	73.9	72	63.6	51.8	39.1	30.8
Ave. high temp (deg. F)	38.1	43.6	47.3	53.9	62.5	72.9	78.3	75.3	68.9	57.6	45.5	39.3
Ave. low temp (deg. F)	11.6	21.8	33.4	42.2	49.6	60.8	70.9	66	57.2	45.1	30.9	20.8

A zone of undifferentiated highlands is one that experiences humid winters and summers with average temperatures less than 32° F. and 72° F., respectively. The highlands areas in southern Utah are generally warmer than the average (Pope and Brough, 1996).

The average monthly temperatures as recorded at the Cedar City Airport are shown in Table 17.9. The average annual precipitation is 10.64 inches. Since the Utah Iron property lies at an elevation of about 6600 feet, the average temperature would be slightly cooler and the average precipitation would be slightly higher. The record high and low temperatures recorded in Cedar City within the past 30 years are 105° F. and −26° F., respectively (www.weather.com, 2003).

The precipitation and snowfall contours in the areas surrounding Iron Mountain vary significantly. The higher precipitation tends to occur in the higher mountainous areas, and the lower precipitation tends to occur in the lower, flatter areas.

17.4.5 *General geology*

The 3 mile wide by 23 mile long Iron Springs District hosts the richest iron ore deposit in the western United States. This deposit was exposed to the surface by erosion at three prominent sites: Iron Mountain, Granite Mountain, and Three Peaks. All three iron ore-bearing quartz monzonite laccoliths are oriented along a northeast-southwest trending axis.These are igneous domes that intruded through the previously overlying sedimentary strata. They are oval-shaped ranging from 3 to 5 miles long, and aligned in a northeast direction. The laccoliths were created when an igneous intrusion emerged from 2000 to 8000 feet below the surface to the base of the Homestake Formation, a Jurassic limestone formation. Iron rich liquids and gases were pushed upward by the intrusion, forming magnetite and hematite bodies among the perimeters of the laccoliths. Exposed by erosion, some were re-covered by alluvial deposition (Seegmiller, 1998). The lowest exposed geologic unit in the area is the Homestake Formation, a Jurassic age blue and gray limestone that ranges in thickness from 200 to 350 feet. This is overlain by the Late Jurassic Entrada shale and sandstone formation, which ranges in thickness from 60 to 250 feet. This, in turn, is covered by a variety of Cretaceous limestones, shales, sandstones, and conglomerates, all associated with the Iron Springs Formation. The Iron Springs Formation ranges in thickness from 1000 to 6500 feet (Guilbert and Park, 1986).

The three laccoliths are composed of quartz monzonite and granodiorite porphyry. Each laccolith is composed of three zones: the Peripheral, the Interior, and the Selvage Joint.The Peripheral zone forms the bounding perimeter of each laccolith. It is composed of fine-grained, erosion-resistant quartz monzonite that ranges in thickness from 100 to 250 feet. Since it is erosion-resistant, the outcrops are generally sharp ridges. The Interior zone is a coarser-grained quartz monzonite that is more susceptible to erosion. Its outcrops are generally rounded knobs and flat stretches. The Selvage Joint zone lies between the Interior and Peripheral zones. It is distinguished from the two other zones by the bleaching of rocks on both sides of the joints that separate the Interior and Peripheral zones. The bleaching produced a hard surface crust that stands out from the softer surrounding quartz monzonites. The width of the Selvage Joint zone ranges from less than 4 inches to over 10 feet. The structures of all three laccoliths are similar. (Guilbert and Park, 1986).

The Interior, Peripheral, and Selvage Joint zones all contain small amounts of iron (averaging about 3%), but not nearly enough to mine economically. The rich iron ore lies in replacement ore bodies in the Homestake Formation. The ellipsoid-shaped ore bodies extend up to 1000 feet along both strike and dip and are up to 350 feet in thickness. They lie between the quartz monzonite laccoliths and the Homestake Formation. In areas where the Homestake Formation was completely replaced, the ore bodies extend as far as the Entrada Formation. Before replacing portions of the Homestake Formation, the iron existed within the molten quartz monzonite. When the quartz monzonite intruded upward through the overlying strata, hydrothermal fluids also surged upward, leaching the iron. When the outer zones of the quartz monzonite were cooled and solidified and the inner zones were not yet completely crystallized, the iron-bearing fluids surged upward, causing bulging and jointing in the outer zones of the quartz monzonite. The fluids then moved upward through the joints, through a thin layer of siltstone, and into the Homestake Formation, replacing only the limestone and none of the siltstone. If the limestone formation in the area had been missing, the fluids would have dispersed throughout a larger area of rock, and the rich ore deposits would not have been formed (Guilbert and Park, 1986).

17.4.6 *Mineralization*

High concentrations of iron (50%–65% Fe) exist in pod-shaped magnetite and hematite deposits scattered throughout the outer edges of the laccoliths. They are hosted by the Homestake Formation. Significant amounts of silicon, magnesium, aluminum, and phosphorus exist in the magnetite and hematite ore. The phosphorus content must be taken into consideration because it can cause difficulty in the steelmaking process (Mackin, 1968).

17.4.7 *Mineral processing*

The iron ore mined previously was classified into two categories, direct-shipping ore and low-grade ore. The direct-shipping ore had an iron content ranging from 45% to 68%. The low-grade ore had an iron content ranging from 20% to 45%. Both categories of ore were sent to a primary jaw crusher and crushed to minus 7 inches. The direct-shipping ore was then screened and shipped out on rail cars as either open-hearth or blast furnace ore. The crushed low-grade ore was sent to a secondary crusher and crushed to minus 4 inches and screened at 1¼ inches. The oversize material was sent to an electro-magnetic belt cobber, where the concentrate was shipped as open-hearth ore. The cobber tails were then crushed by a tertiary crusher to minus 1¼ inches and sent to a mill along with the secondary crusher undersize material. The milled material was screened to minus 3/16 inches. The oversize material was sent to the belt cobber, and the undersize material was sent to a wet magnetic separator. Both products were combined and shipped (Bellum and Nugent, 1963).

17.4.8 *Pit slopes*

Geologic structural data were collected from a pit adjacent to the Utah Iron prospect using cell mapping. A total of 440 structural discontinuities such as joints, bedding planes, and faults were included in the overall data set. The dips and dip directions of each discontinuity were recorded. By plotting the poles of all discontinuities on a stereonet, three concentrations were observed:

Discontinuity	Dip	Dip Direction
1	31°	116°
2	72°	342°
3	89°	74°

These will be used to analyze the final pit slopes for possible planar and wedge failures.

17.4.9 *Initial cost estimates*

The waste and ore characteristics are rather similar so the same costs will be assumed for both. The concentration process consists of crushing, grinding, magnetic separation, gravity separation, and flotation. The expected iron recovery from the concentration process is 70%. The general and administrative (G&A) costs will only be applied to the ore tons. Some initial costs are summarized below:

Cost category	Cost
Mining	$0.91 per short ton
Concentrating	$4.54 per short ton of ore
General and Administrative	$0.91 per short ton of ore

An interesting possibility is to process the magnetite fines into Mesabi nuggets using the coal reserves found in the near vicinity. The interested student is invited to explore this possibility.

17.4.10 *Other considerations*

Most of the future employees will probably reside in Cedar City. The one-way travel distance from Cedar City to the mine would be about 18 miles, 15 miles on Highway 56 and 3 miles on access roads. Highway 56 is maintained by the state of Utah, and the mining company would maintain the access roads.

A spur of the Union Pacific Railroad extends from a load out facility on the east side of the Utah Iron property to the main line located about 10 miles northwest of Cedar City. From here, the ore can continue to ports in California or to other potential customers.

17.5 THE MINNESOTA TACONITE PROPERTY

17.5.1 *Introduction*

The Minnesota Taconite property is located in northeastern Minnesota on the famous Mesabi Iron Range (Figure 17.7). Diamond core drilling of the property was performed by the Longyear Company between 1963 and 1968. The Jones and Laughlin Steel Corporation proposed a mining operation in 1978 but it was never developed. The background information regarding the property has, for the most part, been extracted from the senior thesis

'Technical and Economic Evaluation of the Biwabik Taconite Operation, Mesabi Iron Range, Minnesota' by Manuel Bello Freire of the Department of Mining Engineering, University of Utah (Bello, 2004). Drill hole data have been generously provided by the Minnesota Department of Natural Resources (Schmucker, 2003). Mark Jirsa of the Minnesota Geological Survey (Jirsa, 2005) provided the structural data.

17.5.2 *Location*

Biwabik, Minnesota (population 934), at 2 miles away, is the closest town to the Minnesota Taconite property. To the west, are the population centres of Gilbert (9 miles, pop. 1847), Virginia (13 miles; pop. 9157), and Eveleth (14 miles; pop. 3857). To the east, lie the towns of Aurora and Hoyt lakes. Duluth (pop. 86,918), the largest city in St. Louis County, is 53 miles away. It is located at the western tip of Lake Superior and its deep water port provides the main access to the Great Lakes market. Topographic information for the property is provided by the Biwabik, Gilbert and McKinley U.S.G.S. quadrangles. State highway 135 passes through the town of Biwabik. The other major roads are state highways 53 and 169.

The Duluth, Missabe & Iron Range Railroad (DMIR) which is now owned by Canadian National Railroad passes through the town and could be used to transport taconite pellets to either Duluth or Two Harbors.

17.5.3 *History*

Commercial iron mining has been conducted in Minnesota for more than 120 years. It began on the Vermillion Range in 1884, came to the Mesabi Range in 1892, and finally spread to the Cuyuna Range in 1911. Although the Mesabi Range was a late starter, it quickly established itself as the dominant producer. It is a belt 110 miles long, 1 to 3 miles wide, and a thickness up to 500 feet. Yearly production on the Mesabi Range increased nearly linearly from zero in 1893 to 45 million tons in 1914. Minnesota production stayed at that level until the start of the Depression in 1929 when it fell precipitously. Late in the 1930's it grew once again and remained at the 70 million ton level during WWII and the following Korean War. The peak production exceeded 80 million tons. For the 20-year period stretching from 1960 through 1980 the average production rate was about 55 million tons. Over the past 20 years it has averaged about 45 million tons per year or about the same as the WWI period.

Production in the early life of all of the ranges was in the form of direct shipping ore. From the open pits this meant that the ore was directly loaded onto trains which continued directly to the ports on Lake Superior. Here the ore was transferred to boats for the long journey to the steel mills. With time, the percentage of ore requiring some type of processing prior to shipment steadily increased. Initially it was simply washing to remove some of the silica but eventually other techniques such as heavy media separation were used. The end of Minnesota's natural iron ore industry (Direct Shipping plus Concentrates) occurred in 1984 when the last shipment left the Mesabi Range.

With the approaching exhaustion of the relatively high grade, easily mined, natural ores, the mining companies and their steel company owners began to seriously look at replacement sources of iron. The presence of enormous tonnages of the low-grade iron resource, taconite, was well-known on the Mesabi Range. Gruner (1954) calculated that between the towns of Mesaba and Grand Rapids, a distance of 71 miles, there could be 89 billion tons of taconite running, on average, 27 percent Fe (ranges from 15 to 40 percent Fe) and 51 percent

Table 17.10. Average monthly temperatures at Biwabik, Minnesota.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave. temp (F)	3.5	10.5	22.9	37.9	51.3	60	64.5	62.4	52.5	40.9	24.6	9.7
Ave. high temp (deg. F)	15.6	23.5	35	50.4	65	73	77	74.7	64.4	51.5	33.6	20.2
Ave. low temp (deg. F)	−8.7	−2.7	10.7	25.3	37.7	47.1	52	50	40.5	30.2	15.5	−0.9

www.city-data.com/city/Biwabik-Minnesota.html

silica. Studies to find techniques to economically recover the resource had been underway since the early 1900's. The most notable of these were carried out by E.W. Davis at the Minnesota School of Mines Experiment Station. In 1940 Erie Mining Company was formed after extensive research had demonstrated the possibilities for the successful processing of taconite and the extent of the available reserves. In 1946, Erie authorized a plant to be built at Hoyt Lakes, Minnesota to produce 200,000 tons per year of high-grade pellets. It began operating in mid-1948. In 1954 construction of a plant with an annual capacity of 7.5 million tons of pellets was begun. Operations were started in 1957 (Thomte, 1963). Other producers followed and at the beginning of 2003, Minnesota taconite pellet capacity was at the level of 42.7 M-lt/year.

17.5.4 *Topography and surface conditions*

The surface elevation of the Minnesota Taconite property ranges from 1359 to 1612 feet above sea level. The vegetation in the area is mainly second growth forest with the primary species being black spruce, northern white cedar, tamarack and black pine. The ground cover includes ericaceous shrubs (leather leaf, cranberry, Labrador tea), as well as other kinds of shrubs (speckled alder). It is common to find a continuous sphagnum carpet and some other mosses. The other vegetation in the area consists of forbs (false Solomon's seal), cotton grass and a few types of sedge.

There are many open waters (lakes, rivers, and streams), woody wetlands and emergent herbaceous wetlands on and surrounding the property. The Biwabik area is in the St. Louis River watershed of the Great Lakes Basin. The Pike River flows close to the property and, in addition, there are a large number of lakes in the near vicinity. These include Embarrass Lake, White Lake, McKinley Lake, Deep Lake, Gill Lake, and Leaf Lake. The presence of many wetlands gives the impression of a very wet landscape. The Biwabik iron formation is considered to be the main source of ground water for the area. Since there are no large-scale regional aquifers, water availability to the Iron Range cities is directly related to this geological formation.

Table 17.10 provides average temperature data for Biwabik, Minnesota (www.city-data.com). The temperatures vary between being mild in the summer to extremely cold in the winter. Snow is present nine months of the year which must be carefully considered in any design.

17.5.5 *General geology*

The taconite on the property lies within the Biwabik iron formation. This formation, a member of the Animikie group which is late Precambrian in age, is estimated to be about 1.9 billion years old. Composed mainly of iron oxides (magnetite, hematite and/or goethite),

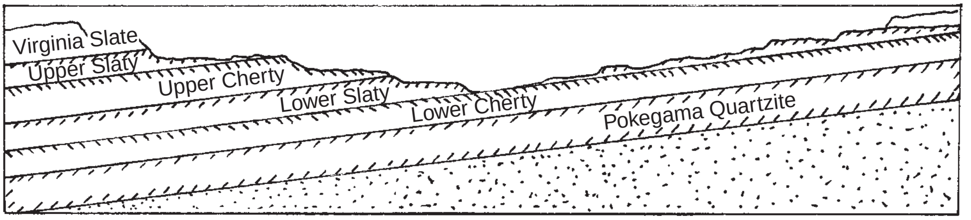


Figure 17.10. Cross-section (looking west) showing the Mesabi Range geology.

Table 17.11. Density of the different rock materials.

Material	Density (kg/m^3)
Overburden (glacial till)	1780
Biwabik iron formation	3142
Virginia formation slate	2595
Virginia formation contact hornfels	2800
Biwabik iron formation contact hornfels	4040
Pokegama quartzite	2673

iron silicates and iron carbonates, it is partly overlain by the Virginia formation and rests on the Pokegama formation. The four primary subdivisions of the Biwabik iron formation, beginning from the bottom, are the Lower Cherty, the Lower Slaty, the Upper Cherty and the Upper Slaty. The overlying rock consists of Virginia slate as well as sandstone and shale. A typical cross-section of the Mesabi Range geology is shown in Figure 17.10 (looking west). The overburden is mainly composed of glacial till which is removed without the need for drilling and blasting.

The two ore-producing layers are the Upper Cherty and the Lower Cherty members. The Upper Cherty member is subdivided into ore and waste rock based on magnetic iron content. The Lower Cherty member is subdivided into top and bottom Lower Cherty depending on the percentage of silica in the Davis tube concentrate. The top Lower Cherty has the lowest iron and highest silica while the bottom Lower Cherty contains the highest iron and lower silica. The primary minerals found in the formation are magnetite, chert and iron silicates (greenalite, minnesotaite and stilpnomalane). The secondary minerals are hematite, carbonates, pyrite and chlorite. Magnetite is the primary ore mineral, and occurs in irregular and regular bands, disseminated single euhedral crystals, and patchy mottled concentrations (Holgers, et al, 1969).

The average grade for the ore body is about 23% magnetic Fe. Table 17.11 provides the densities of the materials involved in the mining operation.

17.5.6 *Structural data*

The Biwabik iron formation strikes roughly N 65° E and dips 6 degrees (on average) to the south-southeast. Faults typically strike west to northwest (west-northwest is the predominant fabric) and dip near vertical. Offset on the faults is typically minimal (less than 100 feet). The sense of displacement varies, and some faults have an apparent scissors offset. It is not certain whether this is a function of the tectonic setting, or merely reflects localized subsidence and collapse due to secondary oxidation and leaching. Many of the 'faults' however, are likely just fracture zones consisting of closely spaced joints. Based upon mapping in nearby mines, Jirsa (2005) of the Minnesota Geological Survey has provided the structural data for the property given in Table 17.12.

17.5.7 *Mining data*

Data from a neighbouring operation which might be helpful in preparing a design are provided below:

1. The geology on the Range is fairly simple. Coal seam algorithms are used to model the 'layer style' ore body.
2. Long tons are the standard unit of measure on the Range. One long ton equals 2240 pounds or just slightly greater than a metric ton.
3. The ore body daylights to the north and dips to the southeast at 8%.
4. Geometries
   – Bench height: Ore: 40′, Waste: 20 to 45′
   – Working bench width: 300′
   – Safety bench width: 50′
   – Bench face angle: 65 degrees
   – Angle of repose for broken rock/ore is 35 degrees
   – Final slope angle: 54 degrees
   – Road grades: 6 to 8%
   – The glacial till overburden is removed using shovels or loaders. Blasting is not required
   – Run-of-mine ore grade (% magnetic Fe): 20%
   – Cut-off grade (% magnetic Fe): 14%
   – In-situ density: 2.45 long tons/cubic yard for both ore and waste (the waste rock is lower grade iron formation)
   – Bulk ore density (blasted): 1.8 long tons per cubic yard
   – Overburden (glacial till) density: 1.5 long tons per cubic yard
5. Mining costs

Operation	Cost ($/lt)
Loading (shovels/loaders)	0.25
Hauling	0.20 (average haul)
Drilling	0.15
Blasting	0.15
Primary crushing	0.15 (material crushed to −9″)
G&A	0.40

6. Operational aspects
   – Operating days per year: 365
   – Operating shifts per day: 3
   – Operating hours per shift: 8
   – Productivity: 600 lt/manshift (determined using all members of the open pit work force)
   – Work force breakdown: Staff (13%), equipment operators (55%), mechanical and maintenance personnel (32%)

Table 17.12. Structural data from neighboring taconite mines (Jirsa, 2004).

1. Biwabik-Canton Mines (sections 2 and 3)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M300	10	83	JQ	
M300	119	87	J1	
M300	215	90	J2	
M300	188	13	B	
M301	145	14	B	
M302	160	24	B	
M302	110	90	J1	
M302	2	90	J1	
M302	65	90	J2	
M303	177	19	B	
M303	5	82	JQ	
M304	3	90	FT	
M305	5	87	JM	
M306	330	85	JZ	
M306	185	90	JM	
M307	210	12	B	
M307	175	9	B	
M307	355	90	JM	
M307	35	85	JM	
M308	105	90	J1	
M308	25	90	J1	
M309	168	20	B	
M309	210	86	J1	
M309	283	87	J1	
M311	200	67	FT	
M312	340	82	J1	
M312	115	87	J1	
M312	20	90	J2	
2. Mary Ellen Mine (sections 9 and 10)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M233	137	6	B	
M233	70	86	J1	
M233	185	90	J1	
M233	30	90	J2	
M233	220	90	J2	
3. McKinley Mine (section 8)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M234	130	12	B	
M234	25	90	J1	
M234	120	90	J1	
M235	70	88	J1	
M236	175	90	JZ	
M237	250	65	JW	
M237	60	90	J1	
M237	95	90	J1	
M237	5	90	J1	
4. Corsica Mine (section 18)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M256	165	8	B	
M256	200	87	J1	monocline defines
M256	100	90	JQ	west edge of mines

(Continued)

Table 17.12. (Continued).

M257	170	55	B	
M258	152	20	B	
5. Laurentian Taconite Mine (section 24)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M239	350	74	JQ	
M240	140	16	B	
M240	20	90	J1	
M240	120	85	J1	
M240	45	90	J2	
M241	15	90	JQ	
051 Mine (near Gilbert, SE corner section 24)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M254	143	13	B	
M254	55	86	J1	
M254	185	87	J1	
M254	128	90	J2	with abundant brecciated quartz
M255	75	55	FT	
Gilbert Mine (sections 25 and 26)				
STATION	DIP DIR(deg)	DIP(deg)	TYPE**	COMMENTS
M251	148	13	B	
M252	128	13	B	
M252	105	90	J1	
M252	215	90	J1	
M253	140	19	B	
M253	228	76	JM	
M253	152	84	JM	

* TYPE = STRUCTURE CODES:
B = bedding; typically also a joint set
J1 = joint, prominent set
J2 = joint, secondary/tertiary sets (typically less well developed)
JM = joint, mineralized with hematite, martite, limonite, goethite
JQ = joint, mineralized with quartz
JW = joint, mineralized and weathered
JZ = joint zone with associated significant collapse
FT = fault.

- Drilling: Rotary drills, 16″ diameter bits
- Penetration rate: 30 ft/hr
- Hole length: bench height + 5 feet of subdrilling
- Powder factor: 0.55 lbs/lt
- Hole charging: Done by vendor
- Loading: Combination of rope shovels (35 yd, 18 yd and 22 yd), hydraulic shovels (35 yd), and front end loaders
- Haulage: Fleet of 240 t trucks
- Crushing: 60″ gyratory

17.5.8 *Ore processing*

The ore coming from the mine will have an average magnetic iron content of about 20%. To facilitate liberation of the iron particles from the waste, the rock must be reduced down to a size of about −400 mesh using a variety of techniques. A description of the process used at the Tilden mine in Michigan may be found on the Web at (www.geo.msu.edu/geo333/tilden.html). The iron powder is moistened, combined with bentonite clay as a binder, limestone and dolomite as a flux, rolled into spheres 3/8 of an inch in diameter, and then fired into the hard round pellets suitable for use in a blast furnace (www. geography.about.com). The final pellets have an iron content of about 65% Fe. The two main kinds of pellets are flux pellets and acid pellets.

For this small property, it is expected that the crushed ore will be sold to one of the neighbouring pelletizing facilities.

17.6 THE KENNECOTT BARNEYS CANYON GOLD PROPERTY

17.6.1 *Introduction*

The Kennecott Barneys Canyon gold property is located on the east slope of the Oquirrh Mountains, 30 miles southwest of Salt Lake City, Utah and 5 miles to the north of the famous Bingham Canyon copper mine (Figure 17.9). Barneys Canyon is comprised of two gold deposits, the Barneys Canyon deposit and the Melco deposit. Exploration and delineation drilling was carried out between 1985 and 1987. Construction of the open pit and heap leaching facilities began in September 1988 and mining was initiated in February 1989. The first gold was poured on September 15, 1989. At that time, the estimated overall property life was 11 years with an annual production of more than 110,000 ounces (Kennecott, 2002). Mining at the property ceased in 2001 but leaching continued until 2005. The property is currently in final reclamation (Gottling, 2005).

The drill hole data included on the distribution disk are for the Barneys Canyon deposit which was mined between 1989 and 1996. They have been very kindly provided by the Kennecott Barneys Canyon Company (Switzer, 2001 and Gottling, 2005).

Some of the background information regarding the property has been extracted from the senior thesis ‘The Oquirrh Mountain Gold Prospect: Technical and Economic Analysis’ by Adam Jacobsen and Anthony Lowe of the Department of Mining Engineering, University of Utah (Jacobsen and Lowe, 2003).

17.6.2 *Geologic setting*

The material included in this section is largely based upon material contained in the thesis by Presnell (Presnell, 1992). The rocks of the Oquirrh Mountain Range vary in age from Cambrian to Carboniferous. Quartzites and limestones are the most abundant, although some shales are also present. The Bingham quartzite, which is fine-grained and nearly white in color, is the most widespread rock of the Range. Although of relatively minor importance from a stratigraphic viewpoint, the limestones are of great economic importance due to their relationship to the ore deposits. Unaltered, the limestones may be gray, blue, or even black.

Two major stratigraphic units, the Park City formation and the Kirkman-Diamond Creek formation, have been identified. The Park City formation consisting of limestone, dolomite and siltstone overlies the Kirkman-Diamond Creek. The latter formation is made up of sandstone and siltstone. These formations have been thrust over shales and basalt flows.

The Oquirrh Mountain range is composed of a series of broad open folds that trend to the northwest. The general north-south trend of the range is due to a strong fault which

Table 17.13. Fictitious Barneys Canyon structural data.

Measurement	Strike	Dip	Measurement	Strike	Dip
1	53	86	21	312	72
2	57	83	22	346	68
3	51	84	23	308	65
4	50	82	24	310	66
5	48	83	25	310	84
6	231	86	26	308	62
7	47	82	27	308	64
8	244	87	28	170	47
9	83	84	29	320	68
10	64	86	30	322	68
11	68	84	31	320	65
12	68	86	32	168	50
13	69	80	33	174	46
14	61	81	34	320	70
15	300	64	35	170	50
16	295	68	36	274	51
17	320	75	37	275	51
18	316	70	38	330	75
19	315	65	39	274	38
20	320	68	40	294	38

truncated the folds along the western front. Numerous minor faults are present. These are important due to their association with the ore deposition.

Fissuring of the rocks, accompanied by slight movement, has been important to the deposition of ores. These ore bearing fissures vary in direction but generally trend north.

Faulting occurred prior to the mineralization. Faults, joints and fissures were the conduits for the mineralized hydrothermal fluids. There is no indication of any post-mineralization movement along the faults.

Gold mineralization is controlled by favorable stratigraphic units and by the structural complexities introduced by faults. The gold occurs as submicron to micron sized, well-disseminated particles associated with iron oxide (pyrite) and is contained in the dolomite. On the atomic scale gold is replacing pyrite contained in the dolomite. Other associated minerals are pyrite and marcasite.

17.6.3 *Resource definition*

The exploration program consisted of 142 holes. The cores were logged and then split with one half being assayed for gold content and the other half stored on the property. A block model was constructed based upon block dimensions of 50 ft × 50 ft × 20 ft. Several types of kriging were used to assign the grades to the blocks. The grades are lithologically controlled for the most part.

17.6.4 *Geotechnical data*

The strike and dip measurements given in Table 17.13 are not from Barneys Canyon. They have been supplied simply to facilitate the making of a slope stability evaluation. The actual slopes stand very steeply as can be seen in photos of the pit (Peterson, 2005).

Table 17.14. Average monthly temperatures at Salt Lake City, Utah.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave. temp (F)	27.9	34.1	41.8	49.7	58.8	69.1	77.9	75.6	65.2	53.2	40.8	29.7
Ave. high temp (deg. F)	36.4	43.6	52.2	61.3	71.9	82.8	92.2	89.4	79.2	66.1	50.8	37.8
Ave. low temp (deg. F)	19.3	24.6	31.4	37.9	45.6	55.4	63.7	61.8	51	40.2	30.9	21.6

www.climate-zone.com/climate/united-states/utah/salt-lake-city

17.6.5 *Topography and surface conditions*

The property is located in the northwest corner of the Lark quadrangle and the northeast corner of the Bingham quadrangle. The topography of the area is very rugged with the surface elevation of the property ranging from about 5500 ft to 6900 ft above sea level. A surface topography map covering the pit can be created using the collar elevation of the 142 drill holes.

No permanent surface water sources have been detected. In the spring, water runs down the streams and some small ponds are formed. The small ponds eventually dry out in the hot summer months.

Topsoil depth ranges from 2 to 6 feet thick. The greater thickness occurs in the valleys and lower areas where water has accumulated. The topsoil is gray in color and contains little in the way of nutrients or biologic matter. This is similar to many areas of the Oquirrh Mountain range. The primary vegetation is ricegrass, horsebrush, sagebrush, and other grasses with oak brush, junipers, and pinions scattered about the landscape. The property has similar vegetation to the rest of the Oquirrh Mountain range. The property has no endangered or threatened species of plants.

The property is home to many different species of mammals, which include; elk, deer, coyote, rabbits, mice and other species of rodents. The elk and deer migrate across the property during spring and fall. Small birds and some falcons populate many areas of the property. No endangered species or threatened wildlife has been found on or near the property.

17.6.6 *Climate*

The average monthly temperatures as recorded at the Salt Lake City, Utah airport are provided in Table 17.14. Although the geographic separation is only about 30 miles, the elevation of the property is about 2000 ft higher than the airport. Hence somewhat cooler temperatures are expected.

17.6.7 *Ore processing*

The information included in this section was extracted from the brochure ‘Barneys Canyon Mine’ (Kennecott, 2002). Only the oxide process is described here since it was the one used to process the ore from the Barneys Canyon pit.

The ore is crushed to −1.5 in using a combination of jaw and cone crushers. It is then sampled, weighed and mixed with water and cement to form an agglomerate. The agglomeration process binds or cements the fine particles to improve solution flow during leaching. Following agglomeration, the ore is transported to the leach pad using a series of overland (fixed) and portable conveyors. At the pad (see the flowchart in Figure 17.11) the material

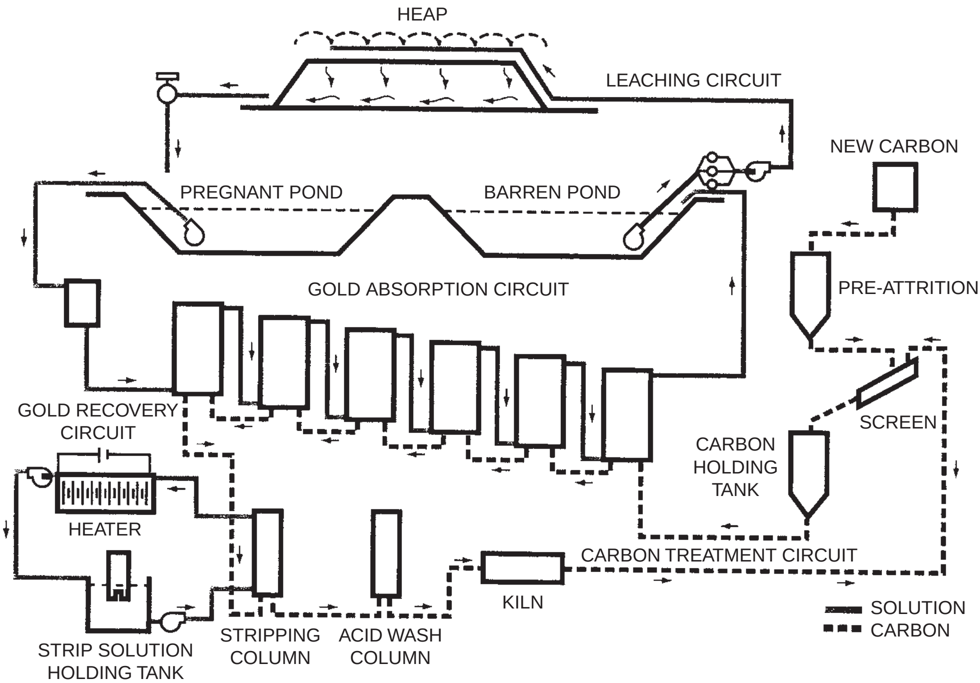


Figure 17.11. Kennecott Barneys Canyon gold leaching and recovery circuit. Kennecott (2002).

is 'heaped' to a height of 17 feet using a radial ore stacker. The ore is then sprinkled with a cyanide solution which percolates through the heap dissolving (leaching) the gold in the process. Solution flowing from the pad, referred to as 'pregnant' solution, is transported by gravity in a double-lined piping system to a lined impoundment (pregnant pond). The solution is then pumped into the process plant and through one of two series of six carbon columns where the gold is removed by absorption on activated carbon granules. The now spent solution, referred to as 'barren' solution flows by gravity to a barren pond and is eventually recycled to the leach pad.

The gold is stripped from the carbon in one of four strip columns using a hot (1900°F) solution. This concentrated solution flows to one of two electro-winning cells where the gold is electrolytically deposited on steel wool. After electro-winning, the loaded steel wool cathodes are processed in an induction furnace to produce gold doré bricks which are shipped to a precious metals refinery for final upgrading and recovery of minor amounts of silver.

17.6.8 *Mining data*

The following historical data regarding the Kennecott Barneys Canyon operation were kindly provided (Switzer (2001), Slothhower (2001)) to assist students in evaluating the property.

1. Costs
   - Mining: \$1.50/ton ore; \$0.50/ton waste
   - Processing: \$4.50/ton ore (crushing, transport, stacking, leaching)
   - Refining cost: \$20/Tr. oz.
   - G&A cost: \$1.50/ton ore

2. Additional information
   - Rock density: 14.1 ft^3/ton
   - Recovery: 85%
   - Production rate: 7300 tpd ore, 30–40,000 tpd ore and waste
   - Operations: 24 hours/day and 7 days/week
   - Personnel: 185 production, maintenance and administrative
3. Pit geometry
   - Bench height: 20 ft
   - Bench face angle: >64°
   - Overall slope angles: 46°
   - Final slopes: Triple benches and then a 60 ft wide safety berm
4. Unit operations
   - Drilling: 6-1/2" diameter holes
   - Charging: Explosive is ANFO
   - Loading: Front end loaders
   - Hauling: 55-ton capacity haulage trucks

17.7 THE NEWMONT GOLD PROPERTY

17.7.1 *Introduction*

The Newmont Gold property is located within the Carlin trend mining district of Nevada (Figure 17.12). The Carlin trend, one of the world's leading mining districts, is a 40 mile long by about 5 mile wide belt of gold deposits extending in a north-northwest direction through the town of Carlin, Nevada. Gold was first discovered in the area in the 1870's but there was little production until 1909. From that time until the Carlin deposit was discovered in 1961, only 22 thousand ounces were produced. The production then boomed with the total reaching 50 million ounces by 2002. Currently, the annual production from the Carlin trend is about 4 million ounces (Price, 2002).

In defining the Newmont Gold prospect, a total of 119 diamond drill holes ranging in depth from 95 to 1124 ft (average depth of 575 ft) were drilled. Their locations are shown in Figure 17.14. The data included on the distribution disk were very kindly provided by the Newmont Gold Company (Perry, 2005). The support of the company in this effort is gratefully acknowledged. Although the data are from an actual Newmont Gold property, they have been somewhat modified to better suit the present requirements. The location assigned does not correspond to the actual property location.

The background information regarding the property has, for the most part, been extracted from the senior thesis 'Evaluation of the Quasar Project for S & W Mining Company' by Craig Wengel and Amanda Smith of the Department of Mining Engineering, University of Utah (Wengel and Smith, 2001).

17.7.2 *Property location*

The Newmont Gold property is located near state highway 766 in the Schroeder Mountains of Eureka county. Carlin, the nearest town is approximately 8 miles south of the property. The latitude and longitude of the property are 40.96 N, 116.31 W and the elevation is about

Figure 17.12. Map of Nevada showing the location of the Newmont Gold property. (www.amerisar.org/nvmap.html).

5700 feet above sea level. This area is commonly known as the North Eastern Great Basin and has a wide variety of terrain.

17.7.3 *General geologic setting*

The information presented in this section was provided to Wengel and Smith (2001) by Joe Miller, a local mine geologist (Miller, 2001).

The Newmont Gold deposit is one of several structurally-controlled oxide gold deposits located along the Good Hope fault. Striking northwest and dipping at about 35° to the northeast, this fault forms the southwestern edge of the Schroeder Mountain uplift. Because of its presence, the Siluro-Devonian age Roberts Mountains formation lies above the Devonian age Rodeo Creek unit. Regionally, there are three Paleozoic lithologic groups exposed which are overlain by Tertiary age fluvial and lacustrine sediments. These groups are a western siliceous assemblage, an eastern carbonate assemblage, and an intermediate mixed siliceous/carbonate assemblage.

The deposit consists of four stratigraphic units. These units include the Devonian Rodeo Creek unit, a Devonian age limestone, the Tertiary Carlin formation, and limited intrusives. The Devonian Rodeo Creek unit occurs in the footwall of the Good Hope fault. It consists of thinly bedded siliceous mudstone and siltstone interbedded with finely laminated silty limestone, resulting in a mixed siliceous/carbonate rock assemblage. The Devonian age limestone is on the southeast side of the property. It is a massive micritic limestone and calcarenite grading upward into thin bedded silty limestone. The Tertiary Carlin formation overlies Paleozoic rocks in many places. It is a layer of poorly consolidated volcanic sediments, siltstones, clays, gravels, and colluvium. The intrusive rocks are limited to small discontinuous dikes in faults. They are composed of quartz, sericite, and jarosite.

17.7.4 *Deposit mineralization*

The information contained in this section was provided to Wengel and Smith (2001) by Joe Miller (Miller, 2001). It is reproduced here as originally presented by Wengel and Smith (2001).

The Newmont Gold property hosts a strata-bound gold deposit, containing submicroscopic, disseminated gold. The mineralization is hosted in carbonate rocks of Paleozoic age that have been metamorphosed to varying degrees. The mineralization is predominantly in the decalcified silty limestone of the hanging wall of the Good Hope fault system. The mineralization which follows the dip of the Good Hope fault is comprised primarily of leach-grade (0.006–0.05 opt) ore with small amounts of higher grade ore (0.05 and greater). The high grade ore zones are located at structural intersections and at depth in the deposit. The deposit is also surrounded by a low grade halo of mineralization. All of the ore is oxidized and contains less than 0.05% sulfur in sulfides. This will enable the entire ore zone to be processed using cyanide and eliminates any concern about acid production in waste dumps.

Fractures in the Good Hope fault system served as the main conduit for the hydrothermal fluids which altered the rock. These hydrothermal fluids were also transported through the rock due to its permeability. Alterations from these fluids include silicification, decalcification, sericitization, and late alunite replacement. Kaolinite and montmorillonite also occur in minor amounts in most of the altered rocks.

Silicification, which occurred in three phases along the fault zone, produced rocks that are mostly quartz and barite. In the first phase, a cherty-quartz filled fractures in the barite. Next, the barite was fractured and cemented with a coarser chert/chalcedony. Later, any remaining cavities and fractures were filled with coarse quartz, resulting in the solid rock present today. Decalcification is the dominate form of alteration in the hanging wall and near the Good Hope fault. This alteration increased the porosity and decreased the specific gravity of the rock.

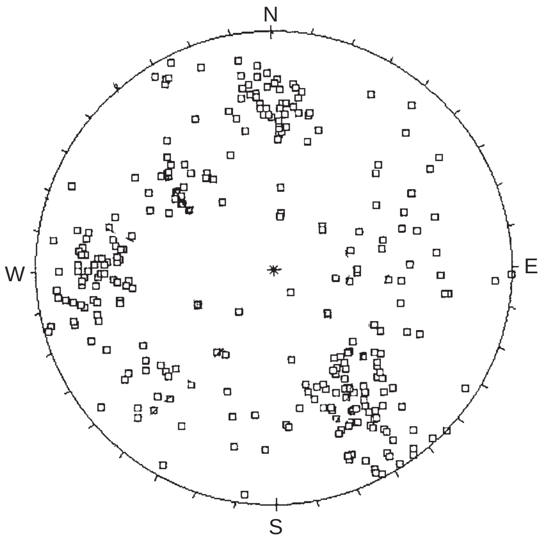


Figure 17.13. Stereonet plot of the structural features used for analyzing the Newmont Gold Property slopes. Wengel and Smith, 2001.

Samples from this alteration zone contain silt to fine sand-size quartz grains in a sericite matrix. Decalcification also occurs locally in thin silty limestone interbeds in the Rodeo Creek unit. These rocks are similar in nature to the Roberts Mountains formation, except that they contain up to 10% chlorite. Sericitization occurs locally near the contact of the Rodeo Creek unit and the underlying Devonian age limestone. Here, however, the rocks do not contain economic gold grades. Supergene alunite alteration is abundant and occurs in the southwest area of the property. Samples from this zone contain silt-size quartz grains in an alunite matrix. The area also contains sheet alunite veins.

The structural data presented in stereonet form in Figure 17.13 were obtained from a neighboring mine. They may be used to assess the slope stability of the proposed pit. For the initial studies, it should be assumed that the friction angle is 35° and the cohesion is zero.

17.7.5 *Topography and surface conditions*

A topographic map has not been provided. It is recommended that one be generated based upon the drill hole collar positions. The area of the Newmont Gold deposit has steep-gradient (erosional) channels with narrow riparian vegetation communities as well as low-gradient (depositional) channels with wide flood plain development. These conditions have lead to the emergence of many species of vegetation. These species include various grasses, willows, shrubs, and forbs. One sensitive grass species, Lewis Buckwheat, is also present. Additionally, the area contains one noxious weed, Scotch Thistle, which can make an area unsuitable for grazing cattle. Vegetation covers approximately 50 percent of the surface area. Although wetland communities are sometimes found in the area, because of the geologic substrate, there are no wetlands on the property.

There are a wide variety of animals on and around the property area. The animals of major concern are those which are endangered, threatened, or considered sensitive species. These species include eagles, hawks, owls, osprey, sage grouse, American white pelican,

Table 17.15. Average monthly temperatures at the Newmont Gold property.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave. temp (F)	25.1	31.5	37.6	44.3	53.1	62.4	70.7	68.7	58.7	47.7	35.8	25.7
Ave. high temp (deg. F)	36.7	43	50.2	59.1	69.4	80.2	91	88.6	78.3	65.9	49.1	37.4
Ave. low temp (deg. F)	13.4	19.9	25	29.5	36.8	44.6	50.3	48.6	38.9	29.6	22.5	14

www.climate-zone.com/climate/united-states/nevada/elko

white-faced ibis, Preble's shrew, myotis, and some rare species of bats. The area is also home to one large game animal, the mule deer, which uses the area as a transition range in spring and fall.

17.7.6 *Local climatic conditions*

Most of Nevada is affected by westerly frontal storms originating from the Pacific ocean (www.wrcc.dri.edu, 2001), (www.silver.state.nv.us, 2001). Precipitation patterns from these storms include westerly winter flows as well as summer thunderstorms influenced by southwesterly flows. This climate is classified as a mid-latitude steppe. Average annual rainfall is 14.3 inches, and average snowfall is 40 inches. The average monthly precipitation is 1.2 inches with higher levels from October to February occurring mostly as snowfall. These precipitation amounts are not high enough to present any landslide or flood potential. They will, however, affect pond levels and road conditions.

Temperatures in the project area have a wide daily and seasonal variability. The average monthly temperatures at Elko, Nevada which would be similar to those at the property are given in Table 17.15. During the day, temperature variations of 30° to 40°F are common due to the high elevation, the proximity to the mountains, and limited cloud cover. Wind speeds are generally low, averaging about 6 mph from the west-northwest.

17.7.7 *Initial pit modeling parameters*

The data provided below are included to allow an initial analysis to be performed.

1. Costs
   - Mine & Haul: $1.00/ton
   - Mine & Haul increment: $0.05/ton/level
   - G&A: $1.38/ton
   - Processing: $4.50/ton ore (crushing, transport, stacking, leaching)
   - Refining: $20/ounce
   - Recovery: 85%
2. Slope data
   - Rock density: 150 lbs/ft^3
   - Bench height: 30 ft
   - Bench face angle: 70°
   - Overall slope angle: 45°

During the study, the reader should try to develop more representative costs and appropriate geometries.

17.8 THE CODELCO ANDINA COPPER PROPERTY

17.8.1 *Introduction*

In March 1999, a relatively small, but representative, drill hole data set from the Sur Sur mine of Codelco's Andina Division was used as part of a post-graduate course on open pit planning conducted in the Department of Mining Engineering at the University of Chile. After performing a pit design using CSMine, the students had the opportunity to visit the mine in person. It was a spectacular setting and a wonderful trip. The mine, which is at an elevation of about 4000 m, is surrounded by the high peaks of the Andes Mountains. In the upper part of the pit, the Rio Blanco glacier forms the east wall and the Rinconada glacier the west wall. The continual movement of the Rio Blanco glacier toward the pit and the Rinconada glacier parallel to the west wall provides some very interesting mine design as well as operational challenges.

The Codelco Andina Division was approached about the possibility of including this data set in the book and they very graciously agreed. The authors have adjusted both the elevations and the hole positions to make the data set more amenable for use with CSMine. An excellent paper describing the property is 'The Sur Sur Mine of Codelco's Andina Division' authored by R. Apablaza, E. Farias, R. Morales, J. Diaz and A. Karzulovic (Apablaza et al, 2000). The information contained in this paper plus additional material supplied by the Andina Division has been used in preparing this section. It is suggested that the geotechnical information be applied when doing the new pit design.

The authors are grateful to Codelco's Andina Division for sharing this information with us and with future mining engineers.

17.8.2 *Background information*

The Andina Division of Codelco mines the Rio Blanco copper/molybdenum deposit located 80 km northeast of Santiago in Chile's Region V. The nearest major town is Los Andes City which is about 40 km to the northwest. Although the existence of the deposit was known since 1920, mining did not become a reality until 1970. Currently, the Andina Division operates two mines: Rio Blanco, a panel caving operation, and Sur Sur, an open pit. Together they produce about 220,000 tons of copper concentrate and 1,900 tons of molybdenum per year.

At the Sur Sur mine which began operation in 1981, 7 million tonnes of copper/molybdenum ore and 15 million tonnes of waste rock are mined every year. Since the surface elevations range from 3,500 to 4,200 m above sea level, this is quite a challenge. Due to climatic constraints, the pit can operate only 320 days each year. This necessitates stockpiling the ore in an old pit that has two ore-pass shafts that feed an underground plant in its bottom.

17.8.3 *Geology*

The Sur Sur orebody represents a hydrothermal breccia complex. Several breccia types can be defined based upon clast size and composition, matrix and/or cement type, clast to matrix ratio, mineralization, and alteration. From an economic point of view, the most important unit is the tourmaline breccia (BXT) which includes 90% of the mineralization and defines the main part of the orebody. It has a north-south trend. The other rock types, Cascada granodiorite (GDCC), Monolito breccia (BXMN), Monolito breccia with tourmaline

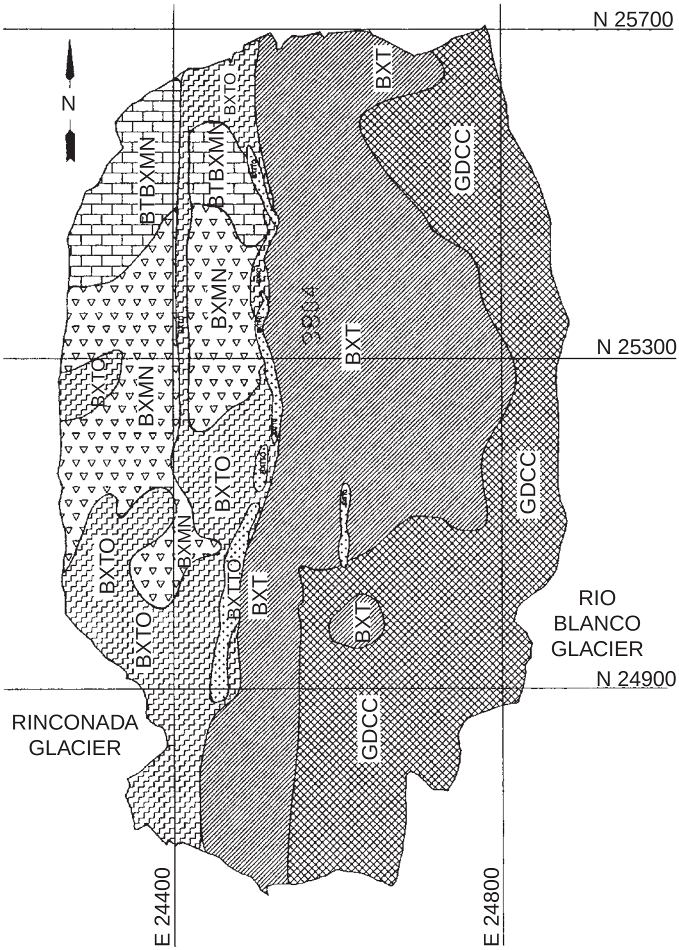


Figure 17.14. Geological units present in the Sur Sur Mine. After Apablaza et al. (2000).

(BTBXMN), rock flour breccia (BXTO), and rock flour breccia with tourmaline (BYTTO) are all located to the west of the tourmaline breccia. They contain low grades or are waste rock. The location of these rock types are shown in Figure 17.14.

17.8.4 *Structural geology*

The structural mapping includes all of the structural features whose persistence is sufficient to affect at least one bench and/or those structures associated with bench scale instabilities. The interpretation of these data made it possible to define the structural domains shown in Figure 17.15.

The rock mass is classified according to the geological strength index (GSI), which takes into account the rock-mass fabric and the quality of the structures. The typical range for the GSI is 50 to 60 but there are also sectors with lower and higher values. Figure 17.16 shows the zonation of the Sur Sur Mine in terms of the GSI. The mechanical properties of

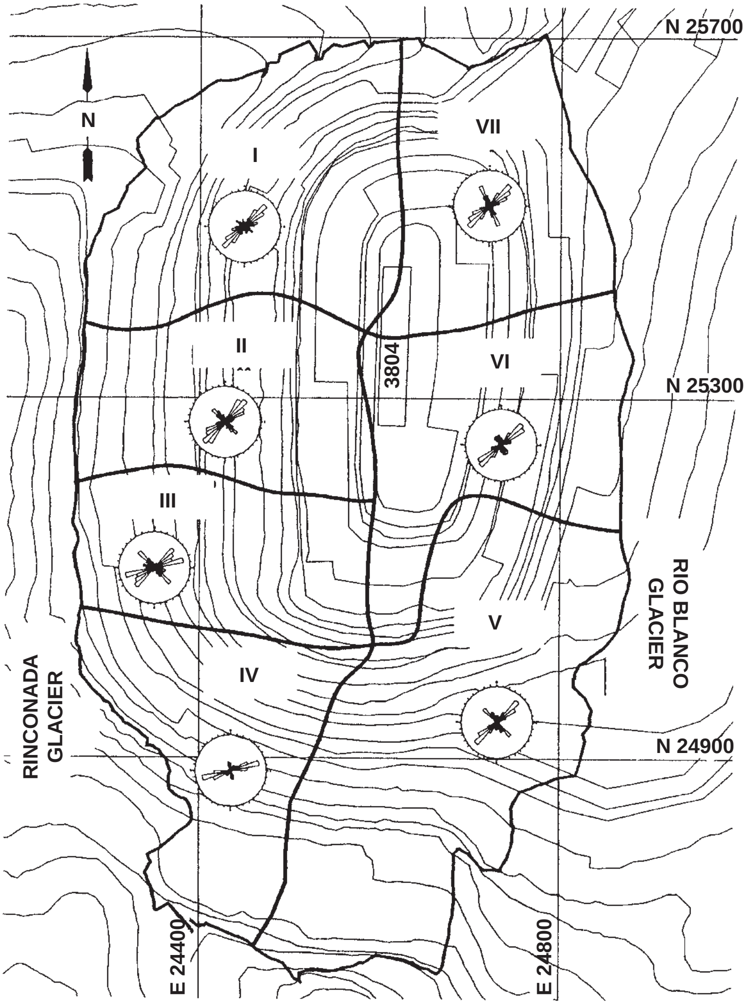


Figure 17.15. Structural domains in the Sur Sur Mine. After Apablaza et al (2000).

the different intact rock types are summarized in Table 17.16 and the typical strength of the structures is presented in Table 17.17. The density is approximately 2.6 g/cm^3.

17.8.5 *Geotechnical slope analysis and design*

The geological, structural, and geotechnical data are used to distinguish/define the different design sectors in the pit. Each sector has specific characteristics that should be considered in the geotechnical analysis and will define the slope geometries. The geotechnical analysis includes an evaluation of the possible types of slope failure, including the effect of major structures.

The bench height is defined by considering the efficiency of shovels and the volumes of eventual unstable wedges. From an operational point of view, the steeper the bench face, the

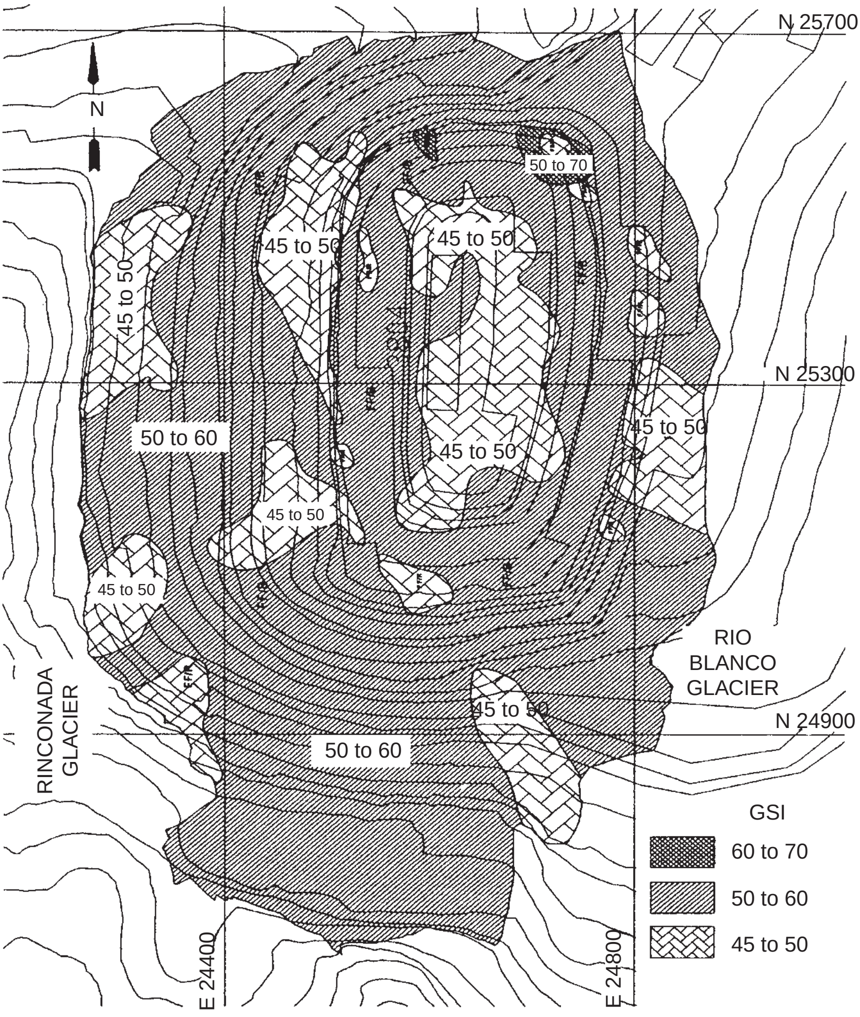


Figure 17.16. Geotechnical zonation of the Sur Sur Mine in terms of GSI. After Apablaza et al (2000).

better. The berm width is first defined using the criteria of Ritchie (Ryan and Pryor, 2000) and then checked by a probabilistic analysis of the structurally controlled instabilities that could affect the benches.

The stability of interramp and overall slopes is analyzed by two-dimensional limit-equilibrium methods. Currently, the Sur Sur pit is in a transition stage, because the current design of 12-m high single benches is being changed to 32-m high double benches, with a 70° bench face inclination, to allow a more efficient use of today's large mining equipment. This new design allows considerable savings and an important decrease in the stripping ratio because it increases the inter-ramp and overall slope angles.

The current design for the Sur Sur Mine is summarized in Table 17.18 and Figure 17.17 while the same data for the final pit condition are presented in Table 17.19 and Figure 17.18.

Table 17.16. Typical property values for the intact rock. After Apablaza et al (2000).

Rock type	γ (ton/m^3)	UCS (MPa)	TS (MPa)	E (GPa)	ν
Tourmaline breccia	2.65	114	9	30	0.20
Cascada granodiorite	2.59	115	9	41	0.24
Monolito breccia	2.55	127	10	33	0.18
Monolito breccia with tourmaline	2.60	120	9	31	0.20
Rock flour breccia	2.47	70	8	27	0.13
Rock flour breccia with tourmaline	2.56	65	8	28	0.15

Notes:
γ unit weight
UCS unconfined compressive strength
TS tensile strength
E deformability modulus
ν Poisson's ratio

Table 17.17. Typical property values for the structures. After Apablaza et al (2000).

Type of structure	Cohesion (kPa)	Angle of friction (degrees)
Major faults with clayey gouge	0 to 25	18 to 25
Continuous minor structures (bench scale)	0 to 100	30 to 45
Minor structures with rock bridges (interramp or overall scale)	100 to 500	30 to 35

Table 17.18. Current slope design at the Sur Sur mine. After Apablaza et al (2000).

Pit Sector	Bench-berm			Interramp			Overall	
	h_b (m)	α_b (degrees)	B (m)	α_r (degrees)	h_r (m)	r (m)	α_o (degrees)	h_o (m)
North	12/16	70	12.5	54	48	35	48	144
East	12/16	70	12.6	50	64	35	42	176
South	12/16	70	11.6	56	64	35	44	304
West	12/16	70	11.4	57	80	35	43	320
Moraine	12/16	70	12.0	29	80	40	–	80

Notes:
h_b bench height (single benches)
h_r interramp height
h_o overall height
α_b bench-face inclination
α_r interramp angle
α_o overall angle
b berm width
r ramp width

The acceptability criterion for interramp and overall slope design is that the factor of safety under operational conditions (i.e. dry slopes, no earthquake, and good-quality blasting) must be equal or larger than 1.3, while for a seismic condition the factor of safety must be equal or larger than 1.1.

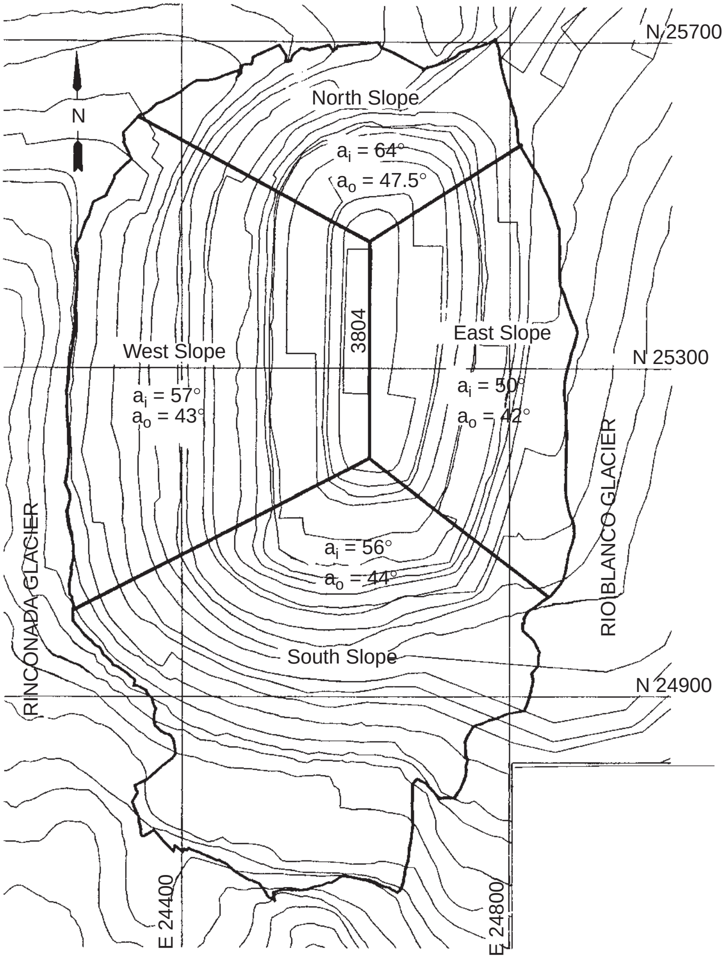


Figure 17.17. Current slope design of the Sur Sur mine using 16 m high single benches. (α_1 is the interramp slope angle, α_0 is the overall slope angle). After Apablaza et al (2000).

Every year the benches excavated in the rock glaciers need maintenance to remove the material accumulated due to the displacement of the glaciers and to recover the program lines. Currently the accumulated yearly displacement is about 20 m.

17.8.6 *Unit operations and initial costs for generating a pit*

For the Sur Sur operation, the following information applies:

Loading machines: LeTourneau Model 1400 (28 yd^3 capacity)

Material density (tonne/m^3)

	In situ	Loose
Ore	2.65	1.74
Waste	2.65	1.74
Morraine	1.51	1.19

Table 17.19. Final pit slope design at the Sur Sur mine. After Apablaza et al (2000).

Pit sector	Bench-berm			Interramp			Overall	
	h_b (m)	α_b (degrees)	B (m)	α_r (degrees)	h_r (m)	r (m)	α_o (degrees)	h_o (m)
North	32	70	12.5	53	160	35	48	300
Northeast	32	70	10.8	55	160	35	49	330
East	32	73	12.6	55	160	35	50	520
Southeast	32	70	11.6	57	160	35	45	590
Southwest	32	75	11.4	58	160	35	47	590
West	32	75	11.4	58	160	35	51	590
Moraine	16 (SB)	74	25.0	29	80	35	–	80

Notes:
h_b bench height (double benches)
h_r interramp height (maximum)
h_o overall height
α_b bench-face inclination
α_r interramp angle
α_o overall angle
b berm width
r ramp width
SB single benches

Trucks: Komatsu Model 930-E (177 tonne capacity)
Blast hole diameter: 9-7/8″
Explosive: Heavy ANFO (MEX – 150)
Blasting patterns (ore/waste):
- Hole length = 18 m
- Subdrill = 2 m
- Burden = 9.5 m
- Spacing = 11 m
- Stemming = 6 m
- Delay between holes = 25 ms
- Delay between rows = 300 ms
- Sequence = hole by hole

The interested reader is encouraged to use the cost data from the Huckleberry mine (CMJ, 2004) presented in Chapter 2 for developing an initial open pit from the Codelco Andina Copper property drillhole data.

17.9 THE CODELCO NORTE COPPER PROPERTY

17.9.1 *Introduction*

The Radomiro Tomic deposit (the Codelco Norte Copper Property) is located in the northern part of Chile about 1 km north of the world famous Chuquicamata Mine. Discovered in 1952 by the Chile Exploration Company, the property was the subject of intermittent drilling campaigns carried out over a number of years. When Codelco became the owner of the property in 1971, an integrated exploration program which would eventually lead to the

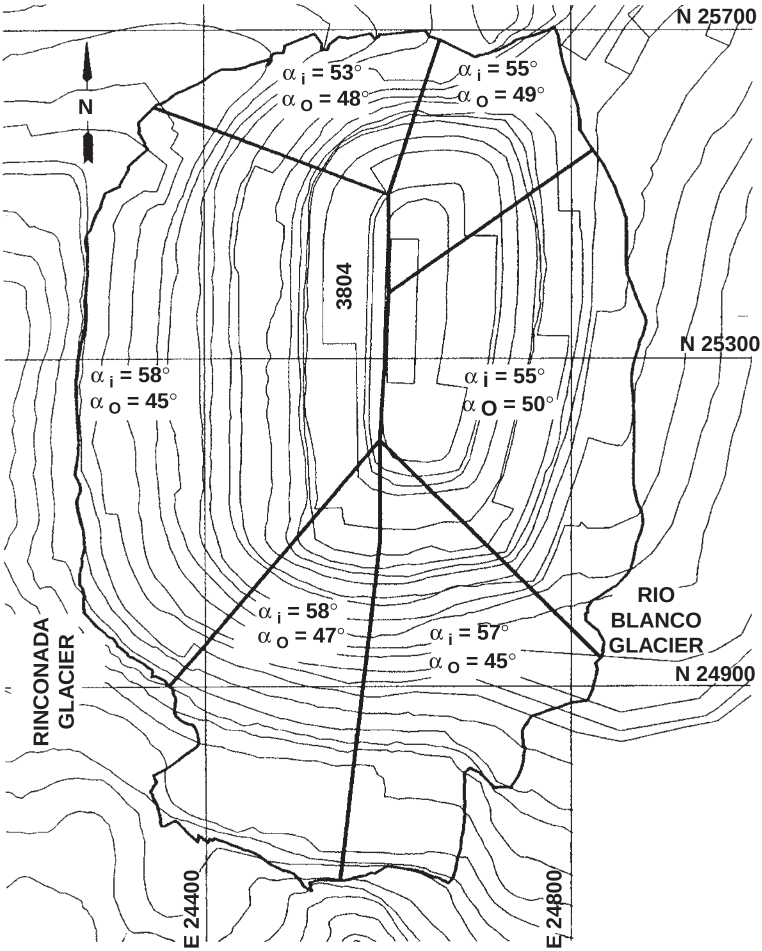


Figure 17.18. Final slope design of the Sur Sur mine using 32 m high double benches. (α_1 is the interramp slope angle, α_0 is the overall slope angle). After Apablaza et al (2000).

successful development of this world class deposit was initiated. In 1989, based upon the promising exploration results, Bechtel was commissioned to conduct the feasibility study for the Radomiro Tomic open pit/heap leach project. The results of the study were reported in mid-1992. However, during an in-depth review of the project assay data and the resulting geological model, a significant bias was discovered. This meant that the feasibility study had to be re-visited. The revised technical base for the Radomiro Tomic project together with the estimated costs was completed in May 1995. The decision was made to proceed according to the schedule shown in Table 17.20.

The Radomiro Tomic mine is a major, low-cost producer of high quality copper cathodes from its oxide ores.

The background to the development of the Radomiro Tomic mine has been presented in some detail in the excellent paper 'Estimation of resources and engineering of reserves,

Table 17.20. Radomiro Tomic project schedule.

Activity	Start date
Access road construction	December, 1995
Power transmission, temporary lines	December, 1995
Temporary water supply	December, 1995
Pre-production stripping	April, 1996
Plant area site preparation	February, 1996
Leach pad construction	January, 1998
Solutions handling system	February, 1998
Production	March, 1998
Full commercial output	May, 1998

Radomiro Tomic project, Codelco-Chile' by Behn et al (1998a, 1998b). The paper is highly unusual in that many of the engineering factors and costs required in the making of a feasibility study have been included. The present authors contacted personnel from the Codelco Norte Division of Codelco regarding the possibility of obtaining drill hole data for inclusion in this chapter. In this way, future mining engineers would be able to repeat, at least to some degree, the Radomiro Tomic feasibility study experience. The diamond drill holes completed over the 1958–1982 period were judged to provide a good, but still manageable, basis for analysis and these have been included on the distribution disk. The authors are very grateful to Codelco Norte management for their willingness to share this information and, by doing so, providing an outstanding learning experience for young mining engineers. The text material included in this section has been largely extracted directly from the paper by Behn et al (1998a, 1998b).

17.9.2 *Location and access*

The Radomiro Tomic deposit is located in Region II of Chile about 3 km north of the famous mining town of Chuquicamata and 1 km north of the Chuquicamata mine. In a broader context, it is located 20 km north of Calama, 200 km to the east of the port city of Antofagasta, and 1700 km north of the capital city of Chile, Santiago. It lies at an elevation of about 3000 m in the heart of the Atacama desert, one of the driest spots on earth. Access to the rest of the world, via air, is provided by the medium-sized Rio Loa airport at Calama. Upgraded asphalt roads connect the mine to Calama and from here there is good road access to Antofagasta, for example. The famous Chilean tourist sites of the Valley of the Moon, San Pedro de Atacama, and Chui-Chui are located nearby.

17.9.3 *Geology*

The principal geological units at the Radomiro Tomic deposit extending from the surface downward are described in Table 17.21. As can be seen, the overlying materials are cemented gravels. These are of relatively high strength and stand quite well under excavation. The primary sources of the copper oxide material being mined are the OXAR and OXAT units. The OXAR sub-unit containing primarily the copper mineral chrysocolla forms the upper oxidized unit. The OXAT mineralization which consists mostly of the mineral atacamite is generally located below the OXAR layer. It constitutes more than 60% of the entire deposit.

Table 17.21. Principal geological units.

Geological unit	Sub-unit	Description
Gravel (GR)	GRE	Sterile gravels: sedimentary deposits overlying rock. Clasts, matrix and cement without mineralization.
	GRC	Mineralized gravels: sedimentary deposits overlying rock. Residual mineralization in clasts and/or matrix. Copper grade CuT >0.2%.
	GRX	Exotic gravels: sedimentary deposits overlying rock, with exotic copper mineralization in the cement. In general with copper grades more than 0.5% CuT.
Leached Capping (LX)		Leached: undifferentiated rocks with supergene alteration, predominantly limonite. Copper grade of CuT <0.2%.
Oxides (OX)	OXAR	Clay oxides: predominant copper oxide mineralization, with copper clays (mainly chrysocolla) greater than 60% of total oxides.
	OXAT	Atacamite oxides: predominant copper oxide mineralization, with atacamite greater than 60% of the total oxides.
Mixed (MX)		Mixed: mixed oxides and copper sulfides, defined by the minimum presence of 30% (by volume) of both oxides and sulfides.
Secondary Sulfides (SS)	SSF	Secondary sulfides, strong: predominant secondary sulfide mineralization, containing at least 20% (by volume) of primary sulfides (chalcocite and covellite).
	SSD	Secondary sulfides, weak: predominant secondary sulfide mineralization, containing less than 20% (by volume) of primary sulfides (chalcocite and covellite).
Primary Sulfides (SP)		Primary sulfides: primary sulfides, containing bornite, chalcopyrite and pyrite.
Barren rocks (ES)		Waste: rock with little or no mineralization (copper, CuT<0.2%), evidenced by the lack of sulfides, copper oxides and limonites.

Most of the drilling was done using NQ size (2-in diameter) diamond bits. Sometimes the holes were started using NQ bits and completed using BX size (1.5 in diameter) bits. The cores were recovered in 1.5 m lengths. From the recovered cores, the densities for the different materials were determined. These are provided in Table 17.22.

A block model was constructed based upon an estimate of the overall resource size. The block dimensions chosen were 20 m (north-south) x 20 m (east-west) x 15 m (high). Variograms were constructed for the various geological units and these were used to assign grades to the blocks. The interested reader is referred to the source paper (Behn et al, 1998a,1998b) for a discussion of the concepts involved and the results.

17.9.4 *Geotechnical information*

Based on four directionally controlled geotechnical boreholes located in the central part of the future pit it was determined that the major structural direction for the Radomiro

Table 17.22. Dry bulk densities for the Radomiro Tomic materials.

Geological unit	Density (t/m^3)
Gravel	2.20
Leached capping	2.46
Clay oxides	2.46
Atacamite oxides	2.49
Mixed	2.50
Strong secondary sulfides	2.50
Weak secondary sulfides	2.50
Primary sulfides	2.53
Waste rock (Barren)	2.59

Table 17.23. Physical characteristics of the materials.

Factor	Parameter	Value
Gravels	In-situ density	2.26 t/m^3
	Swell factor	30%
	Bulk density (run-of-mine)	1.74 t/m^3
	Moisture content	2%
	Cohesion	13 t/m^2
	Friction angle	39°
Ore and waste	In-situ density	2.59 t/m^3
	Swell factor	30%
	Bulk density (run-of-mine)	1.99 t/m^3
	Cohesion	44 t/m^2
	Friction angle	42°

Table 17.24. Slope design criteria.

Factor	Parameter	Value
Gravels	Maximum slope angle	43°
	Maximum interramp angle	46°
	Working slope angle	39°
Ore and waste	Maximum slope angle	44°
	Maximum interramp angle	46°
	Working slope angle	43°

Tomic deposit is a northeast strike with a sub-vertical dip to the northwest. Just as at the Chuquicamata mine to the south, the West Fault is an important structural feature. The physical characteristics of the materials used in performing the slope designs for the future pit are given in Table 17.23.

The slope design criteria are given in Table 17.24.

Figure 17.19 presents a nomogram used for initial slope design. It is based on densities of 2.1 t/m^3 and 2.8 t/m^3 for the gravel and rock, respectively. The phreatic surface is assumed

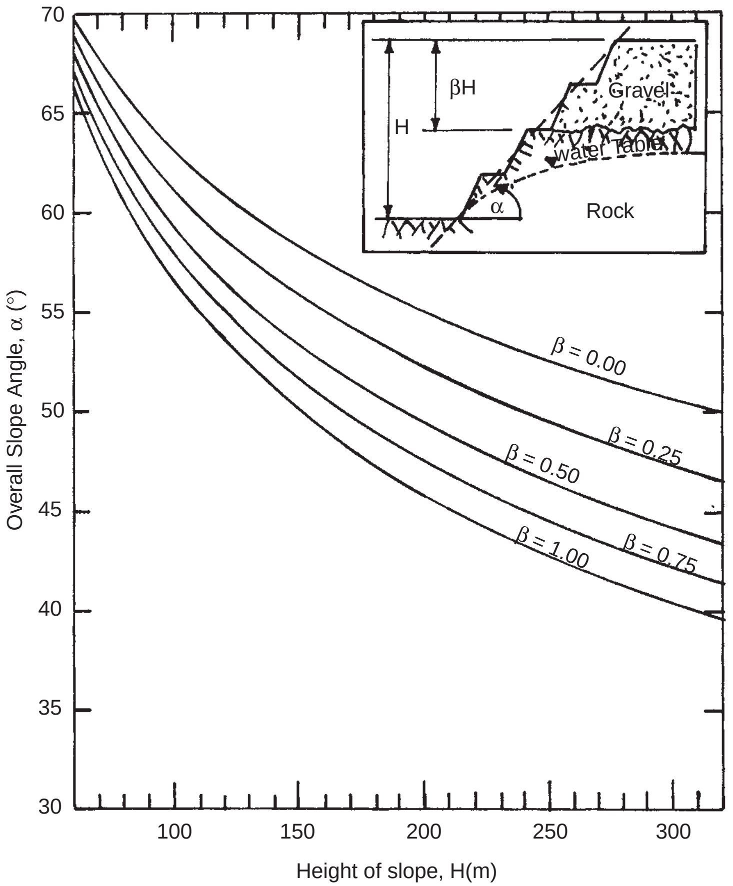


Figure 17.19. Nomogram used in the design of the Codelco Norte Copper property. Behn, et al (1998).

to pass through the toe of the slope and joins the gravel-rock interface at a distance equal to 8 x height of rock slope behind the toe. A factor of safety of 1.3 has been assumed.

17.9.5 *Open pit geometry*

Some of the geometrical factors used in making the pit design are summarized in Table 17.25.

17.9.6 *Material handling systems*

The permanent plant facilities were sited to the north of the open pit outside potentially mineralized areas. Materials handling and treatment were planned to encompass: (1) the conveying of coarse crushed material to fine crushing and acid curing facilities; (2) delivery by conveyor to the heap-leach area for automated stacking; (3) pumping of the pregnant leach solution (PLS) to the solvent extraction-electrowinning (SX-EW) plant and tank farm

Table 17.25. Open pit design criteria.

Parameters	Values
Bench height	15 m
Maximum ramp grade	10%
Angle of repose (waste)	36°
Ramp width (working pits)	28 m
Ramp width (final slopes)	28 m
Berm width (working pits)	16–18 m
Berm width (final slopes)	16–18 m
Catch bench width	32 m
Pushback width	120 m
Minimum pit bottom length	60 m
Minimum pit bottom width	40 m

located to the east of the pad area; and (4) removal of the leach residue by a bucket-wheel reclaim conveyor and thence to a dump conveyor-spreader.

The ore is transported by trucks to the primary crusher where it is crushed to about $-7''$ in size. From there, it continues to the processing plant via an overland conveyor system and is stockpiled. Six belt feeders located beneath the stockpile are used to reclaim the ore. They discharge the coarse ore onto six 54-inch wide conveyors that transport the ore from the stockpile to the secondary screening and crushing facility. Each conveyor, which is supplied with a belt scale, cross-belt magnetic separator and metal detector, is capable of transporting up to 1300 mtph of 300 mm coarse copper ore. The conveyors discharge the ore onto sizing screens. The -50 mm ore passes directly through the screens onto a 72-inch wide secondary screen product conveyor. The $+50$ mm oversize ore continues into crushers for further size reduction. The crushed ore is discharged from the crushers onto a 72-inch wide secondary crusher product conveyor. The screen product conveyor, transporting 2600 mtph of copper ore, and the secondary crusher product conveyor, transporting 4400 mtph of copper ore, discharge onto an 84-inch wide secondary crusher product collecting conveyor capable of transporting 7000 mtph. It discharges onto the first of two 84-inch wide acid-curing conveyors arranged in tandem. The acid-curing conveyors, each transporting 7000 mtph of copper ore, pass through acid washing stations where the copper ore is washed with sulfuric acid. The second of the acid-curing conveyors discharges the washed copper ore to a leach pad stacking system (FMC, 2005).

17.9.7 *Metallurgical testing/process development*

A program of metallurgical testing was conducted to specify the appropriate process design criteria for the OXAT and OXAR sub-units. The physical characteristics of the two units were first determined to assist in the design of the leaching tests (Table 17.26).

The impregnation moisture content is the maximum amount of water that can be added during the curing step. The dynamic moisture content refers to the quantity of liquid in dynamic equilibrium if pumping is interrupted.

The chemical characteristics of the two ore sub-units are given in Table 17.27.

The mineral samples were analyzed for total Cu (CuT), acid-soluble Cu (Sol Cu), chlorides, silica, and others. As seen in Table 17.27, the coarse particles contain the majority of the total and soluble copper values.

Table 17.26. Physical characteristics of the ore.

Parameter	Value
Natural moisture, $\%H_2O$	0.5–2.5
Specific gravity, t/m^3	2.43–2.59
Bulk density, t/m^3	1.36–1.52
Impregnation moisture, $\%H_2O$	6.25–7.50
Dynamic moisture, $\%H_2O$	0.72–1.56
Angle of repose, degrees	38–43

Table 17.27. Chemical characteristics of the ore.

Parameter	OXAT	OXAR
Cu T range (average), %	0.58–0.81 (0.62)	0.46–0.77 (0.57)
Total Cu distribution, %	84% $>$ 20 mesh	86% $>$ 10 mesh
Soluble Cu distribution, %	85% $>$ 20 mesh	87% $>$ 10 mesh
Chlorides, g/t	700–1560	$<$300
Silica (SiO_2), %	63–64.3	63–64.3

Leaching tests were performed using material crushed to −50 mm in both 1 m high and 8 m high columns. The objectives of the tests were:

(1) To establish the application of the curing stage to the two ore types

(2) To define the most appropriate acid dose

(3) To obtain an estimate of the metallurgical recoveries and acid consumption during leaching

(4) To evaluate the kinetics of copper extraction and acid consumption under the test conditions

(5) To determine the evolution of the copper and acid concentrations in the pregnant leach solution (PLS).

Curing of the ore with concentrated sulfuric acid prior to leaching was found to make an important difference and was selected for use in the process. The total acid consumption was higher for both units when curing was not used.

From the 8-m high column tests, it was found that for OXAT the copper recovery obtained was 85–87% after a 46-day leach. The net acid consumption was 9 kg/t and the PLS contained 8 g/l of Cu and 2 g/l of sulfuric acid. For the OXAR, the Cu recovery was 78% after 41 days. The net acid consumption of 8.2 kg/t and the PLS contained 4.4 g/l of Cu and 3.3 g/l of sulfuric acid. The fixed tails were 0.138% Cu and 0.112% Cu for the OXAR and OXAT, respectively.

The final heap design specified 8 m high heaps and a 45-day cycle time. The expected net acid consumption is 7 kg/t for OXAT and 10 kg/t for OXAR.

17.9.8 *Leach pad design and operation*

The leaching facility consists of two parallel leach pads each 300 m wide and 1300 m long, set 50 m apart to allow for the presence of a solution collection ditch. The main aspects of the design and construction of the pad foundation were (1) surface preparation, (2) the placement of the plastic membrane, and (3) the solution drainage system (installation of pipelines, placement of protective layer and deployment of drainage material). The heap pads were designed to be stacked to a height of 8 m with crushed and cured ore delivered by conveyor to a self-propelled, rail-mounted tripper car.

Irrigation piping placed on the newly constructed heap consists of a drip emitter grid connected to two parallel header pipes on 50-m centers. The drip emitter pipes are configured with emitters on a 0.5 m square grid. The flow of the leach solution is controlled by shutoff valves placed in each header pipe. These are located just in front of the attachment point to the 1 m diameter supply headers. The header pipes are taken off at right angles from the supply headers. Irrigation is designed to proceed at a nominal rate of 10 l/h/m^2 over a 45-day leaching cycle.

After leaching, the design calls for the material to be removed from the pad by a bucket wheel excavator and moved by conveyor to the disposal area. Here it will be transferred to a mobile stacker bridge conveyor equipped with a tripper car and a short-discharge conveyor to stack the leached ore in retreat mode.

The leach solution which contains 13–17 g/l H_2SO_4 and 0.6 g/l Cu is pumped to the leach supply headers. After percolating through the heap, the leach solution flows via the drainage piping and solution collection ditches into the lined PLS pond. The approximate total flow rate is 3000 m^3/hour. The PLS, containing an average of 6.5 g/l Cu and 1.5–2 g/l of H_2SO_4 at a pH of about 2.0, becomes the feed to the SX plant. After the copper has been removed, the copper-lean solution from the SX plant flows to a barren pond where acid is added prior to delivery to the heaps. Water is also added to the leaching circuit at the barren pond to make up for losses resulting from ore wetting and evaporation. The lined pond has a 12-h active capacity to allow leach operations to continue in case of interruption in the SX plant.

From the SX plant, the Cu-rich solution is pumped to the EW plant where standard cathodes (5%) and Grade A cathodes (95%) are produced.

17.9.9 *Mine design and plan*

The Radomiro Tomic deposit extends roughly 5 km in a northerly direction with an average width of about 1 km. The general mine layout encompasses a north-south oriented pit open pit with waste dumps situated parallel to the pit to the east and the west. A detailed 25-year mining plan was developed. The plan included a cutoff–grade strategy for optimization of the economics of the deposit, the ultimate pit design, the mining sequence, phases and mine plan design, equipment fleet calculations, man power estimation and the determination of capital and operating costs.

The parameters that were taken into account in determination of the cutoff grade and the ore reserves for a series of phases are summarized in Table 17.28.

Using the reserves and assuming a mining life of 25 years, the values in Table 17.29 were determined.

17.9.10 *Unit operations and manpower*

Once the ore and waste production rates have been obtained and the work system decided upon (Table 17.30), it is possible to select the equipment and determine the manpower requirements. The recommendations are provided in Table 17.31 and Table 17.32, respectively.

Table 17.28. Cutoff-grade economics.

Parameter	Value
Mining cost	$0.60/t material
Crushing, curing, leaching cost	$1.20/t ore
Annual fixed cost	$7,800,000/year
SX-EW cost	$333/t copper
Crusher capacity	36,000,000 t/year
Copper cathodes	150,000 t/year
Recovery	78.2%
Discount rate	10%

Table 17.29. Mining program.

Parameter	Value
Total reserves	802 Mt
Average grade	0.59% CuT
OXAT	620 Mt
OXAR	182 Mt
Annual Production	150,000 t cathode Cu
Mining production (ore)	90,000 tpd
Metallurgical recovery	78.2%
Design stripping ratio	1.5
Waste removal rate	135,000 tpd
Total mining rate	225,000 tpd

Table 17.30. Work system.

Factor	Value
Days per year	365
Shifts per day	2
Hours per shift	12
Operating hours per shift	10.5
Effective hours per shift (60-min hours)	9
Mechanical availability, average (related to specific units)	80%

Table 17.31. Mining equipment fleet.

Equipment	Number	Specification
Drills	3	Diameter, 274 mm; weight 45 t
Blasting	–	Contractor
Electric shovel	3	30 m^3
Truck	19 (ore)	240 t
	11 (waste)	240 t
Front-end loader	1	19 m^3
	2	9–10.5 m^3
Dozer	6	400–500 hp
Grader	4	300 hp
Service truck	2	Fuel
	2	Compressor

Table 17.32. Manpower.

Area/process	Number
Geology	13
Mining	252
Crushing and heap management	81
Solution handling and SX-EW	110
General services and administration	23
Total*	479

* Contractor services such as blasting and various maintenance and cleaning services are not included.

Table 17.33. Pre-production capital cost estimate.

Item	Estimated cost ($U.S. millions)
Mining and geology	137
Crushing facility and heap construction	144
Solution handling and SX-EW plant	131
Infrastructure and indirects	55
EPCM* and owner's costs	123
Contingency	51
Total	641

* Engineering, Procurement, Construction, and Management.

Table 17.34. Operating cost estimate.

Item	U.S. cents/lb fine copper	
Mining and geology	14.73	
Crushing and grinding	4.44	
Solution handling and SX-EW plant	15.87	
General services and administration	2.32	
Sub-total		37.36
Sales cost and marine transport	2.13	
Direct costs		39.49
Depreciation	10.87	
Total cathode costs		50.36

Table 17.35. Economic results before tax (Copper price = $1.00/lb).

Economic factors*	Result
Net present value	$635 million
Internal rate of return	28.5%
Payback period	2.99 years
Breakeven price	63.8 U.S. cents/lb

* Based on an exchange rate of 410 Chilean pesos/$U.S. The July 1997 rate was Ps 414/$U.S.

17.9.11 *Economic analysis*

A complete economic analysis was performed as part of the overall feasibility study. The capital and operating cost estimates are given in Tables 17.33 and 17.34 based upon the foregoing studies and information.

The results of a discounted cash-flow analysis given in Table 17.35 indicate that the Radomiro Tomic project would be economic (to say the very least).

REFERENCES

17.2 THE ARIZONA COPPER PROPERTY

Anonymous. 1991. La Escondida. E/MJ. 192(10): 29–31.
Bernard, G.M. 1995. Rio Algom's Cerro Colorado begins copper heap-leach operation in Chile. Mining Engineering. 47(4): 323–326.
Couzens, T. 1990. Drillhole data from the CSMine Arizona Copper property, Miami-Globe District, Arizona. Personal communication.
Dufrense, M.W. 2000. The Collahuasi Copper Project, Chile. CIM Bulletin 93(1039): 25–30.
Edwards, B., and J. Ballantyne. 1998. ALPM at Asarco's Mission Complex. E/MJ. 199(5): 62–64.
Gonzoles, I., and M.R. Nees. 1996. Integrating the Toquepala and Cuajone SX/EW operations. Mining Engineering. 48(10): 26–30.
Hardwick, W.R., and M.M. Stover 1960. Open-pit copper mining methods and practices, Copper Cities Division, Miami Copper Company, Gila County, Arizona. U.S. Bureau of Mines of Mines IC: 7985.
Hickson, R.J. 1996. El Abra: World's largest SX/EW mine on track to join copper mining elite. Mining Engineering. 48(2): 34–40.
Hickson, R.J. 1996. El Abra: On track to join copper-mining elite. Mining Engineering. 48(3): 43–47.
Jenkins, J.G. 1994. Copper heap leaching at San Manuel. Mining Engineering. 46(9): 1094–1098.
National Oceanic and Atmospheric Administration 1992. Weather conditions at meteorological stations in the U.S. Contained on CD ROM′ – National Economic, Social, and Environmental Data Bank (NS CLIMAT 025512), Nov.
Peterson, N.P. 1954. Globe quadrangle, Arizona: U.S. Geol. Surv. Geol. Quad. 41 (1" = 2000').
Peterson, N.P. 1954. Copper Cities Copper Deposit, Globe-Miami District, Arizona. Economic Geology 49: 363–377.
Phelps, R.W. 1998. Radomiro Tomic: A new plant grows in the Atacama Desert. E/MJ. 199(3): 28–31.
Ransome, F.L. 1903. Geology of the Globe copper district, Arizona. U.S. Geol. Surv. Prof. Paper 12. 168 pages (particularly pages 114–163).
Ransome, F.L. 1919. The copper deposits of Ray and Miami, Arizona. U.S. Geol. Surv. Prof. Paper 115. 192 pages (particularly pages 107–121 and 144–176).
Ryan, P.J., and J.L. Madson. 1996. Candelaria – low cost copper-gold producer in Chile (Part 1). Mining Engineering. 48(8): 35–41.
Ryan, P.J., and J.L. Madson. 1996. Candelaria – low cost copper-gold producer in Chile (Part 2). Mining Engineering. 48(9): 44–51.
Uhrie, J.L., and G.J. Koons. 2001. Evaluation of deeply ripping truck-dumped copper leach stockpiles. Mining Engineering. 53(12): 54–56.

17.3 THE MINNESOTA NATURAL IRON PROPERTY

Faust, H.H. 1954. Report on the Holden Property for Boulder Mining Company, Boulder, New York. Senior Thesis, Department of Mineral Engineering, University of Minnesota. 59pp.
Grout, F.F., and J.F. Wolf. 1955. The geology of the Cuyuna District, Minnesota: A progress report. Bulletin No. 36, Minnesota Geological Survey. University of Minnesota. 139pp.

Harder, E.C., and A.W. Johnston. 1918. Preliminary report on the geology of east central Minnesota including the Cuyuna iron-ore district. Bulletin No. 15, Minnesota Geological Survey. University of Minnesota. 178pp.

Himrod, A. 1940. The Cuyuna range, a history of a Minnesota iron mining district. Minnesota Historical Records Survey Project. WPA. St. Paul, Minnesota. 168pp.

Leith, C.K. 1907. The geology of the Cuyuna Iron Range. Economic Geology. 2: 145–152.

Minnesota Division of Natural Resources. 1999. Maps and core logs of the unidentified iron resource. Personal communication.

Pfleider, E.P., and G.F. Weaton.1968. Iron ore mining. Section 13.4 in Surface Mining. (E.P. Pfleider, editor). AIME, New York. pp. 897–927.

Schmidt, R.G. 1958. Titaniferous sedimentary rocks in the Cuyuna District, Central Minnesota. Economic Geology. 53: 708–721.

Schmidt, R.G. 1963. Geology and ore deposits of the Cuyuna North Range, Minnesota. Geological Survey Professional Paper 407. U.S. Government Printing Office. 96pp.

Schwartz, G.M., and G.A. Theil. 1963. Minnesota's Rocks and Waters. The University of Minnesota Press. Minneapolis. 366pp.

Theil, G.A. 1927. Geology of the Cuyuna Range. Geol. Soc. Amer. Bull. 38: 785–793.

Van Barneveld, C.E. 1913. Iron Mining in Minnesota. Bulletin No. 1. Minnesota School of Mines Experiment Station. The University of Minnesota.

17.4 THE UTAH IRON PROPERTY

Ashcroft, G.L., R.C. Brough, and J.L. Brown. 1992. Utah Climate. Utah Climate Center, Utah State University, Logan, Utah.

Bellum, D.P., and L. Nugent. 1963. Iron mining, healthy Utah industry. Mining Engineering. 16(11): 34–38.

Geneva Steel. 1992. Iron Mines, Southwest Cedar City, Utah, Contour Map.

Guilbert, J.M., and C.F. Park, Jr. 1986. The Geology of Ore Deposits. New York: W.H. Freeman and Company. pp. 460–465.

Jones, L. 2001. Maps and core logs from the Utah Iron property. Geneva Steel, Provo, Utah. Personal communication.

Koski, A.E. 2000. Mine planning software models Empire Mine ore. Mining Engineering 52(12): 47–52.

Krall, C. 2004. Technical and economic evaluation of the Jackrabbit Prospect, Iron Springs District, Utah. Senior Thesis, Department of Mining Engineering, The University of Utah.

Langston, L.P. 1997. Iron County, A Demographic and Economic Profile. Utah Department of Workforce Services, Sept.

Larson, G.O. 1963. Bulwark of the Kingdom. Utah Historical Quarterly. v31. pp. 248–266.

Leith, C.K., and E.C. Harder. 1908. The Iron Ores of the Iron Springs District.United States Geological Survey, Bulletin 338.

Mackin, J.H., and J.D. Ridge, ed. 1968. Iron Ore Deposits of the Iron Springs District, Southwestern Utah. Chapter 49 in Ore Deposits of the United States, 1933–1967. The American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc. New York. pp. 992–1019.

Pope, D., and C. Brough, ed. 1996. Utah's Weather and Climate. Salt Lake City: Publishers Press.

Seegmiller, J.B. 1998. Mining. Chapter 18 in A History of lron County. Salt Lake City: Utah State Historical Society. pp. 320–350.

Utah Department of Natural Resources, Division of Water Rights. 1999. Utah's Water Right Process.

U.S. Geological Survey. 1982. Cedar City, Utah, Topographic Map.

U.S. Geological Survey. 1907. Geologic Map and Sections of Iron Springs Quadrangle, Utah.

Weather Channel Online. 28 Nov. 2003 <www.weather.com>.

'What's a Laccolith?' 15 Nov. 2003. <formontana.net/2a.html>.

Wilberg, D.E., Tibbetts, J.R., and M. Enright. 2002. Water Resources Data Utah Water Year 2002. Water Data Report UT-02-l. United States Geological Survey.

Young, W.E. 1947. Report of Investigations, Iron Deposits, Iron County, Utah. U.S. Bureau of Mines R.I. 4076, May.

17.5 THE MINNESOTA TACONITE PROPERTY

Anonymous. 2003. Minnesota Mining Tax Guide. Minnesota Department of Revenue. September.
Bello, M. 2005. Technical and Economic Evaluation of the Biwabik Taconite Operation, Mesabi Iron Range, Minnesota. Senior Thesis, University of Utah, Department of Mining Engineering.
Brinsmade, E. 2002. 2002 Minnesota Water Quality: Surface Water Section. Report to the Congress of the United States, Water Years 2000–2001. Minnesota Pollution Control Agency.
Grout, F.F., and T.M. Broderick. 1919. The magnetite deposits of the eastern Mesabi Range, Minnesota. Bulletin No. 17, Minnesota Geological Survey. University of Minnesota. 58pp.
Gruner, J.W. 1946. Mineralogy and Geology of the Mesabi Range. Iron Range Resources and Rehabilitation. St. Paul, Minnesota.
Gruner, J.W. 1946. The mineralogy and geology of the taconites and iron ores of the Mesabi Range, Minnesota. Office of the Commissioner, Iron Range Resources and Rehabilitation. 127pp.
Hedin, R.S. 2002. Recovery of marketable iron oxide from mine drainage. 2002 West Virginia Surface Mine Drainage Task Force Symposium in Morgantown, WV. April 16–17.
Holgers, R.H., Baxter, W., and W.G. Fetzer. 1969. Thunderbird Mine of Eveleth Taconite Company. In Case Studies of Surface Mining, editor H.L. Hartman. Proceedings of the II International Surface Mining Conference, Minneapolis, Minnesota. September 18–20, 1968. AIME. pp. 67–83.
Jirsa, M. 2005. Minnesota Geological Survey. Personal communication.
Jorgenson, J.D. 2004. Iron Ore in June 2004. Mineral Industry Surveys. U.S. Geological Survey.
Kirk, W.S. 2002. Iron Ore. U.S. Geological Survey Minerals Yearbook 2002. U.S. Geological Survey.
Ongaro, F. 2003. Iron Mining industry seeks substantial production tax reduction. Iron Mining Association of Minnesota. March 3.
Paulson, A. 2004. Surprise revival for iron mines of Minnesota. The Christian Science Monitor. April 22.
Ramsay, C. 2003. Plant startup boosts taconite production total. Mesabi Daily News, December 27.
Schmucker, T.W. 2003. Drillhole data for a small, unmined taconite deposit. Minerals Tax Office. Minnesota Department of Revenue. Eveleth, Minnesota. Personal communication.
Vanderlinder, P. 2005. Personal communication.
White, D.A. 1954. The stratigraphy and structure of the Mesabi Range, Minnesota. Bulletin No. 38, Minnesota Geological Survey. University of Minnesota. 92pp.

Internet References:
www.taconite.org
www.topozone.com
www.city-data.com
www.cityofbiwabik.com

17.6 THE KENNECOTT BARNEYS CANYON GOLD PROPERTY

Barr, D.S., and T. Acton. 1998. Heap leaching at Newmont's Muruntau operation. Mining Engineering. 50(1): 39–44.
Bernard, G.M. 1996. Andacollo gold production – ahead of schedule and under budget. Mining Engineering. 48(8): 42–47.
Carter, R.A. 1997. Fort Knox banks on a golden future. E/MJ. 198(8): 34–37.
Carter, R.A. 1997. Florida Canyon plays a new hand. E/MJ. 198(6): 38–43.
Gottling, R. 2005. Kennecott Barneys Canyon mine. Personal communication.
Harris, L. 1996. Newmont's Yanacocha project: The joint venture three years later. Mining Engineering. 48(2): 41–47.
Jacobsen, A., and A. Lowe. 2003. The Oquirrah Mountain Gold Prospect: Technical and Economic Analysis. Senior Thesis, Department of Mining Engineering, The University of Utah.
Johnson, M.D., Bikerman, D., and M.Russell. 1996. Mine development modeling at Mina Santa Rosa in Panama. Mining Engineering. 48(4): 51–57.
Kennecott. 2002. Barneys Canyon Mine Brochure.
Marcus, J. 1997. The Briggs Mine: A new heap leach mine in an environmentally sensitive area. E/MJ. 198(9): 16GG–16VV.

O'Neil, T. 2001. Cresson Mine: expands its operations. Mining Engineering. 53(10): 21–28.
Pease, R.C., and W.S. Watters. 1996. San Juan Ridge gold mine begins production. Mining Engineering. 48(12): 26–31.
Peterson, E.U. 2005. Melco-Barneys Canyon Mine. www.mines.utah.edu/pyrite/cmes-seg/fieldtrips/melco2000/index.htm.
Phelps, R.W. 1994. Santa Fe Pacific gold: No derailment involved in its independence day – September 30. E/MJ. 195(7): 28–40.
Pontius, J.A., and J.A. Head. 1996. Cresson Mine: Case history of a rapidly developing mining project. Mining Engineering. 48(1): 26–30.
Presnell, R.D. 1992. Geology and geochemistry of the Barneys Canyon gold deposit, Salt Lake County, Utah. University of Utah. Unpublished PhD dissertation. 363pp.
Presnell, R.D., and W.T. Parry. 1996. Geology and geochemistry of the Barneys Canyon gold deposit, Utah. Economic Geology, 91: 273–288.
Slothower, B.H. 2001. Tour of the Kennecott Barneys Canyon mine. Personal communication.
Suttill, K.R. 1991. Maricunga – The world's next great gold province? E/MJ. 192(10): 33–37.
Switzer, E. 2001. Drillhole data for the Kennecott Barneys Canyon mine. Kennecott Barneys Mining Company. Personal communication.
Yernberg, W.R. 1996. Santa Fe Pacific Gold targets million ounces/year production. Mining Engineering. 48(9): 39–43.
Yernberg, W.R. 2002. Cresson Mine completes two-year expansion. Mining Engineering. 54(12): 22–23.

17.7 THE NEWMONT GOLD PROPERTY

Barr, D.S., and T. Acton. 1998. Heap leaching at Newmont's Muruntau operation. Mining Engineering. 50(1): 39–44.
Bernard, G.M. 1996. Andacollo gold production – ahead of schedule and under budget. Mining Engineering. 48(8): 42–47.
Carter, R.A. 1997. Fort Knox banks on a golden future. E/MJ. 198(8): 34–37.
Carter, R.A. 1997. Florida Canyon plays a new hand. E/MJ. 198(6): 38–43.
Harris, L. 1996. Newmont's Yanacocha project: The joint venture three years later. Mining Engineering. 48(2): 41–47.
Johnson, M.D., Bikerman, D., and M. Russell. 1996. Mine development modeling at Mina Santa Rosa in Panama. Mining Engineering. 48(4): 51–57.
Marcus, J. 1997. The Briggs Mine: A new heap leach mine in an environmentally sensitive area. E/MJ. 198(9): 16GG–16VV.
Miller, Joe. 2001. Private communication.
O'Neil, T. 2001. Cresson Mine: expands its operations. Mining Engineering. 53(10): 21–28.
Pease, R.C., and W.S. Watters. 1996. San Juan Ridge gold mine begins production. Mining Engineering. 48(12): 26–31.
Phelps, R.W. 1994. Santa Fe Pacific gold: No derailment involved in its independence day – September 30. E/MJ. 195(7): 28–40.
Pontius, J.A., and J.A. Head. 1996. Cresson Mine: Case history of a rapidly developing mining project. Mining Engineering. 48(1): 26–30.
Price, J.G. 2002. Remarks made on the occasion of the 50 millionth ounce of gold produced from the Carlin trend, Nevada – May 16, 2002. www.nbmg.unr.edu/slides/jon.htm.
Suttill, K.R. 1991. Maricunga – The world's next great gold province? E/MJ. 192(10): 33–37.
Wengel, C.S., and A.A. Smith. 2001. Evaluation of the Quasar Project for S & W Mining Company. Senior Thesis, Department of Mining Engineering, the University of Utah.
Yernberg, W.R. 1996. Santa Fe Pacific Gold targets million ounces/year production. Mining Engineering. 48(9): 39–43.
Yernberg, W.R. 2002. Cresson Mine completes two-year expansion. Mining Engineering. 54(12): 22–23.
http://silver.state.nv.us, 2001.
www.kitco.com, 2001.
www.topozone.com, 2001.

www.amerisar.org/nvmap.htm, 2001.
www.nv.blm.gov, 2001.
www.brrc.unr.edu/data/animals, 2001.
www.wrcc.dri.edu, 2001.
www.the-privateer.com, 2001.
www.census.gov, 2001.
www.ocs.orst.edu, 2001.

17.8 THE CODELCO ANDINA COPPER PROPERTY

Apablaza, R. 1999. Drillhole data for the Chile Copper property, Andina Division, Codelco. Personal communication.

Apablaza, R., Farias, E., Morales, R., Diaz, J., and A. Karzulovic. 2000. The Sur Sur Mine of Codelco's Andina Division. Chapter 19 in Slope Stability in Surface Mining (W.A. Hustrulid, M.K. McCarter, and D.J.A. Van Zyl, editors). SME. pp. 171–176.

CMJ. 2004. Operating data from the Huckleberry mine. 2004 Sourcebook.

Hoek, E. 1998. Reliability of Hoek-Brown estimates of rock mass strength properties and their impact on design. Intl. Jour. Rock Mech. Min. Sci. 35(l): 63–68.

Hoek, E., and J.W. Bray. 1981. Rock Slope Engineering. London: IMM.

Hoek, E., and E.T. Brown. 1997. Practical estimates of rock mass strength. Intl. Jour. Rock Mech. Min. Sci. 34(8): 1165–1186.

Hoek, E., Kaiser, P.K., and W.F. Bawden. 1995. Support of Underground Excavations in Hard Rock. A.A. Balkema.

Ryan, T.M., and P.R. Pryor. 2000. Designing catch benches and interramp slopes. Chapter 3 in Slope Stability in Surface Mining (W.A. Hustrulid, M.K. McCarter, and D.J.A. Van Zyl, editors). SME. pp. 27–38.

17.9 THE CODELCO NORTE COPPER PROPERTY

Behn, R.G., Beniscelli, J., Carrasco, C.P., Cuadra, P., and G.A. Ferguson. 1998. Estimation of resources and reserves, Radomiro Tomic project, Codelco-Chile. Proceedings, APCOM '98. IMM, London. pp. 241–64.

Behn, R.G., Beniscelli, J., Carrasco, C.P., Cuadra, P., and G.A. Ferguson. 1998. Estimation of resources and reserves, Radomiro Tomic project, Codelco-Chile. Trans. Instn. Min. Metall. (Sect. A: Min. Industry). 107(May–August): A83–A101.

Calderon, A. 2005. Personal communication.

Chuquicamata, Chile. www.fallingrain.com/world/CI/3/Chuquicamata.html

Codelco. 2005. The Corporation Divisions – Codelco Norte. www.codelco.com/english/la_corporacion/division_norte.asp

FMC Technologies. 2005. Radomiro. www.fmctechnologies.com/MaterialHandling/BulkConveyingAndProcessingEquip

Phelps, R.W. 1998. Radomiro Tomic: A new plant grows in the Atacama Desert. E/MJ. 199(3): pp. 28–31.

Rio Loa, Chile. www.fallingrain.com/world/CI/3/Chuquicamata.html

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Photograph: BINGHAM CANYON mine. Courtesy of Kennecott Utah Copper.

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