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Elements of Mining Technology Vol. 1

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By D. J. Deshmukh

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Abbreviations used :

g - gramme; te - tonne; km. - kilometre; h or hr - hour; m - metre; kg - kilogramme.

Note - no 'S' is added for plural

CONVERSION TABLE

Inches × 25.4	=	Milimetres.	Milimetres × .039	=	Inches.
Feet × .305	=	Metres.	Metres × 3.28	=	Feet.
Yards × .914	=	Metres.	Metres × 1.09	=	Yards
Miles × 1.609	=	Kilimetres.	Kilometres × .621	=	Miles
Lbs. per Sq. inch × .0703	=	kg/cm^2	Kg/cm^2 × 14.2	=	Lbs. per Sq. inch.
Cft. × .0283	=	Cubic metres	Cub. metres × 35.31	=	Cubic feet
Cyd. yards × .764	=	Cubic metres	Cub. × 1.307	=	Cubic
Pounds × .453	=	Kilograms	Kilograms × 2.2	=	Pounds
Ounces × 28.35	=	Grams.	Grams. × .035	=	Onnces.
Sq. ins. × 6.451	=	Sq. cms.	Sq. cms. × .155	=	Sq. inches
Sq. ft. × .092	=	Sq. metres.	Sq. metres × 1.195	=	Sq. feet
Sq. yards × .836	=	Sq. metres.	Sq. metres × 1.195	=	Sq. yards.
Sq. miles × 2.59	=	Sq. kilometers	Sq. kilo- metres × .386	=	Sq. miles
Imp. gallons × 4.54	=	Litres.	Litres × .219	=	Gallons (imp)

Long ton (2,240 lbs) × 1.016

= Metric tonne.

S.I. Units

1 kgf = 9.81 Newtons

1 bar or b = 10^5 N/m^2

1 kgf/cm^2 = 98.1 kN/m^2

1 lbf/in^2 = 6.895 kN/m^2

1 H.P = 746 Watts

Gradient

Degress	One in
8	7.12
10	5.67
16	3.49
20	2.75
30	1.73
45	1.00
60	0.58

ELEMENT OF MINING TECHNOLOGY

By D.J. deshmukh

Contents of other two volumes is given below.

VOLUME 2

PART-A

PART-B

VOLUME 3

Chapter -1

Mining Geology

Minerals, Rocks & Rock Structures

What is mining ? "Mining is the process of excavating minerals of economics value from the earth's crust for benefit of mankind". Here we can assume that the earth's crust, the outer surface of the earth, including the oceans, lakes and rivers, extends to depths of 30 to 50 km or so. For mining operations one should have a working knowledge of geology.

The word *Geology means science* of the earth and deals with the nature and origin of the rocks that constitute the earth. A person interested in the extraction of minerals from the earth is, however, concerned with the thin surface of the rocks which make up the earth's crust for a depth of a maximum of 5 km so that geology enables him to locate and to decide the sites most economic for mining or quarrying.

In geology the terms *mineral* and *rock* have precise but different meanings. A *mineral* is a homogeneous and naturally occuring substance having definite physical properties and a composition that may be expressed by a chemical formula. The chemical composition of a mineral, as found in the earth, may be the same as that of an artificially prepared chemical compound in the laboratory but the physical characteristics may differ; e.g. lead sulphide, PbS, is generally available in the laboratory as an amorphous powder. It has quite different physical properties from galena, Pbs, the name of the naturally occuring mineral, often in the crystalline form. Some few minerals occur as single elements, e.g. native gold., silver, graphite, but most minerals are composed of two or more elements in chemical composition, e.g. quartz (SiO_2), hematite (Fe_2O_3), etc. Most rock-forming minerals are oxides, chlorides, sulphides, carbonates, sulphates or silicates. A *rock* may be composed of one mineral only, but is usually a mechanical mixture or aggregate of two or more minerals. For example, granite is a rock composed essentially of three separate minerals; quartz, felspar and mica. Whereas minerals can be considered as aggregates of chemical elements, so rocks are really aggregates of minerals. They contain minerals in varying proportions and they have no definite chemical composition.

Bed rock is any rock lying in the position in which it was formed; it is therefore not broken up.

Country rock of an orebody is that rock which is predominat in the area and which contains the orebody. The country rock forms the footwall and the hanging wall.

A *seam* is a mineral deposit limited by two, more or less parallel planes, a shape which is typical of sedimentary rocks. The term is generally used for coal, e.g. a coal seam.

When excavating a useful mineral, the uneconomic rock or mineral associated with it which has to be excavated and discarded is called rejection, dirt or waste in coal mining practice and *gangue* in metal mining practice.

An *ore* is a rock which contains mineral and which can be used for economical extraction of metal after processing to separate mineral from gangue. Ores usually occur in veins or lodes.

If an ore, when subjected to metallurgical processes, yields only one metal, it is called a *straight ore.*

A *mineral deposit* is a rock or mineral that is of economic value and repays it extraction from the earth.

A *vein* (or *lode)* shown in Fig. 1.1. is a crack in the earth's crust filled with mineral. This filling can occur by precipitation of the mineral from the mineral-rich water or by the cooling of the magma filling the crack or by the separation of the mineral from vapours and gases rising up the crack. Veins, like seams, have a strike, a dip and a thickness but for the same vein, all these are usually quite variable.

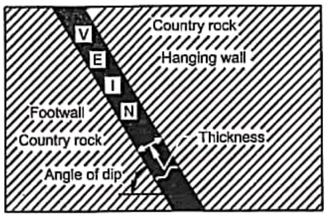


Fig. 1.1

A number of nearly parallel veins constitute *compound lodes.*

Orebody is the part of a vein that carries the ore. Generally all parts of a vein are not ore.

Note that :

the country rock contains the vein;
the vein contains the orebody;
the orebody contains the ore;
the ore contains the minerals;
the mineral contains a metal or metals.

MINERALS

Minerals possess definite physical properties by virtue of which they can be distinguished from one another. The most important physical properties are as follows :

Colour : Some minerals possess a characteristic colour, e.g. galena, magnetite, olivine, etc; but in some others the colour is variable, e.g. quartz.

Specific gravity : Most rock-forming minerals have a specific gravity between 2 and 4.

Lustre : The lustre may be metallic (like galena or iron pyrites), pearly (like talc) or silky.

Taste and smell : Rock salt, alum and some other minerals can be recognised by their taste.

Streak : A few minerals, when drawn over paper or over an unglazed procelain plate, leave a coloured mark known as the streak; for example, graphite gives a black streak; hematite leaves a cherry red streak.

Crystalline : A crystal is geometrical solid bounded by smooth plain surfaces called faces and capable of increasing in size by the deposit of fresh material on the outside of these surfaces. The faces in a crystal show a definite geometrical pattern and the angles between the faces are constant; for example, quartz crystallises in the hexagonal system, while mica or muscovite crystallises in what is called the monoclinic system, and rock salt, in the cubic system. The crystallisation may take place by : (a) deposition from solution (b) slow cooling from the molten state, or (c) direct change from a vapour to a solid.

Cleavage : Many crystals have tendency to split along one or more direction parallel to an actual or possible crystal face. This splitting gives plane surfaces known as cleavage planes. For example, mica cleaves in one direction only; galena (lead sulphide) cleaves in three planes at right angles, forming perfect cubes.

Fracture : When a crystal breaks independently of the cleavage plane, it is said to fracture. The property is prominent in minerals with poor cleavage.

Hardness : This term gives the relative ease with which minerals can be scratched. In practice hardness is measured by reference to a set of minerals given below so arranged that the first member can be scratched by all the others, the second by all except the first, and so on.

Moh's scale of hardness is as follows :

1.	Talc	
2.	Gypsum	Scratched by finger nail
3.	Calcite	
4.	Fluorspar	Scratched by a knife
5.	Apatite	
6.	Orthoclase	Scarcely scratched by a knife.
7.	Quartz	
8.	Topaz	
9.	Corundum	Not scratched by a knife.
10.	Diamond	

It may be observed that :

A finger nail will scratch upto about 2.5.

A pen knife will scratch upto 6.5

When testing the hardness of mineral window glass can be used as a substitute for apatite.

Protodyakonov strength number : Hardness of rock is expressed by Protodyakonov strength number in Russia. The number indicates the relative ease with which a rock can be broken, e.g. strong lignites and weak clay shales have Protodyakonov strength number as 1.5 to 2; strong colas and anthracites have strength number as 2; exceedingly strong quartzites and gabbro-diorites have the number as 20-25, the highest number. Other rocks have the numbers inbetween.

Electrical and magnetic properties of minerals and the properties dependent on light are also made use of in distinguishing minerals which react in a distinctive manner to the tests.

Common Minerals

There are about 107 elements that have been isolated and recognised in the laboratory. Of this number, however, there are only 8 that enter into the composition of the earth's outer portions in abundance. In fact these 8 elements make up some 98% of the earth's observable crust. These are (in order of abundance) :

O_2	–	47%	Ca	–	3.5%
Si	–	28%	Na	–	2.5%
Al	–	8%	K	–	2.5%
Fe	–	5%	Mg	–	2.0%
				Total	**98.5%**

The combinations of some of these eight common elements among themselves have produced the most common rock-forming minerals that constitute the bulk of the rocks. These most common rock-forming minerals are feldspars, quartz, mica, amphiboles, pyroxenes, and olivine.

The other 96 elements are relatively scarce, in that they represent only 1.5% by weight of the earth's crust *e.g.* Cu - 0.0045%, Pb - 0.00015%, Au - 0.0000007%.

There are about 2000 catalogued mineral specimens but the real economic targets of mining activity are 100 minerals including the native minerals, the hydro carbon minerals and a few types of economic rocks used as house construction materials.

Of the above rock-forming minerals some can be considered as essential minerals while some others are accessory minerals and secondary minerals.

(a) *Essential Minerals :* These make up the bulk of the rocks and are always silicates with the exception of quartz and the cabonates.

(b) *Accessory Minerals :* These are present only in small quantities in a rock.

(c) *Secondary Minerals :* These are derived from the break-down of the others.

Classification Of economicminerals :

(a) ***Metallic minerals* : Minerals that yield metals :**

Precious metals : gold, silver, platinum.

Base metals : Copper, lead, zinc, tin.

Steel industry metals : iron, nickel, chromium, manganese, cobalt, molybdenum, tungesten, vanadium, tatalum, etc.

Light metals : aluminium, magnesium, titanium, etc.

Electronic industry metals : Cadmium, bismuth, germanium, mercury, selenium.

Radioactive metals : Uranium, radium, caesium, zirconium, beryllium, rare earths, etc.

(b) ***Non-metallic minerals* :**

Insulating materials : Mica, asbestos, silimanite.

Refractory materials : Silica, alumina, zircon, graphite, etc.

Abrasives : Corundum, emery.

Gems : Garnet, diamond, topaz, emerald, saphire, etc.

General industrial minerals : Phosphate rock, lime stone, rock-salt, baryte, borates, felspars, magnesite, gypsum, potash, clays, sulphur.

(c) *Fuel minerals :*

Solid fuels : Anthracite, coal, lignite, oilshale.

Liquid fuels : Petroleum oil natural gas.

The common rock-forming minerals are described below in brief.

Silicates

***The Felspars* :** These are all complex silicates of aluminium with potash, soda or lime and are most abundant in igneous rocks. Potash felsphar, also known as orthoclase, is the most common type. It is white, grey or pink in colour with a glassy lustre. When weathered it leaves a hydrated silicate of aluminium known as kaolin or china clay. The soda and lime felspars are known as plagioclase. They also decompose and disintegrate in a similar way. Plagioclase felspars occur in most igneous rocks especially the darker varieties rich in lime (sp. gr = 2.6 to 2.7 ; H = 6). H means hardness.

***Mica* :** This occurs as white variety known as muscovite (a silicate of aluminium and potassium) and as black variety known as black mica or biotite (a silicate of aluminium, iron and magnesium). The mica can be easily scratched by a finger nail and has a well-developed cleavage. It is a common constituent of igneous rocks and crystalline schists (Sp. gr = 2.7 to 3.1 ; H = 2.5).

***Hornblende and Augite* :** These minerals are complex silicates of calcium, magnesium and iron. They are both greenish black in colour. Hornblende is also called amphibole. It is rather a dull, black mineral, forms six-sided crystals, is found in most igneous rocks and has Sp. gr. = 3-3.5; H = 5 to 6. *Augite* (also called pyroxene) is black but more brilliant than hornblende, forms monoclinic crystals and is found in most of the basic igneous rocks. It often alters to chlorite. (Sp. gr. = 3 to 3.5 ; H = 5 to 6).

***Olivine* (also called *peridot*) :** It is a silicate of magnesium and iron found in basic igneous rocks such as dolerite, basalt and periodote. It is greenish and looks like quartz (Sp. gr. = 3.2 to 4; H = 6 to 7).

The above minerals which contain iron and magnesium are known as ferro-magnesium minerals and are generally found in abundance in the more basic igneous rocks (basalt, dolerite, etc.)

Oxides

***Quartz* (Silica, SiO_2) :** It is an important constituent of the granite and other acid igneous rocks, and a chief constituent of sandstones where it occurs in the broken form (Sp. gr. = 2.6 ; H = 7).

Magnetite **(Magnetic oxide of iron, Fe_3O_4) :** It is bluish black in colour. Earthy varieties are known as red ochre. The streak is always cherry red (Sp. gr. = 4.5 to 5.3; H = 6).

Limonite (Hydrated ferric oxide, $2Fe_2O_3$, $3H_2O$) : It is amorphous, brown or nearly black in colour and the streak is yellowish brown (Sp. gr. = 3.6 to 4; H ≈ 5).

***Bauxite* :** It is essentially a hydrated aluminium oxide (Al_2O_3, H_2O), dry white in colour. It results from the decay and weathering of aluminium rocks, often igneous, under tropical conditions. (Sp. gr. = 3.5 ; H = 2.5).

Carbonates

The common carbonates are of calcium and iron. All carbonates effervesce when treated with dilute hydrochloric acid.

Calcite **($CaCO_3$) :** It is an essential constituent of marble, chalk or limestone; it is white or colourless and gives a white streak (Sp. gr. ≐ 2.9; H = 3).

Siderite **($FeCO_3$) :** Colour is brown in various shades and the streak is white (Sp. gr. = 3.7 to 3.9; H = 4).

***Dolomite* :** This is a carbonate of calcium and magnesium, Ca, Mg $(CO_3)_2$. Calcite and dolomite constitute the larger bulk of the limestones, but are sometimes found as secondary minerals in the igneous rocks. (Sp. gr. = 3; H = 4).

Sulphides

The common sulphides are the sulphides of iron, lead, zinc and copper.

Iron Pyrite **(FeS_2) :** Brass yellow in colour, this mineral is sometimes found interspersed in a coal seam. *e.g.* in the Pench Valley Coalfield. The streak is greenished or brownish black. Coal containing iron pyrites is observed to be more liable to spontaneous heating than other coal (Sp. gr. = 4.8 - 5.1; H = 6 to 6.5).

Galena **(PbS) :** It is grey in colour and has grey streak. (Sp. gr ≈ 7.2 to 7.7 ; H ≈ 2.5).

Sphalerite, Blende **(ZnS) :** It is also known as black jack. It is usually brown or black, the streak being reddish brown (Sp. gr. = 3.9 - 4.2; H = 3.5 - 4).

Chalcoprite, Copper Pyrite (Cu_2S, Fe_2S_3) : It is golden yellow in colour; streak is greenish black (Sp. gr. = 4.1 - 4.3; H ≈ 3.4 - 4).

Sulphates

Gypsum (Hydrated calcium sulphate, $CaSO_4$, 2 H_2O) : It forms colourless crystals, but some varieties may be white or grey. Plaster of paris is made from gypsum by heating it or expel some of its water of crystallisation and then grinding it to a fine powder (Sp. gr. = 2.3; H ≈ 2).

Chlorides

Rock Salt (known as common salt, NaCl) : It is colourless or white when pure (Sp. gr. = 2.2; H = 2).

Other rock-forming minerals are :

Kaolin **(also called China clay, Al_2 O_3 , $2SiO_2$, $2H_2O$) :** It may be described as hydrated silicate of aluminium. It is a soft, white, calyey material crumbling to powder when pressed between fingers. Felspars when decomposed yield china clay.

***Fireclay* :** Any clay or shale that does not fuse below 1580°C is called fireclay. The predominant constituent of these clays is the mineral kaolinite with the formula Al_2 O_3, $2SiO_2$, $2H_2O$. Besides aluminium and silica the fireclay also contains oxides of iron, calcium, magnesium, sodium and potassium.

***Fluorspar* : (CaF_2) :** It may be colourless, purple, green or yellow; lustre is vitreous but often transparent. It occurs in veins of metalliferous ores with galena (Sp. gr. = 3; H = 4).

Baryte **($BaSO_4$) :** It may be white, yellow, red or blue with a vitreous lustre and is commonly found in mineral veins associated with the ores of lead and zinc. (Sp. gr. = 4.5 ; H = 3).

***Uranium oxides* :** Uranium is the metal used for production of nuclear energy. Two uranium isotopes U-235 and U-233 are fissionable materials. Uranium ore contains 0.03 to 0.1% of uranium. The mined ore is processed to give a marketable concentrate containing 75% U_3O_8.

***Rare Earth Metals* :** This is a group of metals which includes metals like cerium (Ce), lanthanum, erbium, yittrium and others. Thorium is closely associated in nature with this group. Thorium is a metal related to titanium and has a sp. gr. of 11. **Monazite is a**

phosphate of cerium metals, but is industrially important as a source of thorium compounds as it contains a small percentage of thorium oxide or thorium silicate. Monazite : (Sp. gr. = 5.27; H = 5.5).

ROCKS

Rocks are divided as follows into three great groups based on the origin and mode of formation.

1. ***Igneous Rocks :*** Those that have consolidated at or relatively near the surface of the earth from molten material called *magma* originating from within the earth.
2. ***Sedimentary Rocks :*** Those produced chiefly by the breaking up of pre-existing rocks and deposition of the broken material in the form of layers.
3. ***Metamorphic Rocks :*** Those produced from pre-existing rocks by the action of high temperatures and pressures.

Igneous Rocks

Igneous rocks are formed by the cooling of molten material (called *magma*) at or relatively near the surface of the earth. Magma is a naturally formed mixture of molten rocks and minerals deep down the earth. The constituents of magma are mostly complex silicates and oxides of iron, aluminium, magnesium, calcium, sodium and potassium. Silica is always present in 35 to 75%. Most of the constituents of magma are nonvolatile with fusion temperature above 1000°C while some others, small in amount, are of a highly volatile character. Under the pressure of the earth's rocks the magma travels to zones of lesser pressure and along weaker planes and fissures in the existing rocks. When it erupts at the surface it is called lava as in the case of a volcano.

Rocks formed from the same magma have different names depending upon the crystalline texture which depends upon the rate of cooling which, in turn, depends upon the place of cooling, *i.e.* whether at depth or at surface. The rocks which are cooled at depths had very slow rate of cooling resulting in large coarse crystals and are called *plutonic,* those that are cooled at intermediate depths had a comparatively faster rate of cooling yielding smaller size crystals and are called *hypabyssal* ; those that cooled at or very near the surface had a fast rate of cooling yielding no crystals but nearly glassy texture and are called *volcanic.*

The igneous rocks are found in batholiths, lopoliths, laccoliths, dykes, sills and lava flows near valcanoes.

Batholith : It is a large igneous rock mass of irregular outline widening downwards, and without known floor (Fig. 1.2).

Lopolith : It is a large igneous rock mass which differes from the batholith in that the former has a basin-like shape and a gradually decreasing width with increasing depth.

Laccolith : It is an igneous rock formed from cooling magma which caused the previous beds to arch in the form of a dome. It is much smaller than batholith, has a known floor and a domed top.

Dyke : It is a more or less parallel-sided vertical wall of igneous rock formed due to upward instruction of molten magma generally through fissures or cracks existing in the previous rocks. (Fig. 1.2). It may be only a few metres or hundreds of metres thick, and is a common occurrence in the coal fields, e.g. the dolerite dykes and mica peridotite dykes in most of the coalfields. The coal around the dyke is usually semi-burnt, hard and useless. In some cases, however, coal near the dyke is converted into better quality like anthracite, or into naturally occuring coke which is useful. Dykes stand out as prominent ridges existing over long distances when the comparatively softer surface is eroded. An example of such dyke, seen on the surface, is Salma dyke in the Ranigunj field which extends from Damra to the West of Kalipahari and beyond. In the Jharia field, a promient dyke runs along Telmucha, Pipratand, Phularitand and Tundu Metal works, and another runs more or less parallel to the above, but west of Madhuban village across Jamunia stream.

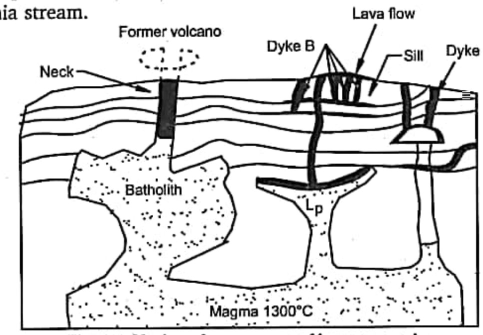


Fig. 1.2 Modes of occurance of igneous rocks
B-Batholith, Lp - Lopilith; Lc - Laccolith

Sill : A sill is a more or less horizontal sheet of igneous rock which extends laterally into the strata, usually introducing between the bedding planes. The thickness may range from a few metres to hundreds of metres. A sill is generally fed by a dyke and is rarely exposed at the surface by erosion of rocks above it.

Lava flows : The molten erupting to the surface is known as lava. The rock formed after cooling of lava is termed simply *lava*. The rocks to the south of the Satpura range in India, known by the name *Deccan Trap,* are lavas nearly 300 m thick covering about 3,00,000 sq. kilometres.

The lava flows, as they erupt to the surface, are also called *extrusive* igneous rocks, and batholith, laccolith, dyke, sill, etc. are called *intrusive* rocks.

Composition and Classfication of Igneous Rocks

Igneous rocks are commonly classfied on the basis of their chemical composition. The essential minerals that form the igenous rocks are quartz, mica (muscovite and biotite), felspars (orthoclase and plagioclase), and the ferromagnesian minerals (augite, hornblende and olivine).

On the basis of the silica content, the igneous rocks are classified as acid rocks or basic rocks as follows :

Silica more than 65%	–	Acid rock, e.g. granite
Silica 55% to 65%	–	Intermediate rock, e.g. diorite
Silica 45% to 55%	–	Basic rock e.g. besalt.
Silica less than 45%	–	Ultra-basic rock, e.g. Peridotite

The following table shows the classification of common types of igneous rocks based on texture and chemical composition.

	Acid	Intermediate		Basic	Ultrabasic
Plutonic	Granite	Syenite	Diorite	Gabbro	Peridotite
Hypabssal	Granite porphyry	Syenite porphyry	Diorite porphyry	Dolerite	
Volcanic	Obsidian	Trachyte	Andesite	Basalt	

Granite, granite, porphyry and obsidian have the same composition but the crystalline texture varies.

The common igneous rocks are :

Granite : It consists of large sized crystals of quartz and orthoclase felspar with some biotite (black mica). Greenish crystals of hornblende also may occur. Granite is the most common rock of the plutonic igneous type.

Very coarse granites are called *pegmatites* and are characterised by large crystals, sometimes 0.3 m across. *e.g.* at the mica mines near Kodarma (Bihar).

Syenite and Diorite : These are not deep-seated but hypabyssal rocks. These are darker than granite but have no quartz and contain felspar and biotite or hornblende.

Gabbro : This is much darker and heavier than granite containing a large proportion of ferromagnesian minerals, such as olivine, magnetite, etc. Dolerite is very widely distributed in dykes and sills and has the same composition as that of gabbro.

Peridotite : It is a common example of ultra-basic rocks and consists of ferromagnesian minerals, chiefly olivine with some augite and hornblende. Peridotites are crystalline, of very dark colour and with a Sp. gr. of 3 or over. The rock weathers easily. Mica-periodotite dykes and sills are common in Jharia, Raniganj and Karanpura coalfields.

Sedimentary Rocks

Sedimentary rocks include all rocks formed by deposition in beds or layers (strata) of the material derived from older rocks and also cover rocks formed by chemical or organic agencies. The materials may be carried by water, wind or glacier, *e.g.*, in the case of sandstone, shale, etc. Majority of these sedimentary rocks are formed by deposition on riverbed, or seabed and into basins or depressions on earth's surface. The consolidaton of loose sediments into hard also by deposition of cementing materials into spaces between particles of the sedments. The chief cementing materials or by some mineral finely spread throughout the rock. The colour of the rock often changes on account of alteration of the original constituents due to atmospheric agencies, cheifly water. The sedimentary rocks were formed practically in the horizontal position but due to earth movements associated with earthquakes or other causes they were tilted from the horizontal position.

Alluvium is a loose or unconsolidated deposit resulting from the breaking up of the bed rock. The broken particles either remain in position or are moved by surface water and are redeposited elsewhere as a more or less loose deposit which may convert into unconsolidated rock. The Indo-Gangetic plain has a thick layer of alluvium, several thousand metres thick.

Some sedimentary rocks are formed from plant or animal remains *e.g.,* coal and some form of limestone. There are some sedimentary rocks which are formed by deposition of material which was once in solution in water but later on separated from it, e.g., some deposits of limestone, ironstone (hematite), gypsum, etc. Where the water carrying carbonates in solution escape drop by drop from the rock in cavities or fissures, the carbonates may deposit in the form of rods or columns of rock. If the deposit is on the floor it is known as *stalagmite* and if it is hanging from the roof it is known as *stalacite.*

Placers are friable sandy-clay, sandy-gravel or similar formations containing some rare metal or mineral such as gold, platinum, tinstone, *etc.*

The rocks adjoining a seam but below it are called *floor* of the seam and those above it are called *roof* of the seam. In the metal mining practice the rocks adjoining an orebody are called its wall rocks, the *handing wall* being above the ore body and the *foot wall* being below it.

Bedding Plane : In a sedimentary rock, a bedding plane is the junction plane of one bed of rock with another. Bedding planes are therefore the surfaces which divide the different layers of water deposited or wind-deposited sediments and are planes of eakness in the rock as the coherence betwen different beds is usually less than than within a bed. Splitting is therefore usually easy along a bedding plane. Bedding planes are a feature common to all sedimentary rocks and to some metamorphic rocks which were derived from them.

Dip and Strike : The strike of a bed is a level line on its surface. In effect it is a contour line in the plane of the bed. The line of true dip in a bed or surface is the steepest line in the inclined surface and is always at right angles ato the strike (Fig. 1.3).The dip of a bed along any direction between the strike and the true dip is called its *apparent dip.*

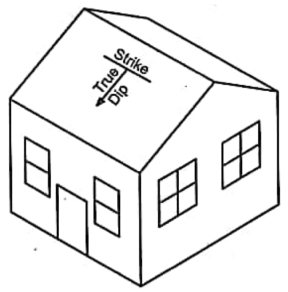
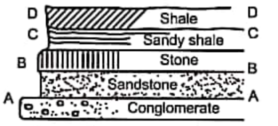


Fig. 1.3

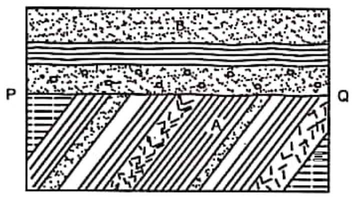
Unconformity : When rocks are deposited one above the other in uninterrupted succession, they are said to be conformable. When they are laid down upon the eroded surface of older strata, the two series of strata are said to be unconformable and the plane of contact between them is called *unconformity.* Unconformity is evident chiefly by the difference of dips between underlying and overlying series of rocks and also be denuded and eroded surfaces of older series of rocks and also by denuded and eroded surfaces of older series and presence of conglomerate at the base of upper series. (Fig. 1.4).

The common sedimentary rocks are :

Conglomerate : It consists of rounded pebbles embedded in a finer grained material forming the rock. Conglomerates occur in lens shaped masses and not in the form of regular beds. If the pebbles are not rounded but angular, the rock is called *breccia.*



AA, BB, CC, DD, are bedding planes



PQ is unconformity ; B - younger series

Fig. 1.4

Sandstone : This consists of particles of quartz cemented together, usually by silica, and sometimes by calcite or other cementing materials. The colour of the sandstone depends upon the cementing materials. Sandstones are porous in nature and permit water or other fluids like gases to flow throgh the pores. The rocks are of common occurence in the coalfields.

Shale : This consists chiefly of clay and has fine texture. It can be split along the bedding planes and also into thin leaves of one millimetre or so.

Fireclay : It is a variety of clay described earlier. It is a refractory material, *i.e.* one capable of resisting very high tempratures without becoming soft or plastic.

Laterite : Laterite is a red ferruginous porous clay, formed by weathring of such rocks as basalt, mica schist and gneisses and composed chiefly of alumina and iron oxide. It is found upto a depth of nearly 3 m in the eastern part of Raniganj and Talchir coalfields. Laterite is the common surface rock in iron ore localities.

Limestone : It consists of $CaCO_3$ (Carbonate of lime) with small impurities. Some limestones are made up of the shells from organic origin, others are formed due to depostion of calcite from solutions. Chalk is a soft porous variety of limestone.

One of the occurrences of limestone is in the form of sea sand and near Dwarka (Gujarat) such sea sand is dredged out from the sea bed and used for the manufacture of cement. It is soft compared to the sand formed out of quartz.

Dolomite : It is a compound of $CaCO_3$ and $MgCO_3$ in various proportions.

Fossils

Fossils are the remains or traces of former living creatures or plants now preserved in rocks. Although the earth is estimated to be at least 4000 million years old, man's footprints and bones do not appear in the fossil record until about one million years ago. The remains of plants or creatures are mostly in decayed or decomposed state. Bones, teeth or shells of animals, the wood or leaves of trees, and sometimes even the footprints or tracks of animals, are preserved as fossils. Such fossils are generally found in fine grained sedimentary rocks such as limestone, shale, etc. They are not found in igneous rocks, though bedded volcanic tuffs (fragmentary materials ejected by volcanoes) may occasionally contain some fossils. To the geoligst fossils are useful as their study gives a clue to :

1. The climate of the period during which the enclosing rock was laid down.
2. The nature of the rock, whether it is a deposit in fresh water, sea or on land.
3. The relative age of the rocks belonging to a particular period extending over thousands of years are characterised by the same fossils over a wide area.
4. The geography of past ages, *i.e.* the situation of ancient continents, seas, etc.
5. The evolution of life throughout geological time.

Metamorphic Rocks

Metamporphic rocks are altered forms of pre-existing rocks which might have been igneous or sedimentary. The alternation may be due to either heat or pressure or by both acting together. Under the effect of pressure and temperature certain of the original minerals are no longer stable and give place to new minerals. Quartz, felspar, mica and hornblende are stable under both igneous and metamorphic conditions. In addition to new minerals, new textures arise during the process of alteration or **metamorphism.** Rocks not altered beyond recognition include quartzite (from sandstone) and slate (from clay). Rocks altered beyond recognition include rocks known as *gneisses* and *schists.*

The original rock-forming material and the names of original rocks and metamorphic rocks produced by it are given below.

Mineral	*Original rock*	*Metamorphic rock*
Sand	Sandstone	Quartzite
Clay	Shale	Slate, Mica-schist
Calcium Carbonate	Limestone	Marble
	Mudstone	Argillite
	Granite	Gneiss
	Slate	Schist
	Basalt	Amphibolite

FORMATION OF MINERAL DEPOSITS

The term *mineral deposit* has been explained in the opening page of this chapter. A mineral deposit is simply a deposit of minerals, not necessarily commercially workable; the viability will vary with the price which fluctuates with time. Thus a mineral deposit, considered unprofitable at one time, may be economically extractable from a mine with increase in its market price.

Magma is the original source of most of the minerals. The magma which is chemically very reactive due to pressure, temperature and compostion of various minerals dissolves adjacent rocks through which it travels, giving rise to new minerals. The constituent minerals, mostly rock-forming silicates and oxides are deposited at various stages as the magma cools down during its passage. Minerals having nearly similar fusion points segregate and concentrate together resuling in **magmatic segregation.** Important deposits of metallic oxides such as magnetite and limentite, and sulphides such as pyrrohtite and chalcopyrite are formed in this way. Magmatic segregation may take place at different depths during the travel of the magma and at different temperatures. Most of the ferromagnesium silicates and other oxides are formed at depth by magmatic segregation.

After deposition of minerals by the process of magmatic segregation the magma is fluid and has a concentration of volatile constituents, *i.e.* various gases and vapours. Cooling of the liquid portion results in the formation of **pegmatites.** Such pegmatites often contain a concentration of minerals which occur only as accessory forms pegmatites introdes in the pre-existing rocks forming dykes and veins. Economics deposits of minrals like felspar, quartz, mica, beryl and aptite are formed in this way e.g. in Giridih, Hazaribag (Bihar), Bhilwara (Rajasthan), etc.

The magmatic segregation and formation of pegmatites leaves the residual magma very fuild and it contains heated gases of great chemical activitiy. These gases penetrate the adjacent country rock and by their reaction with the latter form mineral deposits. Such deposits are known as **pneumatolytic ore deposits.** Examples are cassiterite deposits.

During the final stage of consolidation of magma, its aqueous solutions which consists of heated waters of great chemical activity deposit their mineral load. These aqueous solutions, because of their fluidity, are capable of travelling long distances from their parent source. The ore deposits formed by such aqueous but highly fluid soluitons of magma are known as **hydrothermal ore deposits.** The term also covers deposits formed by descending surface waters which sometimes leach away valuable constituents of existing rocks and precipitate their load of minerals in the cracks, fissures and cavities in the earth's crust.

Surface waters passing down into the fissures and cracks in the earth sometimes carry minerals in solution or suspension derived during their passage over a variety of rocks. The heat beneath the earth's surface renders such descending circulating waters chemically active and sometimes the minerals of pre-existing rocks are replaced, partially or completely, by minerals of the circulating waters, particle by particle. The structure of the pre-existing rock may remain unaltered. The ore deposits so formed are called **metasomatic ore deposits.** The term metasomatism includes the alterations arising in rocks by the passage through them of heated waters from igneous sources. Some deposits of chlorite, serpentine, and chalcopyrite have been formed in this way.

The process of **metamosphism** which results in the formation of metamorphic rocks may generate enough heat and pressure to alter existing mineral deposits of impure or low-grade ores into comparatively more pure and valuable minerals. Some banded hematite formations have changed to banded magnetite-quarzite rocks in Salem and Tiruchirapalli districs by metamorphism. Another example of heat changing pre-existing mineral into a more pure mineral is offered by the conversion of bituminous coal into anthracit in the vicinity of dykes and sills in some cases. Sillimanite (Al_2O_3, SiO_2) in Assam and eastern Maharashtra (Bhadara district)and kyanite (Al_2O_3, SiO_2) are formed by metamorphism. Talc, hydrated magnesium silicate, is also a product of metamorphism of magnesium bearing rocks like dolomite, e.g. near Jaipur in Rajasthan.

Some mineral deposits are of **sedimentary origin** and the deposits of sediment may be formed organically as in the case of coal deposits, or chemically, as in the case of some limestone or chalk deposits. Such deposits are always bedded and stratified.

Alluvial, detrital or placer depositer are formed by breaking up of the parent rock and subsequent tranporation of the mineral particles by stream or wave action. The minerals are found in sizeable concentraion where the velocity, and hence the carrying power of the currents, is decreased. In such deposits the minerals are concentrated into fractions according to their specific gravities and two or more minerals of similar sp. gr. may be found together. Examples of such placer deposits are gold placers, with the gold being associated magnetite, chromite, *etc.* Alluvial, gem deposits, platinum, tin and worfram are some other examples of alluvial or placer deposits.

Laterite deposits are formed by the leaching away of soluble minerals leaving behind in the laterite a valuable ore such as nickel or bauxite. Thus these are normally surface deposits.

Ore deposits which outcrop at the surface undergo weathering in the outcrop zone and may decompose. The weathered upper part the deposits is known as **gossan.** The gossan is usually an oxidised zone which may sometimes change into carbonates. Thus a vein of galena at depth may consist of cerussite ($PbCO_3$) in the gossan. Copper sulhipde of chalcopyrite (Cu_2S, Fe_2S_3) which may occur in a vein at depth changes into malachite, $CuCO_3$, $Cu(OH)_2$ in the gossan, e.g. at Khetri in Rajasthan, Concentration of minerals takes place in the gossan as the lighter or less stable minerals are washed away by percolating waters during weathering. Rich mineral deposits of economic value, therefore, occur as a cap over low-grade ore.

Evaporation of water from solutions containing minerals is a familiar example of one process of mineral formation, *e.g.* common salt produced by evaporation of enclosed sea water.

Formation of the mineral coal is explained in the next chapter.

ROCK STRUCTURE AND FAULTS

Outcrop : The outcrop of any stratum , or rock is that part of it which is exposed at the surface. In most of the cases an outcrop is not visible at the surface, being covered by soil or alluvium.

Anticline and Syncline : Rocks are not always dipping at a uniform gradient, but are sometimes undulating or folding. The folding results from pressure or force which may be local, and vertical or horizontal over a wide stretch. The shape of the land surface may be quite different from the rock structure below. The crest of a fold where the strata are bent up to form an arch is called an *anticline* and the beds dip away from the axis of the fold (Fig. 1.6, Left). If the folding is such that the beds dip towards a common axis or plane the strucutre is known as a *syncliner* (Fig. 1.6 , B). In a syncline the beds form a through and dip towards the axis on both sides. In a dome-shaped structure the beds dip away form a common point whereas in a basin-shaped structure they dip towards a common point. The coal seams and associated rocks at Amlabad colliery (Dhanbad district) have a dome-shaped structure.

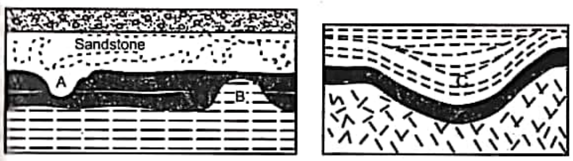


Fig. 1.5 In a mineral bed A-Washout, B-Roll, C-Swilley

Washout : A washout is an irregularity where the bed thins out, generally due to erosion by a river or stream and the roof of the bed is filled up with sand or othe material (Fig. 1.5).

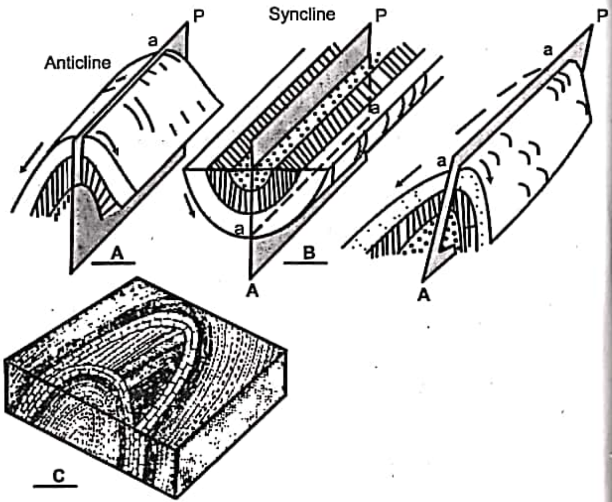


Fig. 1.6 AP - axial plane ; aa - axis of fold
C shows outcrops of eroded beds in a pitching anticline

Roll : This may be considered exactly opposite of washout in that the floor or which a mineral bed is deposited shows an upward extension towards the roof of the mineral bed which thins down in the area of roll (Fig. 1.5).

Swilley : A swilley is a trough or depression in which the seam, with its otherwise undisturbed roof and floor, sags several metres below the general seam level, the seam generally being thicker in the swilley area. (Fig. 1.5).

Outlier and Inlier : An outlier is an outcrop of rocks surrounded by older rocks and, therefore entirely separated from the main mass by denudation. It is mere patch surrounded by older strata on all sides.

An inlier is an outcrop of rocks in the form of a patch of strata surrounded by newer beds on all sides (Fig. 1.7).

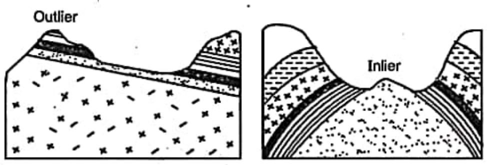


Fig. 1.7

Fault

A *fault* is a fracture in rocks usually associated with lateral or vertical displacement of the fractured beds (Fig. 1.8). Faults occur when the strength of the rocks is insufficient to withstand the stress due to earth movement.

The vertical displacement CD of the fractured bed is called the *throw of the fault* and is measured from *floor to floor* of the seam. It may be from a few metres to hundreds of metres. The horizontal displacement DE is called the *lateral shift or heave and* it represents the width of the ground which the coal seam is missing. The area in which coal seam is thus missing is called *want or barren ground.* The plane of fracture FF is called *Fault plane.* The angle DCE is called the *Angle of hade or underlie of the fault* and is always measured from the vertical. It is usually from 10° to 40°.

A fault is called downthrow or upthrow according to the side from which one travels towards the fault plane. A man travelling over bed M towards the fault would meet a down-throw fault, whereas a man travelling over bed N towards the fault plane would meet an up thoro fault. Throw of a particular fault is not always the same at all places.

A *Normal fault* is one in which the ends of the broken strata do not overlap and the fault plane hades forward towards the displaced bed. The two parallel faults M and N form a step fault, (Fig. 1.9) and the intersecting faults N and S form a trought fault. Normal faults occur in areas of tension and cause an extension of the surface of the faulted region. A *reverse fault* is one in which the older rocks are thrust over new ones by lateral compression as at T in Fig. 1.9 so that the ends of the broken strata overlap each other. In such a case vertical borehole would pierce the same bed twice. Reverse faults occur when the strata are compressed into a space less than their original horizontal extension and are common where strata are intensively folded. In general most of the faults are normal faults and reverse faults are uncommon in coalfields.

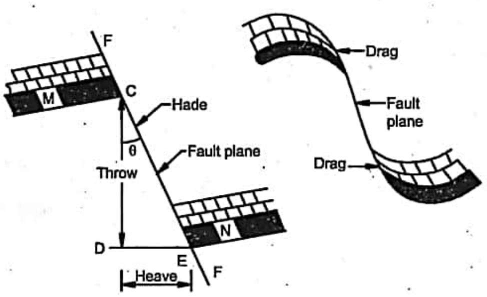


Fig. 1.8 Normal Fault, θ – Angle of Hade, FF – Fault Plane

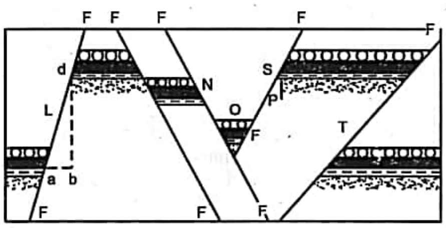


Fig. 1.9 Various types of faults L,M,N and S are normal faults. M and N form a step fault. N and S form a trough fault. T is a reverse fault.

The strike of a fault, or its true bearing is the direction of a level line in the fault plane. The direction of the fault, as shown on a mine plan, is not necessarily its true bearing because the line of the fault shown on a mine plan represents the line of intersection of the fault and the mineral bed.

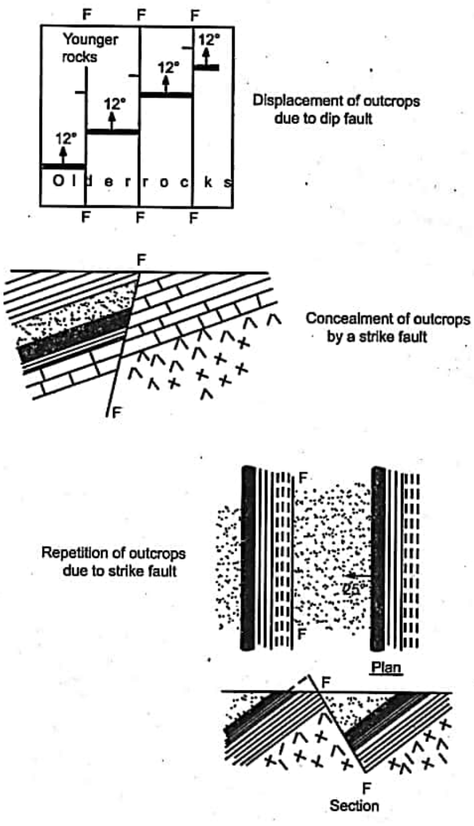


Fig. 1.10

A *dip fault* is one whose general direction or trend is parallel to the dip of the beds. It is also called a *transverse fault*.

A strike fault is one whose general direction or trend is parallel to the strike of the strata. It is a also called *longitudinal fault.* A truly dip fault or a truly stike fault is rare. A fault that runs at some angle between the dip and strike of the beds is called an *oblique fault.*

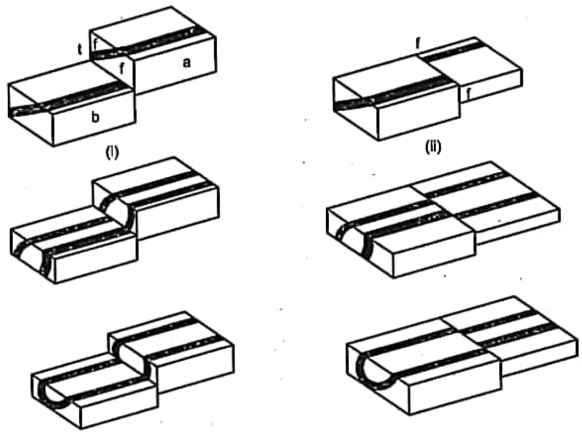


Fig. 1.11 Lateral displacement of outcrops by faults. Right side figures show faults on a level ground after denudation. ff is fault plane.

Effect Of Fault On Outcrops

Sometimes a fault may be traced at the surface by a line of hollows or depressions that are denuded out as the fault zone consists of broken and shattered rocks which easily weather out. More often, however, a fault has little or no effect on the surface contours.

A strike fault has effect of causing repetition of the outcrop of some of the beds (Fig. 1.10).

A dip fault causes lateral dispalcement of the strata.

In Fig. 1.11(i) ff is the fault plane, a the upthrow side, b the downthrow side, and t the vertical throw. After denudation of the surface of upthrow side a has come in level with the surface of down throw side b. The outcrops then appear as in (Fig. ii) showing lateral displacement.

✧ QUESTIONS ✧

1. What are the different types of rocks ? How have they been formed? Give two names of the each type of rocks.
2. What is a fault ? How does it affect an outcrop of a seam ? Explain the various terms used in connection with a fault.
3. What is a fossil ? How does it help a geologist ?
4. Describe briefly the processes which results in the formation of mineral deposits.
5. Write notes on :
   magma, dyke, sill, unconformity, syncline, outlier.

✧ ✧ ✧

Chapter -2

Coal And Coalfields of India

Energy is available to man from fuels (listed below), biogas, hydropower, the sun, the wind, hot water springs, sea-waves, and internal heat of volcanoes. Attempts to harness energy from the seawaves and the volcanoes are in preliminary stage.

Fuels may be divided into 4 classes.

Solid – coke, coal, wood, charcoal, bagasse, lignite, cowdung, etc.

Liquid – diesel, petroleum and its derivatives, coal tar and its derivatives, shale oils, alcohol, etc.

Gaseous– Lighting gas, coke oven gas, producer gas, blast furnace gas, water gas, natural gas, etc.

Nuclear– Uranium based energy, etc.

About half the actual energy consumption in India is commerical, consisting of coal, oil nuclear and hydel. The other half is non-commerical like firewood, agricultural waste and animal excreta. The share of coal is about 60 per cent of all commerical primary energy, in terms of heat content as coal equivalent.

Coal	59.5	percent
Oil	27	percent
Hydro	12.5	percent
Nuclear	1	percent

Wind-power, a source of non-conventional renewable energy, makes very little contribution to the overall generation in India.

Origin of Coal

Coal is stratified carbonised remains of plant material. Two theories, known as the *In Situ Theory and the Drift Theory,* have been advanced by the scientists to explain the heavy accumulations of vegetable and organic matter which formed the coal seams.

Drift theory and in-situ theory : In India, the sequence of the strata containing the coal seams is in most cases suggestive of continuous deposition under water. There is hardly any record of erect stems with attached roots occurring in the coal seams; on the other hand indications of prostrate or inclined trunks are numerous. Most of the coal seams are invariably interbedded with shale and sand stone of definitely sedimentary origin (fresh water or marine). These are other indications have led to the acceptance of the **Drift Theory** to explain the heavy accumulation of vegetable matter. It is believed that the coal seams, as we see them today, were formed out of plants and trees which grew on this earth in abundance millions of years ago under climatic conditions which were far different than today. Due to earthquakes, tornadoes and other tectonic activities these trees fell down, and the plant material and trees, were drifted to lakes, river valleys, estuaries and seas by water. Large sediments of sand and earth also were carried by water and deposited over the vegetalbe matter. The process continued over million of years during which the lakes, sea-beds, lagoons and rivers subsided further and were filled again by fresh plant material and sediments of earth and sand. The organic matter, under its own weight and the weight of the sediments under which it got buried, was compressed and the heat generated by such compression, the bacterial action, and absence of oxygen resulted into transformation of the plant material to hard semiburnt solid known as *coal.* This natural process of carbonisation accompanied by bacterial action, resulted in evolution of gas that got entrapped into the coal. It is marsh and clay deposits were changed into shales and the sand, into sandstone.

In-situ theory : The heavy accumulation of plant material which formed coal in Britain is explained by the **in-situ** theory. According to this theory, the areas where the vegetation grew, subsided and the trees were submerged in water which flew into the subsided area. They were buried beneath the earth and sand brought by the water. Fresh vegetation grew over the debris in the area and the process repeated over thousands of years.

The plants and animals which were buried in the large accumulation of plant, clay and earth sediments are preserved as fossils in many cases and can be seen in coal associated rocks.

Rank Of Coal

Wood, peat, lignite, bituminous coal and anthracite are the main stages of transformation from plant material to coal, anthracite being the final stage or rank. Higher rank indicates coal with high carbon content. Anthracite is the purest coal but does not necessarily have the maximum heating value which depends upon the percentage of carbon and hydrogen. The transformation from wood to anthracite is accompanied by progressive elimination of hydrogen and oxygen and gradual increase of carbon content. The inorganic matter of the original plant material and whatever got mixed with it during the build up of the heavy plant deposits constitutes what is known as *ash* in coal. The sp. gr. of coal is highest in anthracite (about 1.5), and lowest in lignite (about 1.2) whereas sp. gr. of bituminous coal varies between 1.28 and 1.4. the calorific value of lignite is nearly 2400 kcal/kg and that of bituminous coals varies from 2500 to 6600 kcal/kg.

The thickness of coal seams varies from a few centimeters to nearly 140 metres in this country but thickness below 1.2 m is considered uneconomic for coal extraction in our country though in Britain and Germany a coal seam 1 m or 1.2 m thick is considered economically workable. The thickest seam in India, and perhaps the second thickest in the world, is Jhingurda Top seam (140 m).

Gondwana Period

The period during which coal was formed extends over millions of years. From astronomical considerations, the evidence available from radio-activity of minerals, and other data scientists consider the age of the earth as a planet over 4000 million years but the period during which sedimentary strata have been laid down is of more important to geologists. This period, which may be called geological time, is considered to be over 600 million years and is divided into periods as shown in Table 1. Archaean or Precambrian is the oldest period, and is longer than all the others put together.

The Indian coalfields were formed during two main geological epochs,

1. Permian period, and
2. Tertiary period.

A subperiod of Permian is known a *Gondwana period* and the series of rocks formed during that period are known as rocks of Gondwana system. The gondwana system rocks are subdivided into series as shown in Table 2. The names of the rock measures, the corresponding names are given in the table. The *Gondwana Sone*, Koel, Mahanadi, Pranhita Godavari, Narmada, Wardha, Pench, etc. and over the regions of Bengal, Bihar, Orissa, Uttar Pradesh, Madhya Pradesh, Andhra Pradesh and Maharashtra.

Table 1

Geological Time Scale

Era	Period	Age in million years	Life development
Quaternary	Recent Pleistocence	0.1 1	Living animals. Advent of man.
Tertiary or Keinozoic	Pliocene	12	Flowering plants dominate vegetation.
	Miocene	25	Primitive horses and other ungulates.
	Oligocene Eocene	40 60	Apes appear. First Placental animals.
Secondary Or Mesozoic	Cretaceous	110	Giant reptiles and ammonites, but become extinct at close of period
	Jurassic	150	First flowering plants; first birds.
	Triassic	180	Ammonites, ambhibia and reptiles in abundance. First dinosaurs and primitive mammals.

	Permian	215	Rise of reptiles. Trilobites disappear.
		Carboniferous	275 Large non-flowering plants; first reptiles.
Primary Or Palaeozoic	Devonian	325	Age of fishes. First amphibia, abundant corals & brachiopodes
	Silurian	360	First land plants. Graptolites disappear at end.
	Ordovician	450	Abundant trilobites & graptolites.
	Cambrian	600	Abundance of tri-lobites.
Pre-Carmbrinan	Precambrian	1500	Soft bodied animals & plants.
Archaeans	Archaeans	4000	Lifeless.

They possess bituminous coals and produce nearly 97% of the total production in the country. The tertiary period coals produced during the periods Eocene to Pliocene are available in Assam, Nagland, Meghalaya, Arunachal, Sikkim, Kashmir, Kutch, Rajasthan, and Neyveli (Tamilnadu) as deposits of lignite or a stage between lignite and bituminous, as in the case of Assam and other areas of North East India. The tertiary coals of North East India are altogether of different origin compared to the the Gondawana coals and are very low is ash but high in sulphur content; some of them have very high caking index and can be considered medium coking. Their total production accounts for only 2% of the coal output in the country. Semi-anthracite coal is available only in a few small pockets near Darjeeling and in Jammu Kashmir where the heat of intense mountain building movements of the sub-Himalayan zone convertd patches of semi bituminous coal into semi anthracite. At Rajhara Colliery (Palamau district, Bihar) the coal that is extracted is sometimes called semi anthracite because of its

high calorific value, low v.m. content, (3 to 10%), and appearance. Its ash percentage varies from 13 to 15 and moisture is about 2%. The coal is difficult to ignite but once ignited, continues to burn for long without smoke (smokeless fuel).

The map of India shows major coalfieds and lignite fields.

India holds position of being the fifth largest coal producing country in the world. Its coal reserves have been officially put at 1% of the world's total coal reserves. In the world's coal reserves share of CIS countries (including Russia) is 53%, USA 27% and China 9.4%.

Coalfields of tertiary period

1. Arunachal : Namchuk, Miri, Abor Hill, Daphla.
2. Assam and Nagaland : Nazira, Jaypore, Makum, Mikir Hills, Doigrung, Longai.
3. Meghalaya : Garo Hills, Kariabari, Rongrenggiri, Western and Eastern Darenggiri, Khasi and Jaintia Hills, Cherapunji, Harigaon, Umraing, Lakadong, Baljona Dogring.
4. Jammu and Kashmir : Jammu (Jangalgali), Kalakot, Metka, Mahogala, Chakar, Dhansal-Swalkot, Ladda, Chinkah.

Lignite fields of tertiary period

1. Kashmir : Kashmir Valley.
2. Rajasthan : Palana.
3. Tamilnadu : South Arcot (Neyvelli) ; Cuddalore-Pondicherry Area.
4. Kerala : West Coast.
5. Chutch : Umarsar Area.

Table 2

Correlation of the Lower Gondwana Strata in India

GONDWANA SYSTEM (Age-Carboniferrous to Jurassic)

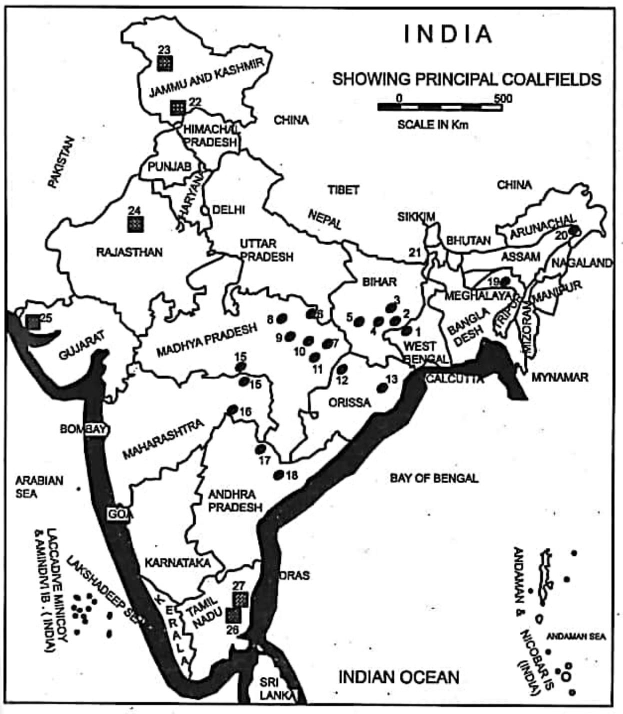
Upper Gondwana System consisting of Mahadeva Series, Pachmarhi Stage, Jabalpur Stage, etc. but practically no coal

Lower Gondwana System:

Gondwana Divisions	Damodhar Valley	Son & Mahanadi Valley	Satpura	Wardha Valley & Godavari Valley	Age
Panchet Series	Panchet	–	Almod	Mangli	Lower Triassic
Damuda Series	Raniganj Measures or Stage; also called Upper Coal Measures	Himgiri Pali	Bijori 1,000 m	Kamthi	Upper Permian
	Barren Measures or Iron stone shales	–	Motur		Middle Permian
	Barakar Measures or Stage ; also called Lower coal Measures Karharbari	Barakar	Barakar Umaria	Barakar	Lower Permian
Talchir Series	Talchir	Talchir	Talchir	Talchir	Upper Carboniferrous

U N C O N F O R M I T Y

Metamorphics Metamorphics



COALFIELDS

1. Raniganj. 2. Jharia. 3. Giridih. 4. Bokaro. 5. Karanpura. 6. Singrauli. 7. Bisrampur. 8. Umaria. 9. Sohagpur. 10. Chirimiri. 11. Korba. 12. Hingirampur. 13. Talchir. 14. Pench Kanhan. 15. Kamptee. 16. Wardha Valley. 17. Tandur 18. Pranhita Godavari Valley. 19 Khasi Hills. 20. Upper Assam 21. Darjeeling.

LIGNITE FIELDS

22. Kashmir, 23. Baramula. 24. Palana 25. Kutch. 26. Neyvelli. 27. Bahoor. Coal production in 94-95 was 245 million tonnes (CIL's share 233 Mte).

Bended Constituents of Coal

The bituminous coal contains four different visible bands, vitrains, clarian, durain and fusain which are rock types and not chemical entities. Thus different vitrains have different chemical compositions. There are subdivisions of these main rock types into macerals and petrological units, *e.g.*, vitrinite, fusinite, exinite, resinite, micrinite, etc. All the four bands are not present in the same sample of coal and very often only two bands exist.

1. *Vitrain :* It forms the brights layers, vitreous in texture in a coal sample and is to be seen in high quality coals. It is friable, breaks in small cubical or rectangular pieces and is clean to touch.
2. *Clarain :* It also available in coal as bright layers, parallel to the bedding planes. It does not soil the fingers, resembles vitrain and is present in high quality coals.
3. *Durain :* This constituent band has dull appearance and is lenticular in shape. Compared to vitrain and clarain it is hard.
4. *Fusain :* It is dull appearance, friable, soils the fingers, and is the most impure band in coal. The layers resembles charcoal. The band is present in all types of coals.

Coal analysis And Commerical Grades

Size gradation of coal

Coal is classified generally into the following four sizes for purposes of marketings ;

Steam coal - 50 mm to 200 mm and above.

Rubbe coal - 25 mm to 50 mm.

Slack coal - 0 mm to 25 mm.

(Sometimes coal of 0 mm to 50 mm is also considered as slack coal by some coal-users).

Run-of-mine (R.O.M.) coal - unscreened coal of all sizes.

The quality of coal depends upon (1) the chemical composition of the original plant material which formed the coal, (2) the mode of accumulation and burial of the plant debris, (3) the extraneous inorganic matter than got intermixed with the plant material, (4) the extent of bacterial decay, (5) the manner and duration of carbonisation, (6) age of the deposits, and (7) subsequence geological disturbances.

Analysis of coal is expressed in two ways :

1. Ultimate analysis 2. Proximate analysis

Ultimate analysis gives the percentage of elements presents (oxygen, hydrogen, carbon, nitrogen, sulphur and phosphorus) and is useful to consider the suitability of coal for certain purposes, particularly in the chemical industry.

The **proximate analysis** gives the percentage (by weight) of **ash, moisture, volatile matter and fixed carbon.** Ash is the inorganic residue left when coal or coke is incinerated in air to constant weight under specified conditions. Moisture is the water expelled in various forms when coal is heated under specified conditions. Volatile matter which consists of various gases in coal is equal to the total loss in weight *minus* the moisture when coal or coke is heated under specified conditions. Fixed carbon is obtained by subtracting from 100 the sum of percentage of moisture, ash and volatile matter.

Coal that has been exposed to contact with water in the seam, in a washery, during storage or transit in rainy season in open wagons or trucks, may carry free or visible water adhered to the surface or in cleavages and cracks. Such moisture is known as *free moisture.* Some moisture is always present in the coal and formed part and parcel of it during its natural formation ; such moisture is called *inherent* moisture. *Total moisture* in coal is the sum of free moisture and inherent moisture. Proximate analysis gives the percentage of inherent moisture. The external moisture i.e. free moisture content is determined by drying in air samples of the as-received fuel on trays until their weight is constant. Determination of inherent moisture is explained a little later.

There are **four different ways** in which the **proximate analysis** of coal can be reported.

1. analysis on as-received sample basis
2. analysis on air-dried sample basis.
3. analysis on equilibrated-conditions basis.
4. analysis on unit-coal basis or dry mineral-matter-free (dmmf) basis, or pure coal basis.

Reporting analysis on **as-received sample** basis is not a common practice. The coal samples have to be collected in sealed containers so that the moisture adhering to the coal is not dried up during transit or storage pending analysis. Some power houses attach importance to such analysis to know the real coal and the external moisture adhering to it so that payment is made only for the coal to the suppliers.

Reporting analysis on **air-dried sample** basis is commonly adopted practice. An air dried sample is that which is spread in thin layers in a laboratory and exposed to its atmosphere for a few days till its weight is constant. Generally 2-3 days suffice, depending upon relative humidity. A sample which is spread in thin layers in a laboratory for 6-10 hours and air allowed to pass over it an laboratory temperature and humidity, also attains constant weight and is treated as an air-dried sample. Where a large number of samples are to be analysed, a quicker method of getting an air-dried sample is to heat a laboratory sample in an oven to nearly 50°C and keep it at that temperature for 1½ to 3 hours, (as per British Standard Specifications). In an airdried sample the external moisture adhering to it dries up, the time depending upon relative humidity. Analysis of air-dried sampled can be carried out in most of the laboratories which are not equipped with special chamber having conditions of equilibrated atmosphere.

Reporting **analysis on equilibrated conditions** basis is a more scientific method. The analysis is done on coal samples passing through IS 20 sieve after equilibrating under the conditions given below for 48 hours.

Atmospheric temperature = 40° ± 2°C

Relative humidity = 60% ± 2%

Only some laboratories are equipped with chambers having such conditions of equlibration. This analysis provides a uniform basis for comparision of coals from different parts of the country is local temperature and relative humidity do not affect the moisture

percentage in coal analysis. Commerical grades fixed by the Govt. of India are based on analysis carried out under equilibrated conditions. Coal containing less than 2% moisture are known as low moisture coals; if moisture is more than 2%, it is high moisture coal.

Reporting analysis on **unit-coal basis (dmm basis)** provides a better way of comparing ranks of different coals. The analysis indicates the nature of the coal substance, that is, the pure combustible or organic part of coal, viz. carbon and volatile matter. Moisture and ash do not contribute to the heating capacity of coal and in dmmf analysis only the v.m. and fixed carbon are reported as percentages of the coal substance.

Laboratory Methods For Proximate Analysis

Samples of coal are collected from wagons, trucks or conveyor belts in a manner laid down in the Indian Standard Specifications or in agreements between the suppliers and consumers. To get a representative sample for analysis or other tests in the laboratory, the collected sample in subjected to the processes of crushing, coning and quartering. These operaions are repeated two or three times to get a laboratory sample, usually of 50 grammes, which can pass through IS 20 sieve (corresponding to BS 72 sieve). For the proximate analysis only 1-5 grammes of the sample prepared in the aforesaid manner is taken for each test.

Moisture percentage (by weight) of coal is determined by heating an air-dried sample of coal in a shallow glass dish covered with a ground-glass cover plate at 105° to 110°C until the weight is constant. Loss of weight gives moisture content (inherent moisture). **Volatile matter** is determined by heating a laboratory sample of coal in a muffle furance out of contact with air for abot 7 minutes at a temperature of 925°C. Loss of weight gives the weight of moisture and volatile matter. Percentage of V.M. is calculated after determining moisture percentage separately as stated here. **Ash percentage** is determined after heating a sample of coal in an open silica dish in a well ventilated muffle furnace to nearly 815°C till the weight remains constant. The inorganic residue left on the dish is ash. Percentage of **fixed carbon** is obtained by subtracting from 100 the total percentage of ash, moisture and V.M. The mineral matter equals 1.1 × ash percentage.

Grades Of Coking And Non-Coking Coals

The existing grades of coals were formulated by the Govt. of India in July 1979. for non-coking they are based on useful heat value calculated from the results of proximate analysis. The calorific value of coal or any fuel can be determined in a laboratory in a bomb calorimeter but in the case of coal it can also be calculated if the ultimate analysis or proximate analysis results are known. Ultimate analysis should give the percentage of carbon, hydrogen, oxygen, nitrogen, sulphur, ash, and moisture. The substances which produce heat when a fuel burns are chiefly carbon and hydrogen, and sulphur to a slight extent. Any oxygen which may be present causes a loss of heat, because it uses up its equivalent amount of hydrogen, leaving only the excess as a heat producer. Moisture in the coal, together with that prodcued by the combination of any oxygen present, takes up heat in its conversion into steam, and of course the mineral matter or ash is a non-producer of heat. Calculated heating values do not, as a rule, agree very closely with practical tests or even with calorimetric results.

The formula circulated by the Govt. of India for calculation of useful heat value *for non-coking coals* only is :

Useful heat value $= 8900 - 138\ (A + M)$

Where the U.H.V. is in kilocalories per kg of coal, A is ash % and M is moisture %. Both relative humidity and 40°C temperature as per ISS : 1350 – 1959.

If the non-coking coal contains less than 2% moisture and less than 19% volatile matter, from the value calculated by the above formula, deduct 150 kcal/kg for every 1% V.M. below 19% and that gives the actual U.H.V. for the coal.

Table 3

1. *NON-COKING COAL* (Except Andhra, Assam, Arunachal, Meghalaya and Nagaland)

Grades		UHV Range
A	–	Exceeds 6200 Kcals/kg.
B	–	From 5601 to 6200 Kcal/kg.
C	–	From 4941 to 5600 Kcal/kg.
D	–	From 4201 to 4940 Kcal/kg.
E	–	From 3361 to 4200 Kcal/kg.
F	–	From 2401 to 3360 Kcal/kg.
G	–	From 1301 to 2400 Kcal/kg

Long flame coal (as per Table 2 of IS no. 770-1964). V.M. and gross C.V. are on unit coal basis.

Group	V.M.	Gross C.V.	Moisture % at 60% R.H;40°C
B-4	over 32	8060 - 8440	3 - 7
B-5	over 32	7500 - 8060	7 - 14

2. *COKING COAL*

Range of Ash %

Upto 15% Steel - I

Exceeding 15% upto 18%	Steel - II
Exceeding 18% upto 21%	Washery-I
Exceeding 21% upto 24%	Washery-II
Exceeding 24% upto 28%	Washery - III
Exceeding 28% upto 35%	Washery-IV

3. Semi Coking Coal Grade 1 is with ash + moisture% upto 19

   Semi Coking Coal grade 2 is with + moisture% 19 to 24

4. Hard coke, B.H. Ordinary; ash upto 36%

   Hard coke, B.H. Superior, ash upto 31%

Ungraded coals are coals whose ash, or ash + moisture content exceeds the above specifications but such ash or ash + moisture does not exceed 50% of coal.

Rejects are coals whose ash or ash + moisture is in excess of 50% of the coal.

Ash% of coking shall be determined after air drying as per ISS : 1350 - 1959. If the moisture so determined is more than 2%, the determination shall be after equilibrating at 60% relative humidity and 40°C temperature. Coking coals gradation is on ash% only.

Non-coking coal is one which does not belong to the category of coking, semi-coking or weakly coking.

No grades are fixed for coals of Andhra Pradesh (Singareni group of mines).

The proximate analysis of some coal seams is as follows :

Kargali top seam (Sawang Colliery) excluding bands, air-dried basis, percent; Moisture 0.7-0.8; Ash 18.4 - 19.7; Volatile matter 28.4 - 30.4; Fixed carbon 62.5 - 49.1

Purewa top seam (Singrauli Coalfield), on equilibrated basis i.e. 60% R.H. & 40°C excluding dirt bands, percent; Moisture 5.3 to 8.5; Ash 22.6 to 40.0 ; volatile matter 24.0 to 30.9 ; Fixed Carbon 48 to 21; Calorific value kcal/kg. 3300 to 4200.

The general characters of coal of the Barakar and Raniganj Stages are shown below :

Barakar	*Raniganj*
Low moisture (1 to 2% in some case upto 3%)	High moisture (3 to 8% or more)
Low volatile (20 to 30%)	High volatile (30 to 36%)
High fixed carbon (56% to 65%)	Medium fixed carbon (50 to 60%)
Excellent steam coal and often excellent coking coal.	Generally poorly coking, though some are moderately so; good gas coal and long flame steam coal.

Coking Coal

Some types of coal, when heated in the absence of air to a temperature above 600°C, form a solid, porous residue called *coke* and the coal so heated is said to be *carbonised*. Coal which is capable of carbonisation is said to possess *coking* property. There are two processes of coal carbonisation.

1. Low temperature carbonisation
2. High temperature carbonisation.

In the **low temperature carbonisation** the coal is heated in the absence of air to a temperature of about 650°C. The coke formed is not strong enough for metallurgical purpose but is soft and can be used as domestic fuel, nearly free from smoke. If the process is carried on in a plant which recovers the gases of combustion, such gases yield a few by-products and chemicals *e.g.* at Neyveli Lignite Complex. In the collieries, however, low temperature carbonisation is carried on in a crude way, only to form soft coke without recovering the gases which escape into the atmosphere. The simplest way is to stack steam coal, set it on fire and cover it up with slack coal so that the steam coal burns without contact with air. After burning for a period of 36 to 48 hours when the burning stack partically ceases to produce smoke, it is quenched with water and soft coke is produced.

Two low-temperature carbonisation plants for bituminous coals have been set up in the country *(i)* Ramkrishnapur in Andhara Pradesh with coal input of 900 te/day, and *(ii)* Dhankuni in West Bengal with coal input of 1500 te/day, to produce nearly 0.35 million te of soft coke per year and nearly half a million m^3 of standard gas per day.

In the **high temperature carbonisation** coal is heated in absence of air to a temperature of 1000°C 1100°C in specially constructed ovens. The gases of combustion are allowed to escape into the atmosphere in the beehive coke oven plants, (e.g. at Ena colliery and many other collieries in Jharia field) but in the by-product recovery plant they are recovered to obtain a number of chemicals, *e.g.* at Bararee, Loyabad, Lodna, Bhowra in Jharia coalfield. By product recovery plants are installed at all the steel plants, Giridih colliery and at Durgapur Projects Ltd. at Durgapur. The coke formed after allowing the coal to burn for 30 hours and then quenching it with water is hard and suitable for metallurgical purposes. The coke oven gas formed during H.T. carbonisation is used as a gaseous fuel in the steel plants and in some other industries.

Table

Yield of products per tonne of coal, on carbonisation

Product	H.T.C.	L.T.C.
Coke	0.70 to 0.74 te Hard Coke with 1-3% V.M.	0.75 to 0.8 te smokeless coke with 8-12% V.M.
Gas	280 m³ to 360 m³ of 4000 to 5000 kcal/m³	110 to 170 m³ of 6000 to 7200 kcal/m³
Coal tar	22 / to 45 /	80 / to 110 /
Benzol	45 / to 65 /	80 / to 160 /
Ammonium	Sulphate 11 kg	5 to 6 kg.

Hard Coke Manufacture in Bee Hive Coke Ovens

There are two types of Bee Hive coke oven plants (or batteries).

1. Ordinary or conventional type (called *Sabji Bhatta* in Hindi coloquially).
2. English type (sometimes called *Tata Bhatta* or *CFRI Bhatta* coloquially).

In external appearance both ovens look alike but the difference lies mainly in the manner of circulatiion inside the oven of the smoke and hot gases resulting from burning of coal. In the English type, there are flues *i.e.* passages for hot gases below the refractory brick floor of the coal bed (but not on the sides) and the coal is better heated within a short period compared to the ordinary oven, in which the hot gases and smoke do not so circulate before escaping into the atmosphere. The burning period of raw coal for conversion into hard coke, called *coking cycle*, is nearly 36 hours in English ovens and it can be reduced to even 24 hrs with better quality of coal. In the ordinary ovens the normal burning period is 72 hours. The conversion factor for the two types is : In English ovens 4.0 te raw coal yield 3 te hard coke; in the ordinary type 4.5 te raw coal yield 3 te hard coke. The capacity of each oven varies from 3.5 to 5 te of raw coal. In both types of batteries, the ovens are constructed back to back and the two ovens have a

common chimney. For economical operation, a battery should consist of minimum 12 ovens. Raw coal has to be charged in a crushed form (-10 mm size) to the ovens.

In a by-product coke oven plant the primary derivatives from coal, by weight, are :

1. Coal tar, 3% yield on coal, approximately i.e. 30 kg/te of raw coal.
2. Crude benzole, approx. 0.8% yield on coal.
3. Ammonium sulphate, approx. 1.1% yield on coal. Crude pyridine production is 0.006% yield on coal.
4. Hard coke, 0.70 to 0.74 te per te of raw coal.

The secondary derivatives from the above by-products are napthalene, phenols, creosote oil, benzene, pyridine, *etc.*

The tests for coking coals cover those properties of coal that usually enter into consideration of its suitability for carbonisation, *etc.* These are :

1. Caking Index.
2. Proximate analysis.
3. Chemical composition of coal ash.
4. Fusion range of coal ash.
5. Swelling test.
6. Plastically test.
7. Determination of undesirable salts.

Caking Index

The caking property of coal is represented by a numerical called *caking index* (also called *agglutinating index*). The figure represents the maximum ratio of sand to coal in a mixture which, after carbonisation, gives a coherent mass capable of supporting a 500 g weight. The sand should be of a specified quality. The maximum ratio of sand and coal is determined in the laboratory by a process of elimination of mixtures of different ratios. For this test the weight of coal and sand mixture, as prescribed by the Indian Standard Specifications is 25 g, but the procedure adopted by Tata Iron and Steel Co. uses only 5 g of mixture. It is essential that the loose powder produced by the weight should not exceed 5% of the weight of the sand : coal mixture. The caking index gives only a relative idea of the capacity of coal to yield coke. For production of hard coke suitable for metallurgical purposes, the coal should possess caking index of 22 and above as determined by British Standard procedure. * Coal, which by itself gives, on carbonisation, coke suitable for metallurigcal purposes, is called *prime coking coal* and has a caking index of 22 and above. A medium coking coal is that which gives coke slightly inferior to metallurgical coke. Such coal has caking index between 17 and 22. Semi coking coal (also called blendable coal) falls much short of the requirements of prime coking coal and has a caking index of 10 to 17; for example, coal in Mugma coalfield. Semi coking coal produces reasonably good soft coke. Though caking index is a major criterion to decide the coking quality of coal, there are other qualities, as given below, which a coking coal should possess.

i. It should not swell on carbonisation; otherwise walls of coke oven will be damaged.

ii. It should have low phosphorous (less than 0.15%) and low sulphur (less that 0.6% content).

iii. it should have carbon content high enough (above 58%) to give coke with minimum 75% carbon.

iv. it should have low ash content nearly 17 to 18% or below. The maximum ash that can be tolerated in the coke is 22.5%.

v. V.M. content should not exceed 26%.

vi. it should be able to yield coke of certain physical characteristics given below (I.S.S. - 439 : 1965)

(a) Shatter index - over 85

(b) Haven stability - over 40%

(c) Porosity - 35 to 48% and above.

(d) Micum index, total on 40 mm, % by wt. (minimum) – 75

(e) Breslau hardness - 80 or over (indicating hard coke)

(f) size - 38 mm to 160 mm.

* Tata Iron & Steel Co. follow their own caking index.
Tata's index × 1.5 = BSS caking index, i.e. caking index 14, as per Tata's procedure = 21 as per BSS procedure. In the text BSS index is used.

Plasticity test and the Plastometer

When heated in the absence of air coking coals first become plastic and then solidify again on further heating. On an average the coals soften between 320°C and 350°C, attain maximum fluidity at temperatures between 350°C and 400°C, and solidify again between 400°C and 425°C. the temperature of softening and solidifying and the degree of plasticity vary from coal to coal. Non coking coal do not show any plastic behaviour. Property of coking coal to become plastic on heating in absence of air is known as its **Plasticity** and it is used as an index of its coking capacity. Although caking index, free swelling index, *etc.* indicate the coking property of coal, they do not reveal small changes in the coking property. However, they are adopted for routine testing because of their speed. Plasticity tests are generally time-taking and hence they are used mostly for periodical checking of individual coals which go into the oven for carbonisation, and in a steel plant, for the routine testing of one or two coal-mix samples in a day.

It may be mentioned that there is not optimum value of plasticity index for obtaining a good coke. It is also observed that :

1. Plastic property of coal improves on washing.
2. No relationship can be established between the maximum fluidity of coals and their V.M. and F.C. content.
3. Caking index does seem to be related to the fluidity of coal.

Hard coke containing ash 20% or less by wt. is classified as Gr. I coke; more than 20% but less than 24% ash is Gr. II coke.

The following **coal seams** produce **prime coking coals :**

Jharia Coalfield – IX, X, XI and upper seams into XVIII seam.

Giridih Coalfield – Upper Karharbari, Lower Karharbari and Bhadua seam.

Raniganj Coalfield – Chanch, Begunia, Laikdih, Shyampur-5, Ramnagar, Khudia.

Medium coking coal is available from the following coal seams *after washing.*

Jharia Coalfield – V to VIII, Mohuda Top and Mohuda Bottom.

Raniganj Coalfield - Dishergarh and Sanctoria seams in some areas only.

Bokaro and Ramgarh Coalfield – Kargali, Kathara, Sawang, Jarangdih, Bermo, Karo, Uchitdih, Kedla VII and VIII of Ramgarh block one.

Kanhan Valley coalfield - Damua - Rakhikol. It is more on the verge of semi coking coal.

Assam coals have very low moisture and low ash (3-7%). They produce excellent hard coke but as they are high in sulphur, their use is not permitted in blast furnaces. Sulphur is present in Assam coals in organic form (3-7%) and therefore difficult to separate;; (V.M. 35-40%).

Reserves of prime coking coal, so essential for the manufacture of iron and steel, are limited in the country, but that of noncoking coal are plentiful. If the limited reserves of prime coking coal in the country get exhausted, the medium coking coal by itself will not be able to produce metallurgical coke and prime coking coal will have to be imported. Keeping this in mind the Steel Plants are manufacturing hard coke in their coke ovens from a blend of nearly following composition.

Prime coking coal - 60%
Medium coking coal - 32%
Blendable coal - 8%

The blendable coal is generally selected on its gas-evolution quality as sufficient gas has to be produced in coke ovens for various heating operaions in steel plants. For this reason Bhilai Steel Plant uses blendable coal of Dishergarh, Poniaty and other coal seams of distant Raniganj Coal Field in preference to relatively nearer Churcha seam (Chirimiri Coal Field) which is of low ash content, nearly 15% as the latter has comparatively low V.M. content.

The trend in the production of prime coking coal and medium coking coal (after washing) shows that it will not keep pace with the demand of planned increased production in steel plants. The Government is therefore importing prime coking coal at a very high price.

Coal Washing

The ash percentage in coal can be reduced by a process known as *coal washing* in coal washeries which are established mostly in Jharia, Raniganj, Bokaro and Karanpura coalfields for treating prime and medium coking coals. Impurities in coal are plenty and they are collectively known as ash. When coal is extracted in underground or opencast mines, external impurities get mixed up with it *e.g.* rock, clay shale and in the case of mines having sad-stowing, sand also gets mixed with coal during mine operaions. The impurities have higher specific gravities than good burnable coal. The operation of washing coal therefore aims at floating the coal and sinking the impurities in water. As a matter of fact, coal won't float on water. Coal has specific gravity of 1.28 to 1.3 against 1.0 for water. The fact that coal with a specific gravity of 1.28 to 1.3 sinks in water at one speed and its impurities with higher specific gravities sink faster, provides the basic principle for coal washing. The difference in settling rates is used in a number of commerical washing devices or washers. A fluid mixture of sand (Sp. gr. 2.6) and water in which the sand particles are kept in an agitated form without permitting them to settle, has a specific gravity of nearly 1.60, so that coaly matter floats over such mixture and the impurities settle down. This is the principle adopted in Chance coal washer, the first coal washer in India. In this process water is pumped into the cone (Fig. 2.3 B) through its sloping sides to produce a rapidly rotating, rising water flow into which an overdense sand suspension is added in quantities to achieve the desired sp. gr. in the mixed flows. Raw coal is added to the top of the cone. The clean is removed by a weir in the surface of the suspensions and sinks are removed by a lock hopper in the base of the cone.

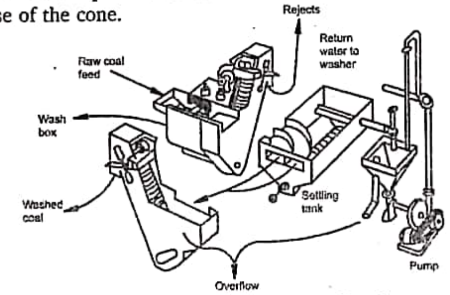


Fig. 2.1 Schematic view of a coal washer

In another method of washing, the process of Jigging is employed to separate good quality coal from its impurities. **Jigging** is a simple process of particles-stratification in which the particle rearrangement results from an alternate expansion and compaction of a bed of particles by a pulsating fluid flow. The bed of particles is carried on a performed deck through which water passes upward and downward in repeated *pulsion and suction* stroke (30-50 time per minute) and at the same time the bed is moved forward throgh the jig chamber in a lateral direction by the horizontal flow of water. The stratified bed is cut at various heights by sub-surface gates to remove shales and middlings.

The raw coal fed to the Chance washer and the jig washer is + 6 mm and – 100 mm in size.

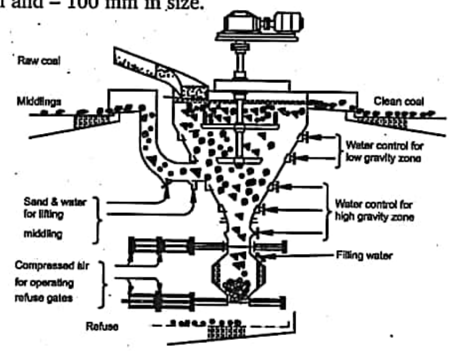


Fig. 2.2 Chance process of coal washing using sand and water

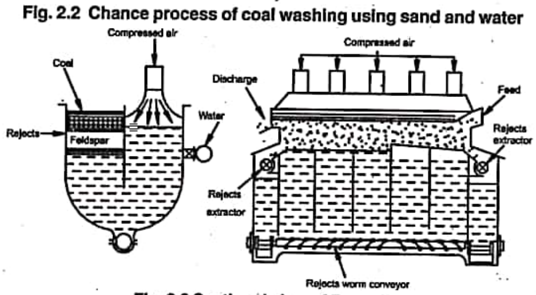


Fig. 2.3 Sectional view of Baum Jig.

Ultrafine coal particles of -28 mesh size are cleaned by a process known as froth flotation. The process is generally inapplicable to low-rank coals. In principle, froth flotation consists of bubbling air through a suspension of fine coal and water to which various chemicals are added with a view to produce stable froth. Coal particles attach themselves to the froth but not the particles of rejects because of the preferential physical attachment of air bubbles to coal particles. The froth, with the attached coal particles, floats to the surface and is removed.

The material that forms ash in coal is finely disseminated over the entire coal mass in Indian coals unlike in the coals of other foreign countries like Britain, Germany, U.S.S.R., etc. The reason for this is the manner in which our coals have been formed as explained in the Drift theory. Indian coals, therefore, cannot be washed as easily as foreign coals in commerical washers. To separate the ash from the heat-producing coaly material, the coal is crushed to a small size along with attached extraneous impurities, before feeding to the washer. After washing, the end products are washed (also called *clean*) coal, middlings, and rejects in a 3-products washery and only washed coal and rejects if it is a two-product washery. The clean coal has much less ash percentage compared to that of the raw coal and it is despatched to steel plants for use in the captive coke oven plants or to other consumers interested in low ash coal. The middlings, though higher in ash content, have sufficient heating value and can be used it steam boilers designed for high ash coal. The rejects are discarded near the washery. The yield of clean in a washery may be 60 to 75% of the raw coal depending upon the quality of input.

Uses Of Coal

Chief use of coal in industry is as a fuel for steam raising in the boilers in power houses and locomotives, gas producers, brick kilns, furnaces and for household purposes in the form of soft coke. Boilers of railway locomotives and most of the Lancashire boilers used for steam raising are fitted with fire grates which are suitable for coal of large size. Hence coal over 50 mm in size is called steam coal; coal of 25 mm to 50 mm in size is called rubble coal. Slack coal is that which is smaller than 25 mm in size though sizes upto 50 mm are also sometimes called slack coal. Run-of-mine coal is that which is not screened or sized and therefore contains steam, rubble and slack sizes.

Table - Coal Washeries in India

The name of the owning Company is in the bracket followed by the yearly input capacity of the washery, in million tones. (M te)

Existing washeries

Prime coking

Jamodoba (TISCO) 1.72. Lodna (BCCL) 0.40. Durgapur (HCL) 1.50. Dugda-I (BCCL) 2.40. Bhojudih (BCCL) 2.00. Patheridh (BCCL) 2.0, Durgapur (DPL) 1.35. Chasnala (IISCO) 2.00). Dugda-II (BCCL) 2.40. Sudamdih (BCCL) 2.00. Monidih (BCCL) 2.00.

Medium coking

West Bokaro (TISCO) 0.57. West Bokaro, new (TISCO) 1.80. Karagali (CCL) 2.32. Kathara (CCL) 3.00. Sawang (CCL) 0.75. Gidi (CCL) 2.84. Barora (BCCL) 0.42. Nandan (WCL) 1.20.

Non-coking

Nowrazabad (WCL) 0.5

Total installed capacity 33.57 M te.

Proposed washeries, Construction by VIII th 5yr. Plan.

Prime coking

Madhuband (BCCL) 2.5. Bhalgora-I (BCCL) 2.5. Pootkee (BCCL) 3.0. Dharmaband-I (BCCL) 2.5. Dharmaband-II (BCCL) 2.5. Bhalgora-II (BCCL) 2.5. Mukunda (BCCL) 4.0

Medium coking

Bharatpur (SECL) 3.50. Jhingurdha (NCL) 3.00 Bina (NCL) 4.50. Jagannath (SECL) 2.00. South Balanda (SECL) 1.65. Mukunda (BCCL) 5.0. Rajmahal (ECL) 10.000 Kalinga (SECL) 5.00 Ananta (SECL) 3.0 Piparwar (CCL) 5.00 Dudhichuna (NCL) 5.00 Selected Dhori (CCL) 2.25. Kakri (NCL) 2.50 Jayant (NCL) 7.00 Khadia (NCL) 5.00 Nigahi (NCL) 5.00 New Majri (WCL) 1.00 Lajkuara (SECL) 1.00 Ib. (SECL) 1.00

The boilers for steam raising in large power houses are designed for inferior (high ash) coal in pulverised form. The coal that can be easily *ground or pulverised* has a high Hardgrove grindability index and hard coals have low H.G.I. For boilers using pulverised fuel, coal should have H.G.I. of 50 and above and low caking index. Moreover the ash should have initial deformation temperature (I.D.T.) above 1100°C, the hemispherical point over 1100°C, the hemispherical point over 1100°C and flow point over 1200°C. Lower figures result in clinker formation which restricts air flow to the coal burning on fire gates and adversely affects coal combustion. For these reasons the data required above coal by power house design engineers relate to the following points.

1. Proximate analysis.
2. Ultimate analysis.
3. Hardgrove grindability index of coal.
4. Composition of ash.
5. Initial ash deformation temperature.
6. Hemispherical temperature of ash
7. Ash fusion temperature.

Of all the constituents of ash the ones which matter most are iron and managanese oxide constituents as they lower the fusion temperature. Almost all ashes start softening above 1200°C and completely fuse at temperatures higher than 1300°C.

Coal is used for manufacture of hard coke which serves for heating purposes in smithy shops and in the furnaces. The hard coke also acts as a reducing agent when smelting oxidised iron or like Fe_2O_3 in the blast furnaces.

Gases produced from coal during carbonisation, if recovered in a by-product recovery plant, can be used for production of many chemicals, explosives, plastics, inks, paints, perfumes, fertilizers (*e.g.* at Talcher Fertilizer Plant) and also used for production of synthetic petrol. The **gaseous fuels** available form coal/coke are :

1. Coke oven gas 2. Producer gas 3. Water gas

Coal gas or coke oven gas - the process of manufacture involves complete gassification of coal subject to the limitations of the plant utilised for manufacture. The carbonisation is carried out by heating the cola in the absence of air in horizontal or vertical retorts if gas is the desired major product, or in the coke ovens if coke is the major product required. Either method produces gases with similar compositions and a solid residue with an ash content that is higher than the original coal. The main differences in the gas making processes are in the quality of coke and in the quantity and character by-products. Typical analysis (% by volume) after the gas is washed is as follows : $CO = 5.8$; $CO_2 = 1.5$; $N_2 = 1.0$; $H_2 = 57.6$; $H_2S = 0.7$; $CH_4 = 29.6$; $C_2H_6 = 1.3$; $C_2H_4 = 2.5$.

Heating value – of coal gas and coke oven gas is between 4000 and 5000 kcal/m³. Such high heating value makes its transport over long distances economical. Calcutta receives all its gas supply from the coke oven plant of Durgapur Projects Limited, 160 km away through 300 mm dia. pipes. Coke oven gas is produced in large quantities at integrated steel plants making their own hard coke in coke oven batteries. The coke is used in the blast furnaces, and the coke oven gas along with blast furnace gas (heating value of B.F. gas nearly 850 kcal/m³), supplies heat energy for the plant functions.

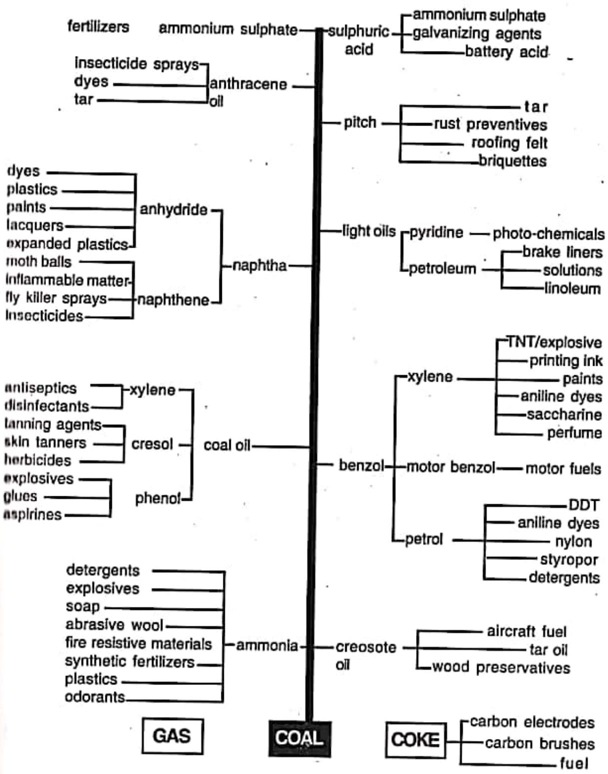
Producer gas – It is formed by blowing a mixture of steam and air through a bed of hot coal and therefore its nitrogen content is very high and its heating value low. Because of its low heating value it is not transported any great distance and is used within or near the plant that produces it. Gross heating value of producer gas manufactured from bituminous coal normally ranges from 1250 to 1600 kcal/m³. Range of gas composition is as follows (% by Vol.) : $CO = 20 - 30$; $H_2 = 8 - 20$; $CH_4 = 0.5 - 3$; $CO_2 = 3 - 9$; $N_2 = 50 - 56$; $O_2 = 0.1 - 0.3$.

Water gas – Water gas is made in a manner similar to producer gas except that steam only is passed through the hot carbonaceous material. Hot air is first blown through the fuel bed to raise the fuel bed temperature to the the desired level and steam is blown through until the temperature drops sufficiently to virtually stop the reaction. Then air is blown through again to increase the bed temperature. Because water gas is made up mainly of H_2 and CO, it burns with a characteristic blue flame. Heating value is approximately 2600 kcal/m³. Analysis % by vol : $CO_2 = 5.1$; $CO = 40.2$; $H_2 = 50.0$; $CH_4 = 0.7$; $N_2 = 4.0$.

Coals suitable for gas production are those having high V.M and low caking index *i.e.* absence of the tendency to form clinkers. Coals from most of the seams in Raniganj coalifield are suitable for gas

generation. Coals of HIngir-Rampur colliery and of Ghugus, Ballarpur and nearby collieries in the Wardha Valley coalfield are also suitable for gas plants. Coals from Kanhan valley coalfield in M.P. have a comparatively high caking index and as they form clinders, they are not considered desirable for gas production.

PRODUCTS AVAILABLE FROM COAL



The norms of consumption of coal in some leading industries are as follows :

(a) *Iron & Steel :* per tonne of pig iron produced, 1 te of hard coke equivalent to 1.3 te of washed coking coal (18% ash) or 2.2 to 2.8 te of raw coking coal; For every 1% additional ash in coal the consumption of coal for the blast furnace goes up by 7% to 8%.

(b) *Power Plants* : Inferior coal 3 tonnes per annum per kW of installed capacity; noncoking; in boilers with pulverised coal firing.

(c) *Cement Manufacture :* 1 te of coal for 3 tonnes of cement; non coking , slack.

(d) *Paper :* 2.25 te of coal per te of paper and 1.3 te of coal per te of pulp (in new plants); non-coking.

(e) *Textiles :* 0.38 te of coal per 1,000 m of cloth; non-coking.

(f) *Rly. Locos :* Non-coking steam coal.

(g) *Sheet glass :* 1.46 te per te of glass manufactured.

(h) *Brick Burning :* For 1 lakh bricks 12-18 te; non-coking, slack.

(i) *Tile Burning :* For 1 lakh tiles, 20 te; slack; non-coking.

Petrol From Coal

The industrially advanced countries of the world have developed technology of producing petroleum and gas from the tradtional fuel, coal. Chemically, petroleum oil has a higher hydrogen-carbon ratio than coal; this is responsible for its lightness, mobility and versatility as fuel and use in chemical processes. To convert coal into petroleum the main requirement is to increase the proportion of hydrogen by synthesis. This hydrogenation was first achieved by the German scientist Bergius in 1914.

The **processes** which can be considered suitable for preparing petrol and allied chemicals from coal are :

1. Hydrogenation of coal, *e.g.* Bergius Process.
2. Solvent extraction of coal, *e.g.* Pott Broche Process.
3. Hydrogenation of tar obtained from pyrolysis/carbonisation of coal, *e.g.* Fischer Tropsch Process.

Of these processes hydrogenation of coal was adopted on a large scale in Germany during the II world war. It was named after its inventor, Bergius, as **Bergius process.** During the war years Germany was producing by this process nearly 5.0 million tonnes of synthetic petrol to cater to its war machinery as the outside natural petroleum supply was cut off. The process involved inmixing powedered coal with a vehicle oil (which is derived from the process) and subjected to hydrogen pressure based on iron bad poor activity and plant costs were very high because of the high pressure of operation. About 2.5 te of coal would produce one te of petroleum products. Due to the unfavourable economics the plants were all shut down after the war.

Another low-cost synthesis of oil is now finding popularity in Japan, the U.K. and some other countries. In this process powedered coil is carbonised to produce large quantities of tar and gas. The tar distillates are then hydrogenated to give petroleum products like gasolene, kerosene and diesel oil as well as other chemical byproducts.

In India the C.F.R.I. has conducted experiments on Assam coals which are easily amenable to hydrogenation for conversion into petrol. But the initial experiments have proved the economics of such conversion not favourable.

Besides the production of petroleum from coal there are now two more revolutionary trends in the synthetic fuel sector using coal as the base, namely (1) hydrogasification of coal and (2) *in-situ* gasification of coal. In hydrogasification the coal is gassified with oxygen and steam at higher pressure producing, when purified, a high energy gas consisting of methane and hyrogen.

Under *in-situ* gasification the underground seam of coal is totally gasified while oxygen or air is supplied through the shafts and large dia. boreholds or galleries. The ash remains underground while the low-grade gas is used in gas turbines generating electricity. Several power stations are being operated in the U.S.S.R. by this process from thick seams of brown coal (lignite).

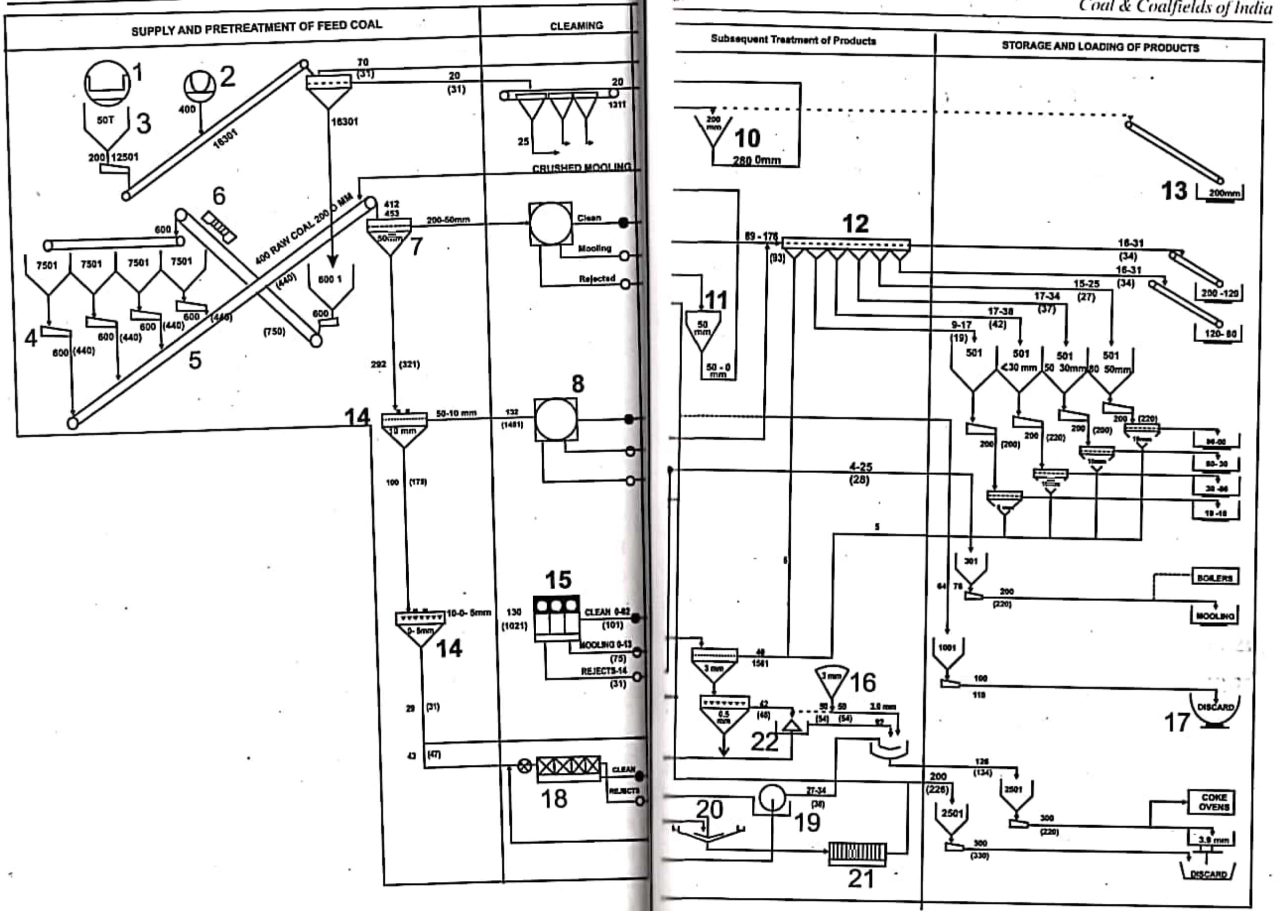


Fig. 2.5 Typical equipment & flow sheet in a coal washery
(AfterIS : 5209 – 1968)

Reserves Of Coal

The quantity of coal available in a coal seam *in-situ* is called its reverses which are generally expressed upto a specified depth. The reserves are categorised into three classes depending on the relative reliablity of data.

(a) ***Proved reserves :*** These are reserves estimated from dimensions of the coal seam revealed in outcrops, trenches, mine workings and boreholes by assuming their continuity of geological evidence for a distance upto 200 m from the point of observation.

(b) ***Indicated reserves :*** These are determined by taking an area of 1000 m × 1000 m for each point of observation where coal seam is indicated. This may be extended to 2000 m × 2000 m for known geological continuity.

(c) ***Inferred reserves :*** This is an estimate of coal based largely on broad knowledge of geological character of beds but for which there is ot measurement.

The *minerable reserves* or extractable reserves are those which can be economically extracted with the present knowledge of technology and with due regard to mining Acts and Regulations; for example, in working a coal mine solid coal has to be left *in-situ* in the following cases.

1. Vertically below towns, railways, canals, lakes, rivers, buildings and public roads and other important surface features and installations and 50 m beyond if not stowing of underground mines is possible.
2. Below and above a coal seam on fire.
3. For protection of shaft pillar, panel barriers and roadways which have to be maintained till the life of the underground mine.

4. Underground barriers against accumulations of water, gas or fire areas.
5. Barrier of coal against railway, road, river, canal, town, etc. in opencast mines. Coal is also blocked up at the boundaries of an opencast mine as the quarry sides have to be sloping or kept in the form of long benches or long steps.
6. Barrier at the common boundary of adjacent mines.
7. Barrier near a fault; burnt or inferior coal near dykes, sills, etc.

Mineable reserves cannot be extracted centpercent and some loss in inveitable in the methods of mining adopted so that the total production available from a coal seam many vary from 50 percent to 90 percent of mineable reserves. In the calculation of reserves shale or dirt bands 50 mm and above are excluded, but if highly mechanised opencast mines are planned, bands of 1m thickness and above only are excluded and reserves are estimated separately for coal : overburden ratio of 1:1, 1:2, 1:3, 1:4 and 1:5.

Geological maps of three prominent coalfields are given in the following pages.

Jharia Coalfield

There are 18 coal seams belonging to Barakar measures. No. 1 seam is the bottom-most and No. XVIII, the top-most seam. The thicknesses of the seams are as follows in metres.

1-2.1-4.3; II - 2.1-18.3; III - 2.1-7.3; IV - 2.1-12.0; V/VI/VII - 11.9-22.6; VIII - 2.1-8.8; VIIIA - 1.8-6.0; IX - 1.2 - 11.6; X (IX and X combined) - 3.1 - 18.3; XI - 1.2-7.6; XII - 1.5-5.5; XI - XII (combined) - 2.7-2.4; XIII - 2.1-9.1; XIIIA - 2.4-4.4; XIIIB - 1.5-4.0; XIV - 1.6-10.7; XIVA - 1.5-3.7; XV - 2.7-11.9; XV A - 1.1-2.7; XVI - 1.2-4.9; XVIA - 1.8-3.0; XVII - 1.2 - 5.8; XVIII - (top most) - 1.8-5.2.

In the Raniganj measures of Jharia coalfield there are only a few coal seams as follows :

Three seams of Lohapiti area	1.2 m each
Two seams of Kachara area	0.9-1.3 m
Bhurangiya	1.2 m
Mohuda top	2.7-3.7 m
Mohuda middle	1.2 m
Mohuda bottom (lowest seam)	1.8-2.4 m

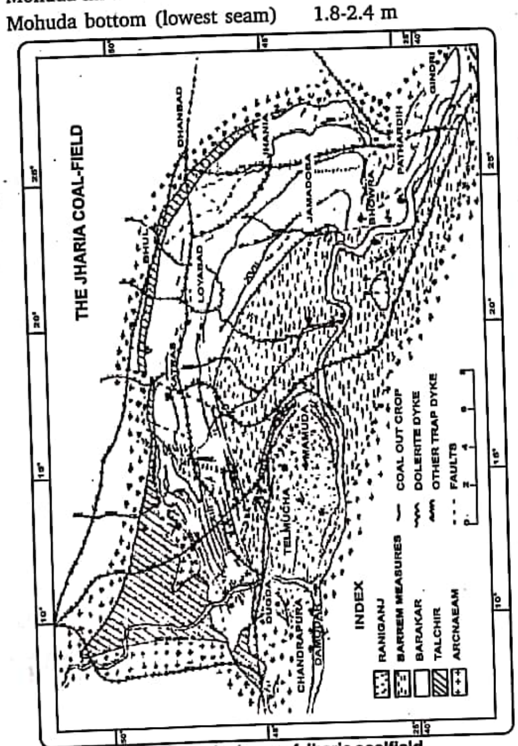


Fig. 2.6 Geological map of Jharia coalfield

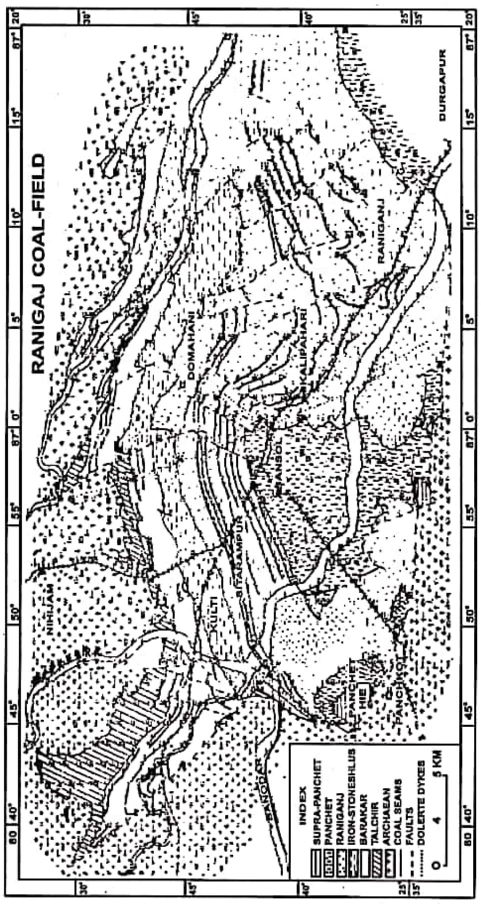


Fig. 2.7 Geological map of Raniganj coalfield

Raniganj Coalfield

In this field the same seam is known by different names in different localities as stated below :

Raniganj measures

Hirakhum - Bharatchak - Narsamuda - Kalla (1.5-3m)

Gopalpur - Upper Dhadka - Satpukuria - Ghusick - Siarsol - Upper Kajora - Khandra (2 m - 6m).

Barachak - Nega - Raniganj - Lower Kajora - Jambad - Bowlah - Bankola (5m - 14m)

Lower Dhadka - Narainkuri - Bansra - Sonachora - Bombahal (2m - 3m)

Sripur - Toposi - Kenda - Chora - Purushottampur)2m - 9m).

Bara Dhemo and Raghunathbati - Manoharbal - Rana - Poriarpur - Satgram - Jotejanaki - Dobrana - Darula - Sonpur (2m - 9m).

Hatnal - Koithi (2m - 4m)

Sanctoria - Poniati - Bamanbad (1.5 m - 5 m)

Taltor - Gangutia (1.5m), the bottom most seam.

Barakar measures

Chanch-Begunia-Shampur 1 (3m - 4m).

Shampur 2 (3.5 m)

Shampur 3-Kharbari (4.3-4.6 m)

Shampur 4-Ramnagar (4.8 m).

Laikdih - Shampur 5 (5.4 m - 13 m).

Gopinathpur - Salanpur C - Kasta (Upto 13 m).

Bindabanpur - Salanpur B (2 m - 8m).

Damagaria - Salanpur A - Kalimati (1.5 m - 45 m).

Pusai (1 to 8 m), the bottom - most steam.

The general characteristics of coals of Barakar and Raniganj measures and coking nature of different seams in the two fields have been stated in the earlier pages.

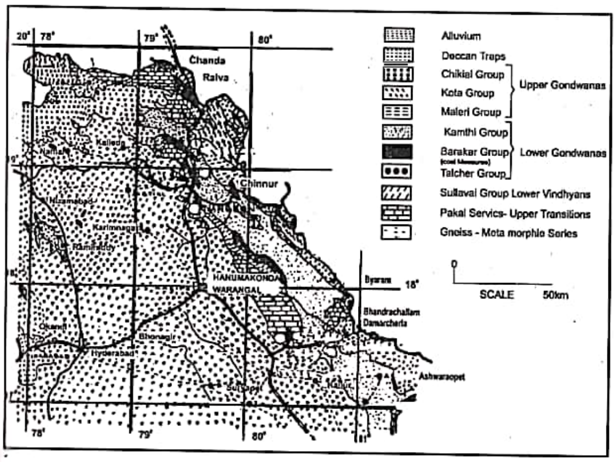


Fig. 2.8 Geological sketch-map of Pranhita - Godavari Valley coalfield.

1. Tandur and Kanala coalfield. 2. North Godavari coalfield. 3. South Godavari coalfield. 4. Yellandu coalfield. 5. Kothagudium coalfield.

Pranhita - Godavari Valley Coalfield

In this coalfield which is an important coal producing region in South India the coal seams have high moisture and high v.m. content. The seams in different areas are :

Tandur and Kanala area.

Top seam (or Salrjung seam) 8.2 m

Bottom seam (or Ross seam) 2.1 m

North Godavari area

No. 1 seam	2.4 to 3.0 m
No. 2 seam	4.0 to 4.5 m
No. 3 seam	2.1 to 3.3 m
No. 4 seam	1.8 to 2.4 m

South Godavari Area :

No. 1 seam 6.4 m , No. 2 seam, 3.5 m. No. 3 seam, 7.0 m. No. 4 seam, 4.8 m.

Kothagudium area

Top seam (or thick seam or Queen seam) 10 to 30 m.

Bottom seam (or King seam) 2.2 m in the north to 31 m in the south.

Singareni area

Top Seam (or Queen seam) 10 to 30 m

Bottom seam (or King seam) 2.4 m.

Queen seam (7.17m) is available in Kothagudem and Yellandu. At Manuguru a thick seam (9-19 m) is available.

Coal mining activities started in this field in 1889 and Singareni Colieries Co. Ltd. which works all the miners of this field has its activities in four districts, Adilabad, Karimnagar, Warangal, and Khammam. Godavari Valley Coalfield has about 6% of national coal reserves and produces about 10% of national coal production. Geological reserves are estimated to be about 10,435 million tonnes to a depth of 1200 m from the surface. Of these, nearly 4937 million tonnes come under the category of proved reserves. SCCL has mined about 275.70 million tonnes of coal from Godavari Valley Coalfield in the last 100 years (1889-1989).

✧ QUESTIONS ✧

1. How is coal formed ? explain the theories that have been advanced in this respect.
2. How is analysis of coal expressed ? Give the grades of coal as decided by the Government of India. What is the difference between Raniganj series coals and Barakar series coals ?
3. What is coking coal ? Name the seams which produce coking coals?
4. What are the different uses of coal ? State the norms of coal consumption in various industries.
5. What are proved, indicated and inferred reserves of coal ? What are mineable reserves ?
6. State the circumstances in which coal has to be left *in-situ* in a mine.

✧ ✧ ✧

Chapter -3

Boring & Drilling

A study of the rocks exposed at the surface, their structures and the geology of the areas gives sufficient information of the conditions that may be available below ground. More detailed information is, however, obtained by boring and interpretation of the data available from it. The term *drilling* is nowadays coming into use for boring.

In mining an allied branches of engineering drill holes may be drilled for the following purposes.

1. To prove the existence of minerals, to get an idea of rock structures and to obtain knowledge of the rocks and the mineral beds, such as depth, nature, thickness and gradient.
2. To get core of the rocks from which gearing pressure of the ground can be found out. This information is necessary for heavy winding engine foundations required for deep shafts.
3. To know throw of the faults; this is conveniently done from underground workings.
4. To drain off gas or water from old workings.
5. To carry electric cables, signal wires, stowing pipes or water pipes to the underground from surface.
6. To have tube wells for water supply to colonies.
7. To blast rock/mineral in a mechanised quarry where holes of 9 to 18 m depth and 125 to 300 mm in dia. may be drilled.
8. To render possible the injection of cement into the strata in a method of shaft sinking known as "cementation method of shaft sinking".
9. For ventilation of underground mine workings.

An interesting application of bore holes drilled from the surface to seal off with incombustible material underground fire and thereby to quench it, was witnessed at Kurasia colliery (M.P.) in 1961.

Sizes of drill holes drilled in mining areas usually vary from 25 mm to 125 mm though larger diameter holes may be required for carrying stowing pipes, water pipes, or for blasting in mechanised quarries. For deep holes, the size is larger at the start and is gradually reduced with depth.

Method employed for drilling

The methods used are :

1. Percussive drilling,
2. Rotary drilling

The percussive method is further sub-divided as :

(a) Drilling by rigid rods.

(b) Rope drilling, also known as *cable drilling or churn drilling.*

The rotary method may be used with a view to get a core of the rocks passed through, or simply to drill a hole, and this decides the type of drilling tool used. The rotary method is subdivided as :

(a) Drilling by saw toothed cutter.

(b) Drilling by tricone rock roller bit.

(c) Drilling by diamond drill bit.

(d) Drilling by chilled shots.

Percussive Drilling

In this method which is the oldest one of drilling, the hole is drilled by striking a number at short in intervals on the rock by a chisel-type tool and between the blows the tool is rotated slightly. The rock is chipped away with each blow and a circular hole is formed. During drilling the chisel is suspended from the surface by rods or wire rope and the weight of the chisel, rods, etc. is utilised to give the striking force.

Drilling with rods

The rods are Ni-Cr or carbon steel. Each rod has a male screw at one end and a female screw (scrwed socket) at the other. Steel rods are usually in lengths of 3 m with nearly 38 mm × 38 mm square cross-section. For rotary drilling, described later, the rods are hollow circular in cross section, have flush joints, and the length varies from 0.5 to 3 m.

The drilling tool used varies greatly in shape and cutting edge according to the type of ground to penetrate. A few common types are shown Fig. 3.1. For soft surface deposits which consists of alluvium, clay auger and worm auger may be used. These are given a rotary rather than a percussive motion. The straight chisel is commonly used for hard strata and the V chisel and T chisel, for very hard strata.

Every type of drilling requires a derrick which may consist of three or four legs and may be of wood or tubular steel. It is used chiefly for lifting the rods from the hole with the aid of a winch. In the mining localities, some petty drilling contractors undertake percussive drilling of 50 mm to 75 mm holes upto a maximum of 25 mm depth with the help of rods without installing any derrick.

Manual Drilling

The general arrangement for manual drilling is nearly like that shown in Fig. 3.3 except that water is not supplied through hose pipes under pressure and the cranck operated beam is replaced by a rocking lever connected to the drill rods. The drilling rods are given a percussive motion with the help of a corking level to which they are attached through a stirrup and a brace-head. A brace-head is simply a pair of crossed handles fixed to the end of a short top rod which is screwed to the column of the rods. Two or three men press down the free end of the rocking leve, thereby lifting the rods while one man turns them slightly by means of the brace-head. The men then let go the free end so that the rods fall and the drilling tool gives a blow on the rock. Water is poured in the hole at intervals and the process is repeated. As the hole gets deeper, the rods are lowered in the stirrup by a screw, and when this can no longer provide for the increasing depth, a short rod is added to the column of rods and the screw runs back to repeat the same process of drilling. Instead of the stirrup, Dlinks may be used. Short rods are added till the depth drilled by such small rods is slightly more than the length of full-length rod and the short rods are then replaced by a full-length rod. A device known as retaining key is used at the time of raising or lowering the rods of square cross-section. The same purpose is served in the case of rods of circular cross section with flush joints, by a device known as "bulldog safety clamp".

During drilling the bottom of the hole soon gets filled with cuttings and has to be cleaned out frequently. This is done with the help of a sludger which consists of a long cylinder or pipe, open at the top and with a flap valve at its lower end. The flap valve opens upwards.

When attached to the end of the rods and worked up and down the sludger gets filled with the sludge. It is then withdrawn to the surface and the process repeated till the hole is cleaned. The cuttings brought to the surface in the sludger give an indication of the rock being drilled. The bottom of the drill hole is always kept full with water during drilling.

Table 1

Uses and limitations of common methods of exploration drilling :

Types of drill	Common Sizes of drill holes (mm)	Maximum economic depth of hole (m)	Types of bits used	Rock formations where used.
1. Percussive with rods	50-75	250-300	Chisel shaped	Sand and clay and sedimentary rocks of soft and medium hardness; in fissured formations; Does not give core.
2. Churn drilling or cable drilling	75-500	300-600	Steel choppy bits of many styles.	Placer deposits; evenly firm & moderately soft formations.
Rotary drilling				
3. Non-coring	100-300	500	tricone rock-rollers bits	Any rock formations except very hard.
4. Diamond drilling	30-200	100-3000	Diamond bits of various types tungsten carbide bits; tricone bits are mostly non-coring.	Any rock formation except fissured. Holes can be drilled from surface or underground working at any angle to the horizontal; used mainly for coring.
5. Calyx or chilled shot drilling	75-1800	upto 450	Calyx; chilled steel shots;	All rocks except the hardest; unsuitable for soft and fissured formations; can drill at angles upto 35° from the vertical.

Sometimes a rod or chisel breaks in the drill hole during drilling. Devices like the crow's-foot and the spiral worm (Fig. 3.2) may be used to catch the broken rod under a joint in the borehole. Broken pieces of chisel are sometimes raised with the help of powerful magnets. In diamonds drilling, described later, the diamonds sometimes become loose and fall in the hole. The operation of tracing the broken and lost parts in the hole and withdrawing them to the surface is known as *fishing the borehole.*

Lining a drill hole : During drilling a steel pipe is used for lining the drill hole from surface upto the hard rock, and the drilling tool and rods pass through the pipe. Obviously, the length of the drill-hole upto which the lining pipe has to be fitted should be of a larger diameter. The lining pipe is generally withdrawn after the hole is completed though it may sometimes be necessary to leave it in its position to prevent caving of sides, *e.g.* drill hole for stowing, water pipes, etc.

The lining is done by hammering first a special steel pipe with a cutting edge. Pipes of 6 m lengths and having screwed joints are added to that pipe.

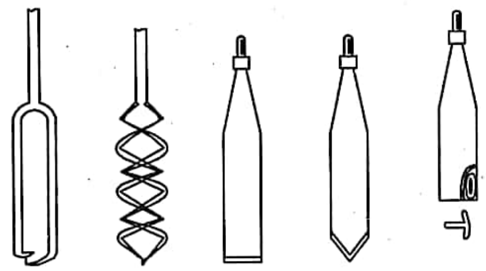


Fig. 3.1 Common types of drilling chisels and augers.
(A) Clay auger (B) Worm auger (C) Straight chisel (D) V-chisel (E) T-chisel

Power Drilling With Rods

Drilling with manual labour without the help of power is suitable for holes upto 150 mm in diameter and upto a depth of 30 m or so. Beyond that depth, it is impossible to drill without the use of power from a diesel or petrol engine, the common source of power in isolated drilling sites. Vertical boilers have sometimes been used to avail of steam power, specially where drilling had to be done in a colliery area where coal is easily available.

Fig. 3.3 shows the general arrangement where power is available for drilling with hollow rods. A power operated winch is used to raise and lower and the rods. The walking beam is operated by a crank through gearing from an engine to give the drilling tool 25 to 30 blows per minute and a stroke of nearly 225 mm. The beam is mounted on steel springs which give elasticity and cause sudden recoil of the frilling bit thereby preventing jamming. The rods are attached to the rope with a swivel attachment and bracehead. The rope of power-operated beam is slackened from time to time to keep the drilling tool in contact with the rock. Water is forced down the hole of the hollow drill rods by a pump to keep the cutting tool cool. Such water returns to the surface from the outside of the rods with the sludge. With this arrangement it is possible to drill a depth equal to the length of one full rod at a time. Water flushing practically eliminates the use of a sludger.

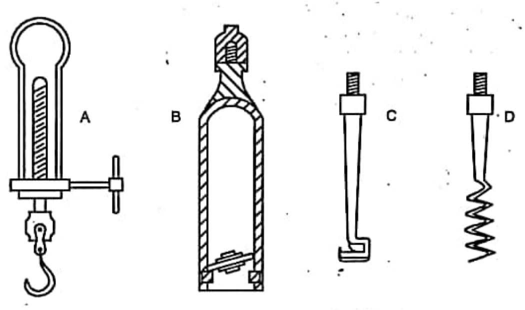


Fig. 3.2. (A) Stirrup and screw. (B) Sludger (C) Crow's foot (D) Sprial worm

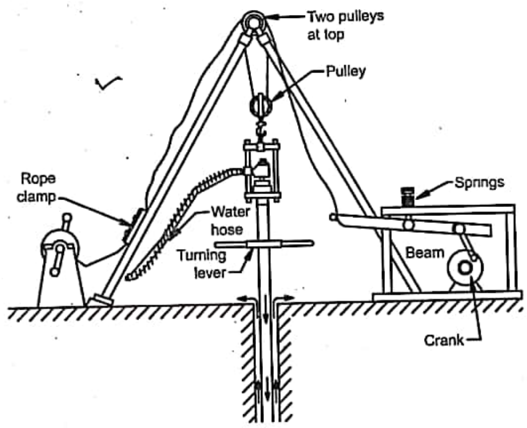


Fig. 3.3 Power drilling with rods. Arrows indicate water course

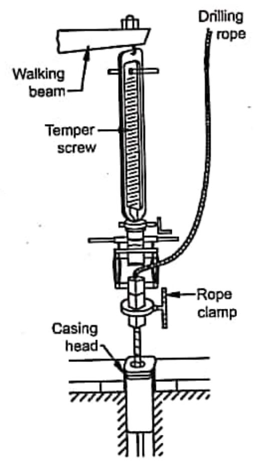


Fig. 3.4 Rope attachement in churn drilling

Rope Drilling Or Cable Drilling

Where the percussive method of drilling is employed cable drilling is commonly adopted for holes deeper than 30 m. In this system the rigid rods are replaced by a steel wire rope to which the drilling tool is attached. The surface arrangement is practically the same as for drilling with rods, but the end of the walking beam is attached to a temperature screw (Fig. 3.4). The rope from a winch is taken to the clamps of the temper screw across the pulley of the derrick.

Feed of upto 1.2 m is possible with the use of the temper screw. When no more feed is possible the temper screw is run back and the rope reclamped 1.2 m higher up.

During rope drilling no device is necessary to give a twist to the drilling tool between successive blows as the lay of the stranded rope causes the tool to twist slightly. The steel rope may be 18 mm diameter for a depth of 300 metres. The ropes have always a lefthand lay, so that the spin of the rope which tends to rotate the drilling tool also tends to tighten the joints between them.

Cable drilling is also known as *churn drilling.*

ROTARY DRILLING

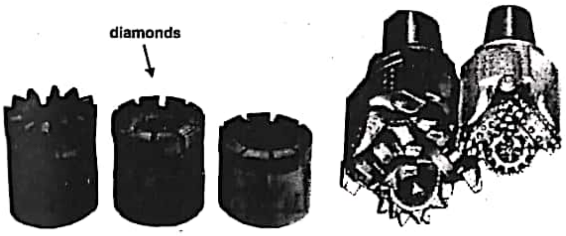
For rotary drilling, hollow drill rods of steel or aluminium are used. These are thread-connected and transmit torque and feed pressure to the drilling bit or drilling tool which is attached at the end of a column of the drill rods. Rotation of the drill rods is through gearing driven by a prime mover at the surface. As the rods rotate, the drilling tool abrades the rock and the cuttings are cleared by pumping water under pressure or compressed air down the hole through the hollow drill rods. The water or air, along with the cuttings, comes to the surface through the space between the drill rods and the sides of the drill hole. In some drillings, specially those for oil exploration, mud which is not very viscous, is circulated instead of water. The mud which keeps back any water, gas or oil pressure encountered during drilling is known by various trade names such as *bentonite, aquagel,* etc. and these muds serve different in the mining areas it may be necessary to resort to mud flushing when passing through a fractured or friable zone.

Aluminium rods weight only half as much as steel rods, but owing to their bigger gauge they possess 90% of the mechanical

strength of the latter. The couplings, which are the parts most exposed to wear, are made of chromium-nickel steel. Aluminium rods offer numerous advantages, such as increased machine capacity, easier handling, more rapid and simple recovery of the drill string and faster rotation, all of which contribution to simplifying drilling and reducing costs.

The various methods of rotary drillng are known by the type of drilling tool used but the diamond drilling method is quite common.

Diamond Drilling



(a) Diamond drill bit (b) Tricone rock roller bit

Fig. 3.5

This method is commonly adopted where cores of rocks passed through are desired for accurate records of the strata or for testing the rocks for their strength, composition, porosity, etc. Fig. 3.5(a) shows the common type of drill bit which consists of a cylindrical cast steel shell having in its lower face a number of small sockets in which pieces of black diamonds are set. These diamonds are not useful as jwellery but are used in the drill bits for their hardness and the bit is suitable for the hardest rocks. The hole sizes in diamond drilling are designated as NX, BX, AX, and EX. The drill rods and the drill bits are specified under two main groups, X series and W series, as per the standards laid down by DCDMA (Diamond Core Drill Manufacturer's Association), an international Association. The drill hole diameters and core sizes (in mm) available are given below.

Standard	Drill rod outside dia.		Hole dia. mm	Core dia. mm.
	X series in inches	W series in mm.		
NX....	2.3/8	NW – 67	75	54
BX	1.29/32	BW – 54	60	40
AX	1.5/8	AW – 44	47	28
EX....	1.5/16	EW – 35	38	21

NW series rods are of W and conform to international standards for conventional drilling. NQ series drill rods are manufactured by Longyear for wire line drilling technique. There are Q series standards for wireline drilling rods but some manufacturers have their own sizes.

The core sizes where wire line drilling technique is adopted are : NX holes - 44 mm; BX holes - 25 mm.

The surface arrangements for diamond drilling include, as shown in Fig. 3.6.

1. a derrick,
2. an engine for supplying power,
3. an winch,
4. a pump for supplying water under pressure for flushing,
5. a setting tank,
6. a platform for keeping the drilling rods lifted for removal of core or changing the bit,
7. core boxes for keeping the cores, and
8. a driving and feed mechanism for the drill rods.

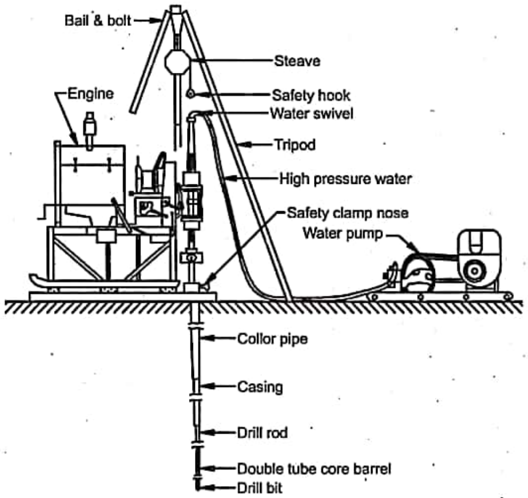


Fig. 3.6 Surface Arrangement for Rotary Drilling

The diamond drill bit is rotated at a speed of nearly 300 r.p.m. and the pressure on the diamonds is between 1.5 and 2 kgf/cm^2. The pressure acting upon the diamonds of the drill bit and the rate of advance of the drill bit into the rock are controlled by an arrangement known as "feed mechanism". The feed mechanism is hydraulic for deep holes, but may be replaced by screw feed for shallow holes. Beyond a depth of nearly 60 m, the weight of the rods keeps the bit pressed against the rocks and the feed mechanism may not be necesary. At greater depth the feed mechanism is operated in such a way that the weight on the drill bit is not excessive.

Screw Feed

On smaller machines, driven either by hand-power or mechanically, the feed is by a screw feed arrangement comprising a series of differential gears.

In this arrangement the drill rods pass through a hollow screw shaft, threaded on the outside, and porivde with a long keyway. A bevel pinion, rotated by the bevel gear of the main driving shaft, has feathers engaging the keyway on the screw shaft, to which it imparts rotation. It also drives gear wheel A, engaging with gear wheel B on a rotation. It also drives gear wheel A, engaging with gear wheel B on a countershaft. Usually three different combinations of gear are provided here, any of which can be utilised to vary the rate of advance to suit the type of rock. Grear B, through the counter-shaft, drives C, which engages a fourth wheel D, threaded internally to fit the threads of the screw shaft. If, for example, the number of teeth on gears A, B, C and D are as shown in the diagram, viz. 38, 36, 24 and 25, one revolution of A = 38/36 revolution of B and C, = 3/836 × 24/25, revolutions of D. Therefore 75 revolutions of A will cause D to rotate 75 × 38/36 × 24/25 = 76 times. Consequently for every 75 revolutions of A,D revolves 76 times, and the screw shaft moves forward by a distance equal to the pitch of its thread. If this is 6 mm, then the rods advance 24 mm for every 300 revolutions. The movement of the screw shaft is imparted to the rods by the chuck.

A pressure gauge attached to the roller friction collar records the varing pressure on the bit. The setting of the feed decided upon is obtained by sliding a lever to the desired position for locking the appropriate loose gear, giving the speed required for the ground being bored.

The feed ratio may be changed while the drill is operating; but if any necessary change is neglected, so causing the drill rods to advance too rapidly (*e.g.* when a sofer stratum is suddenly encountered) the rate of advance is automatically checked by the slipping of a spring-loaded friction cone-clutch on the bottom of the countershaft.

Diamond Drill

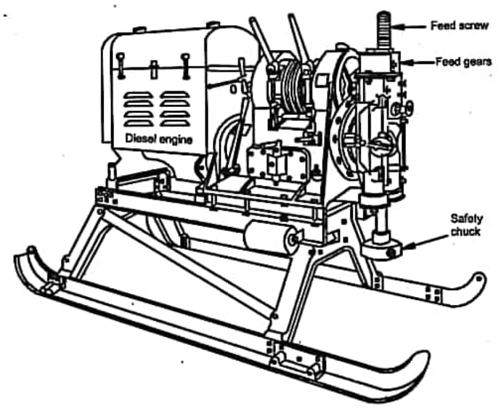


Fig. 3.7 (a)

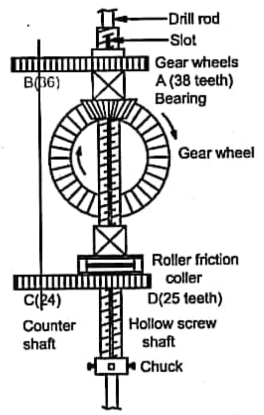


Fig. 3.7 (b) A screw feed head on a diamond core drill Engine

Hydraulic Feed Mechanism

Fig. 3.8 shows the general principle of a typical hydraulic feed mechanism (excluding the engine and frame) for a deep boring, the main features, starting from the bottom, being the chuck, the bevel driving gears, the hydraulic cylinder and the ball-bearing suspension box. At the top is seen a lifting bail and water-swivel whose functions are self-evident.

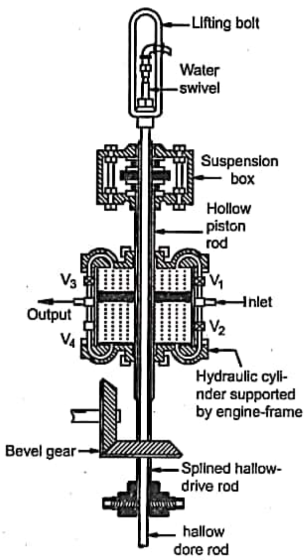


Fig. 3.8 Hydraulic feed mechanism

There are three hollow rods, one within the other, namely, the inner bore rod, the middle drive rod and the outer piston rod. The hollow bore-rod is clamped to the drive-rod by the chuck. The drive rod is secured to and supported by the collar-plate in the suspension box, the latter remaining supported by being fixed to the hollow piston rod which carries the piston in the hydraulic cylinder. This cylinder is firmly fixed to the engine frame work.

The rate of feed of the bore-rods is governed by the rate at which the piston descends, for this governs the descent of the suspension box and therefore, of the collar plate and drive-rod to which the uppermost bore-rod is clamped. It will be seen that the drive rod is free to rotate within the suspensions box and within the hollows piston rod.

The drive-rod had on its outer surface ribs or splines which pass through slots or grooves in the box of the horizontal bevel drive-whell. It can thus be rotated by the gear wheels (whose position is fixed) but it is free to descend through the horizontal wheel, carrying with it the bore-rods and boring tools at their lower end.

The piston in the hydraulic cylinder may be moved either up or down by admitting water under pressure to the appropriate side of the piston through the inlet pipe and one of the controlling valves, V_1 or V_2, and by simultaneously releasing an equal amount of water from the other side of the piston through one of the valves V_3 or V_4 and the outlet. A single lever operates the four valves simultaneously to produce any desired pressure, either downward or upward, on the piston. Thus the weight of the rods may be partly taken off the boring tools by upward pressure; or the whole weight of the rods can rest on the boring tools and additional downward pressure can be applied. In this way complete control may be obtained over the pressure on the boring tool and over the rate of forward feed of the rods.

The cuttings are cleared from the drill hole by cirulating water under pressure which is forced down the drill rods by a pump through a flexible hose pipe and waterswivel connection. The return water from the hole goes to a settling tank and it is used over and over again.

Core Recovery

To collect the core of the rock drilled, a device known as the core barrel is used. It is length varies from 0.5 to 3 m. There are two types of this

1. The single tube core barrel, and
2. Double tube core barrel.

A single tube core barrel is suitable for homogeneous formations where the core is not eroded by flushing water and a solid core can be taken without risk of blockage in the barrel.

The connection of the diamond crown, the single-tube core barrel and the mud bucket (also called calyx) are shown in fig. 3.10. The core lifter is placed within the bevel shell which has its inside conically shaped to receive the former. The core lifter is corrugated on the inner face and is a split ring. It occupies the wider portion of the bevel shell when drilling takes place so that it has little or no tendency to grip the core. Atter certain progress in drilling when the rods are lifted to take out the core, the split ring descends inside the bevel shell and grips the core. The latter may now be broken off by a twist and raised to the surface. The core is replaced after about 250 m of drilling.

The larger particles of drill cuttings which the circulating water fails to carry upto the surface settle down in the mud bucket.

Where supply of flushing water is plentiful, calyx is not necessary. The water under circulation is nearly 900 litres per minute.

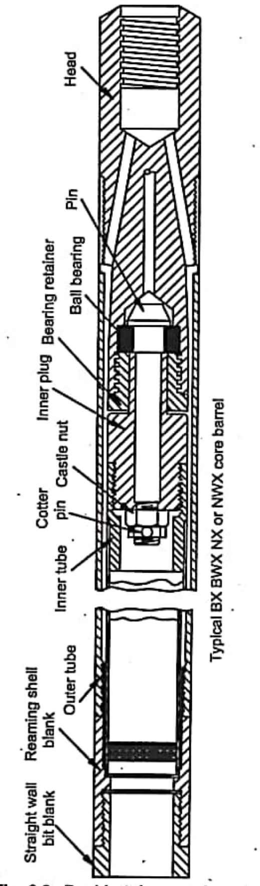


Fig. 3.9 Double tube core barrel

In soft and friable rocks, the core is partially washed away due to the circulating water flowing past it. The rotation of the barrel greatly assists in grinding the core so that its recovery in a single tube core barrel is poor. To avoid these difficulties a double tube core barrel is used, specially where good core of soft rock is desired. In a double tube core barrel in inner tube with holds the core does not rotate during drilling as it is suspended on ball bearings mounted in the block at the top of the barrel permitting the inner core barrel to remain stationary. Moreover, water does not flow past the core but in the annual space between the inner tube and the outer barrel and through channels near the bottom of the hole. A double tube core barrel improves drill bit economy and overall drilling performance. Core revoery is good in hard uniform rocks, but poor in loose, soft, friable or weathered rocks. Vibrations of drill rods result in poor recovery. In hard rocks, to achieve good core recovery the drill should be run at low speed and heavy pressure; in soft rocks, reverse procedure should be adopted. The combination of pressure on drill bit and its rotational speed should be such as to give vibration-free drill string during drilling.

Wire Line Drilling

Normally core barrels of 0.5 m to 3 m length are employed. For removal of core during the conventional method of drilling, all the drill rods have to be withdrawn to the surface, after firlling a length equal to length of core barrel. The withdrawal of the rods and their re-introduction into the borehole with the additional drill rod, after removal of core, takes a considerable time, nearly 75 to 90% of the total time spent on drilling. A wire line drilling technique is an improvement to reduce this time. The rods are not taken out to remove the core which is collected in the core barrel tube during drilling. The tube is pulled out the surface through the drill rods with the help of a catcher which is lowered through the rods by a 5 mm dia. wire rope. The catcher grips the tube containing the core. Core size less than BX is not possible with wire line coring equipment which is therfore used for drilling holes of NX and BX size only. The speed of drilling with this equipment is nearly 18 m per shift (8 hours) in the types of rocks met with in the coalfields. All the drill rods need to be withdrawn to the surface only when the bit has to be changed.

Wire line drilling is possible upto a depth of 1000 m. As stated earlier, the rods used for wire line drilling have specifications as laid down in "Q" series decided by DCDMA. An ordinary drilling equipment can be adopted to wire-line drilling and hoisting equipment with suitable modifications.

Some of the recent drills in the market are equipped with hydrostatic drive. In such drive an electric motor or a diesel engine drive a water pump. Its pressure is used for rotation of drill rods through a hydralic motor and also for hydraulic feed. Its main advantage is that speed can be varied from zero to a certain limit without any fixed ratios that are possible in a geared drive.

Water Loss During Drilling

When drilling in fractured zone or strata with cavities the circulating water is lost in such zone and fails to appear at the surface. This is known as *water loss.*To seal up the cavities or fractures around the hole, saw dust, husk, etc. are mixed in circulating water. If this is ineffective, it may be necessary to ream the bore hole and fix a casing pipe to cover the fractured zone. The casing pipe can be removed after completion of drilling. At depth, instead of resorting to a casing pipe a special type of mud like bentonite or kaolinite may be used to overcome the water loss. Drilling mud is a suspension mixture of certain types of colloidal clay in water and/or oil and used as drilling mud. The mud most commonly used in diamond drilling is a slurry of clay and water. properly controlled drilling mud slurries can prevent caving or collapse of borehole sides by building thin, impermeable protective coatings of clay particles to the walls of the hole. The mud is generally used as a final resort and the water circulation pump has to be replaced by a suitable mud circulation pump for this purpose.

Diamond drilling method is suitable for drilling at any angle to obtain cores of friable strata as well as the hardest rock. It has been adopted for drilling upto a depth of 3000 m and hole diameter upto 200 mm. Drilling upto such large depths is not required in coal mining areas where the maximum drilling depth is upto 1000 m, as coal mines are rarely deeper than that.

Other methods of rotary drilling differ from the diamond drilling method essentially in the type of drilliing bit used. The drill bits used are as follows:

The saw-toothed crown. The drilling tool is a saw-toothed steel crown or cutter. The teeth are set alternately inward and outward to give the necessary clearance. The speed of rotation is only 5 to 10 r.p.m. and the drill bits are suitable only for drilling through rocks of medium hardness; only holes of diameter not less than 150 mm are possible.

Rock roller bits. These are suitable for hole diameter between 75 mm and 300 mm. In mining areas these are commonly used for drilling large diameter holes in mechanised quarries. Flushing of the hole with compressed air instead of by water under pressure is the common practice with this type of bit. Rock roller bits can be used for deep hole drilling with speed and are suitable for mostly vertical downward holes. (Fig. 3.5, b).

Chilled steel shots : These shots are prepared by heating very finely divided steel particles to a very high temperature and then suddenly cooling them in ice cold water. Chilled shots are used in conjuction with a plain steel shell or cylinder with a diagonal slot near the bottom. They are fed through the hollow drill rods and pass to the bottom of the hole where they get caught between the bottom end of the cutting shell and the rock. As the shell and the drill rods rotate, the chilled steel shots cut the rock by a milling action. The method is suitable for vertical and large diameter holes of 100 mm to 750 mm.

It is also called calyx drilling, but is not much favoured these days as diamond drill bits have gained wide popularity and are available in large diameters upto 250 mm.

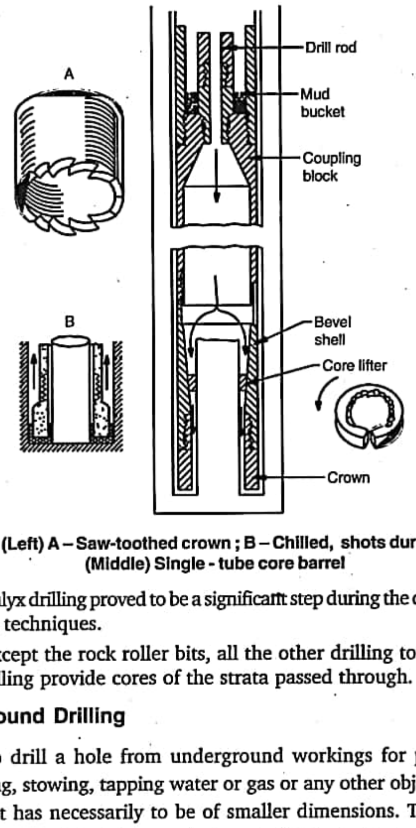


Fig. 3.10 (Left) A – Saw-toothed crown ; B – Chilled, shots during drilling (Middle) Single - tube core barrel

Calyx drilling proved to be a significant step during the development of drilling techniques.

Except the rock roller bits, all the other drilling tools used for rotary drilling provide cores of the strata passed through.

Underground Drilling

To drill a hole from underground workings for purposes of prospecting, stowing, tapping water or gas or any other object, the drill equipment has necessarily to be of smaller dimensions. This restricts the size and weight of the machine and therefore puts limitations on the size of the bore hole and its length (or depth). The power available underground may be electricity at 440/550 volts, or compressed air (in most of the metalliferous mines). Flame proof diesel engines are rarely employed as power sources. At some underground working places the water supply for drilling may be on a very limited scale and not as plentiful as on the surface.

Underground drills for exploration have often to be shifted to blind ends of roadways with narrow dimensions. They are, therefore, usually with skid plates. Components of aluminium alloy are nowadays increasingly used for such drills to reduce weight.

Water Development Society, Hyderabad, manufactures drills of the following sizes. The drills are avaiable with hydraulic, semi-hydraulic and pneumatic operating systems with an option of diesel or electric prime-mover. Underground drills can be provided with DTH & Drifter attachment.

RANGE OF DRILLS

Model	Hole dia.	Depth, m	Application	mounting
WRC - 650	150 - 170 mm	30 - 60	Open cast mines	Crawler
WDC - 150	150 - 165 mm	30 - 60	Open cast mines	Crawler
WDC - 100	100 - 115 mm	30 - 60	Open cast mines	Crawler
SUPERTRAC 15	100 - 115 mm	30 - 60	Open cast mines	Crawler
WDM - 400	150 - 165 mm	30 - 60	Open cast / underground	Skid
WDM - 100	100 - 115 mm	30 - 60	Open cast / underground	Tyre
WDM - 400 M DTH & Driffer	[100 – 115 mm] [50 – 100 mm]	30 - 60 30 – 45	Underground	Skid
WDM – 100 U	100 – 115 mm	30 – 60	Underground	Trolley
WDM - 100 UF	50 – 105 mm	30 – 45	Underground	Skid
WDM – 100 U	100 – 115 mm	30 – 60	Underground	Skid
WDM – 100 FM	100 – 115 mm	30 – 60	Underground	Skid

BORE HOLE DEVIATION, BOREHOLE SURVEY & DEFLECTION

The departure of a bore hole, whether vertical or inclined, from its set course on its own is known as deviation of the bore hole. Such deviation of the bore hole. Such deviation is often very significant, usually 2° per 30 m length. The deviation is both in azimuth and off the vertical i.e., the bore hole tends to wander away from the proposed plane and also changes its angle with the horizontal.

Measurement of the deviation of the borehole is called *borehole survey.*

In the driller's terminology *defelction* of a borehole is the deliberate and international deviation of a borehole brought about by the driller with a view to correct the borehole's coarse or to force it into a curve which is sharper than, or in another direction to, that caused by deviation.

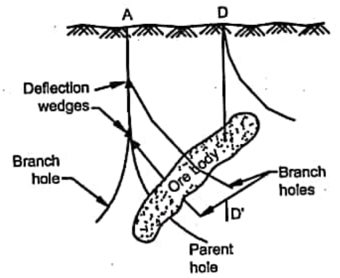


Fig. 3.11

Fig. 3.11 Deviation of borehole and directional drilling. Borehole at D, because of deviation, reports "No orebody". One parent hole at A covers a wide area for prospecting with the help of deflected branch holes and eliminates the need for a number of prospecting boreholes from the surface.

Major factors affecting deviation, in general, are depth and angle of the borehole and the nature of the strata drilled. Certain drilling techniques also influence the deviation and with judicious application of the same, deviation can be controlled and kept at a minimum.

Experienced drillers have observed from long experiece that borehole deviation follows certain rules which apply generally but to which there are some exceptions. These rules may briefly be stated as follows :

1. Deviation may occur in hole of as small a depth as 100 m.
2. Deviation increases with depth.
3. Borehole tends to follow the bending plane between a hard rock and a soft rock. contact zones of dykes, veins, lenses, especially hard and soft act as natural deflecting planes. In soft ground and shear zones deviation is common.
4. Some drilling techniques have a major primary influence on deviation, such as hole diameter, bit pressure, core barrel and casing.

Diameter of hole : Large diameter holes tend to deviate less than the small diameter holes. It has been observed that in 'BX' size the deviation of borehole is 2° to 3° per 30 m and with other conditions being similar it may become double of this in 'AX' size and triple in 'EX' size.

Bit pressure : High bit pressure accelerates deviation; thus, a sharp bit drilling a t a moderate penetration will require less bit pressure (and consequently deviates less) than a blunt bit or a bit that is fed faster.

Core barrel : A diamond drill hole tends to stay straight. One of the factors against deviation is the constraining of the barrel by the hole. This is affected by the stiffness of the barrel and its clearance in the hole and such long, heavy walled core barrels close to hole side will reduce deviation. Worn core barrels aggravate tendency to deviate; this is also the case if light bore rods are used.

Deveiation can be suitably controlled and reduced by

(a) paying attention to causes stated at (4) above.

(b) using full diameter drill rods for the entire length of the bore-hole and using proper couplings etc., with good threads to avoid angularity and lack of concentricity.

(c) correcting the hole's course by deflection methods, described shortly, where necessary.

Bore Hole Survey

Measurement of a drill hole deviation is called surveying a bore hole and all methods of bore hole survey are aimed to locate any point along the hole-course with respect to three dimensional co-ordinates attaining varying degrees of accuracy. Some methods are used to measure the deviation in angle only; others measure both angle and bearing deviation but at a predecided point only and the most sophisticated instruments can measure both angle and bearing deviation in a continuous manner all along the bore hole course.

Some of the common methods are given below :

Etch method using Hydrofluoric acid

This method can be adopted to measure the deviation in angle only (off the vertical). HF has the property of corroding glass. The HF solution partly filling a glass test tube, corrodes the inner wall of the test tube upto the column of the liquid and thins its wall upto the plane of separation of air. The upper part of the tube not in contact of liquid will remain unchanged. This plane of separation can thus be perceived, later on, on the same empty test tube, by a line, knonw as 'etch line'. When the test tube stands vertical, the etch line will be normal to the longer axis of the tube. In any other tilted position the etch line will make corresponding angle. This angle, known as 'etch angle' can be measured and 'tilt angle' can be found. This principle is applied in HF method of bore hole survey. To measure the inclination of diamond drill holes a small quantity of dilute HF solution is poured in a glass tube, lowered into the drill hole in a proper container and leaving it for sufficient time for the acid to etch a line on the inside of the tube and then measuring the angle of etch. HF with concentration generally 5%, 8% and 10% are preferred.

Hydrofluoric acid is properly mixed by shaking the container well. By adding distilled water to HF proportionately (by volume) the desired concentration can be obtained. About 50 ml of prepared acid is poured into the test tube and thoroughly shaken. After inserting the stopper in the tube, it is put into a container (suitable for 'AX' 'BX' or 'NX' size depending on the size of the hole) made up of either brass or mild steel. The container should have water tight joints even under heavy pressure. The container coupled with the drill rods, is lowered into the bore hole at desired depths. Inclination readings are normally taken at every 50 to 60 m and four containers are lowered at a time at desired intervals. For shorter bore holes 10% acid with about 40 to 45 minutes etch time is sufficient.

When the drill rods and container are withdrawn from the drillhole the etch tube should be removed as quickly as possible, emptied immediately and washed to stop further etching.

The line of etch is marked on the tube with fine dots using Indian ink, care being taken to mark the high and lower points carefully. The angle of etch or the apparent dip angle is measured on a protractor or on a graph paper. The angle of this etched line with the long axis of the tube gives the inclination of the drill hole at the depth tested.

The acid tube etch method is commonly adopted and is considered reasonably reliable. It can be done even for a cased borehole and can be carried out by the drilling crew. It is the oldest method of bore hole surveying.

Some instruments used for measuring deviation of boreholes off the vertical as well as off its intended bearing are the Carlson compass and Tropari instrument. A **Borehole Camera** developed by M/s Eastman Kodak Co. of Germany gives continuous record of borehole deviation from top to bottom.

For use in mining areas where deep holes are not required such photographic instruments suffer from the following disadvantages :

1. They are expensive.
2. They require facilities to process the film.
3. They are equipped with very small magnetic compass needles and are subject to disturbance by strong fields around rods.

In Kolihan, Madhan Kudhan, Akwali and other sections of the Khetri copper belt, Rajasthan, drill holes have been surveyed by HF (etch) method and the data obtained indicate that if this survey is conducted accurately it can furnish reliable data regarding the inclination of bore holes. While variations may be noted in some of the individual readings, the overall result is comparable with the inclination as measured by using a more sophisticated instrument like 'Tropari' except that this method gives the reduced level of any point along the hole course but its position in two dimensional co-ordinates is uncertain.

Directional Drilling

The term "Directional Drilling" means controlling the course of a borehole so as to follow a pre-determined path and complete the bore hole at the desired sub-surface location. The main objectives of directional drilling are :

1. To control deviation in a drillhole requiring during its course.
2. To return crooked holes to their normal course.
3. To sidetrack obstructions like lost tools, broken bits, etc.
4. To alter the course of the drill hole to enable a zone intersection at stipulated angle.
5. To provide multiple intersections by drilling branch holes from previously drilled parent hole, and
6. For many such drilling problems.

However, the range of directional drilling is limited to a circle, having the drilling machine as its centre, with a diameter equal to the depth of the borehole.

Deflection of boreholes

The methods used for deflection of boreholes are :

1. Casing deflection
2. Use of wedges, *e.g.*, the Hall-Rowe deflecting wedge.
3. Sonic orienting assembly
4. Arc cutting.

Deflection techniques give good results when the deflection is in the same direction as the natural deviation. As a matter of fact, deflection can be very inconvenient and ineffective if it works against the natural deviation. Deflection of boreholes resulting in a few branch holes off the parent hole eliminates the need for a number of holes from the surface during prospecting (See Fig. 3.11).

Deflection techniques are usually adopted in deep oil well holes. They are very rarely used in the mining industry in India so far. In United Kingdom the deflecting technique has been used in a project for underground gasification of coal at a site near Newark, Nottinghamshire. Mining Magazine, May 1986, narrated the following information about the project :

Underground gasification requires at least two boreholes drilled from the surface and connected together in the coal seam. Air or some other reactant is pumped down one hole to creat combustion and gas is removed from the other.

The coal seam for gasification in question is 2 m thick, 600 m below surface. N.C.B.'s Engineers in United Kingdon will drill four boreholes from the surface. Of these the first one, D, though commencing as a vertical borehole, would be deflected and curved through 90° to enter the coal seam horizontally and remain in the seam for a distance of more than 300 m (Fig. 3.12). Special instruments would have to be used in order to keep the drill in the coal seam. After completion of such deflected borehole. N.C.B.'s Engineers will drill three vertical surface boreholes to join with the in-seam hole. Steam and oxygen would be forced down one vertical hole to stimulate the production of gas to be extracted through one or both of the vertical holes. The gas would be cleaned before being burned on site unless some local use could be found for it. It is envisaged that the full trial could take upto six years.

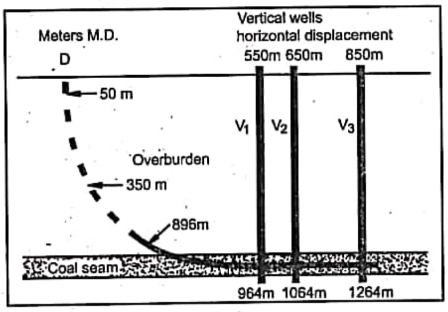


Fig. 3.12 A deflected borehole and 3 vertical boreholes for underground gassification of coal

✧ QUESTIONS ✧

1. What are the various methods of drilling ? Give their limitations.
2. A hole has to be drilled from the surface to prove a coal seam at a depth of nearly 300 m. Describe the drilling process. Give a list of equipment used.
3. Write short notes on : water loss, rock roller bits, core recovery, wire line drilling.
4. What are the methods for bore hole survey. Describe one method.
5. What is the difference between a deviated bore hole and a deflected bore hole. What are the methods employed for deflecting a bore hole ?

✧ ✧ ✧

Chapter -4

Shaft Sinking

After a mineral bed has been proved, if its extraction is considered economic, a decision is taken by the planning engineers whether the mineral is to be extracted by opencast mining or by underground methods of working. To extract mineral by underground methods of working, the access may be by an incline (a tunnel from surface to the mineral bed), by an adit, or by a well which is called *shaft* or *pit* in mining terminology.

Shape and Size of a Shaft

Shafts are circular in shape and rectangular shafts are rare in this country, the exceptional cases being some of the shafts in metal mines. A circular shaft is best able to resist heavy side pressure for a given cross-section, offers the least rubbing surface to ventilating air current. It is easy to sink and line with bricks or concrete. The finished diameter of a shaft varies from 4.2 m to 6.7 m.

A rectangular shaft sunk in recent years is the main shaft at Mochia Magra Mines, Zawar, Rajasthan (Hindustan Zinc Limited). It is of 5.2 m × 3.8 m in cross-section, vertical, 321 m deep from the shaft collar with 30 cm thick concrete lining.

Shaft sinking is costly operation. The mining companies pay to the shaft sinking contractors amounts varying from Rs. 50,000/- to Rs. 75,000/- per metre of overall depth of the shaft sunk and this amount includes sinking, lining with concrete, head gear, winding engine, compressors and all the machinery required for sinking and lining upto the final depth (A turnkey job). The high cost involved demands much care in selecting the shaft site. It is, therefore, a standard practice to bore a pilot hole at the proposed shaft site to have a core of the rocks. Such hole need not be at the shaft centre, but may be within 50 m radius of the shaft centre and often only one hole would serve for twin shafts. The hole gives an idea of the rocks to encounter during sinking and provides data essential for :

1. confirmation of shaft site,
2. selection of water control methods,
3. estimation of sinking time and costs,
4. design of shaft and permanent lining.

The present practice prefers holes with cores of 100 mm diameter. Such large diameter cores and holes are preferred for reasons of deviation control, good core recovery, satisfactory laboratory strengh & permeability tests.

Surface Plant and Equipment

The surface plant and equipment required for sinking is as follows :

1. Steam boilers or diesel engines for winding engine, pumps, etc. unless electric power is available.

2. Winding engines and winders fitted with locked coil ropes.

3. Steel headgear. The headgear may be temporary nature and after the sinking is over, it is replaced by a permanent headgear and permanent winders to suit the output.

4. Double drum winches for waXing scaffold, and other winches for lighting cables, shot-firing cables, pump suspension ropes and pump cables.

5. Air compressors for jack hammer drills used for drilling into rock and other compressed air operated equipment.

6. Fan of nearly 300 m³ per minute capacity.

7. Generator with diesel or steam engine for lighting.

8. Folding doors to cover the shaft top.

9. Shaft centering arrangement.

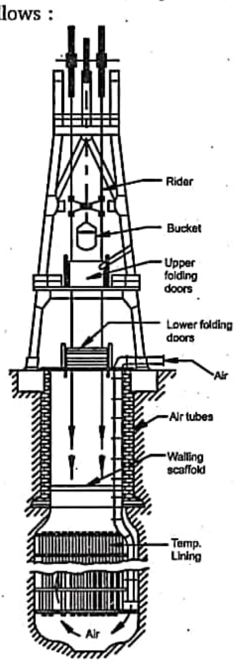


Fig. 4.1 General Arrangement of Sinking Shaft

10. Signalling arrangements from pit bottom to pit top and from pit top to winding engine.

11. For disposal of debris, chutes, buckets, and tipping tubs with tramline, etc.

12. Workshop including smithly shop, mortar mill and other usual machines.

13. Lamp room, first aid room, magazine, stores, office, etc.

Special difficulties encountered during sinking would require use of additional equipment.

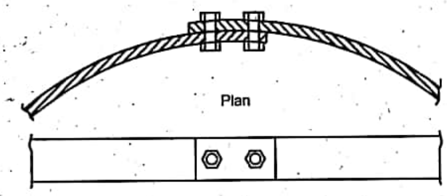


Fig. 4.2 Wrought iron curb

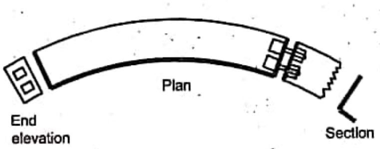


Fig. 4.3 Segment of walling curb

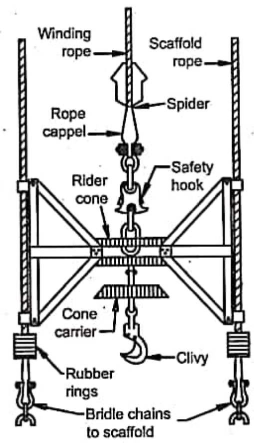


Fig. 4.4 Rider as used during sinking

The centre of shaft is marked by concrete pillars, each having a plate with centre line scribbed on it. These pillars are required always as reference marks when sinking. They should, therefore, be so placed as not to be damaged by sinking operations or covered by debris.

The strata through which a shaft has to be sunk may be divided into three groups.

i. Sub-soil or alluvium.

ii. Hard rocks below the alluvium and above the mineral bed (generally consisting of sand stone, shale, thin coal seams, etc. in coal mining areas).

iii. The coal seam, or the mineral bed.

The perimeter of excavation at the surface is marked by pegs. The radius of such excavation is equal to finished radius of shaft + thickness of brick or concrete lining + a clearance of 230 to 300 mm.

The starting point of a vertical shaft at the ground surface is called collar or shaft collar. The sub-soil or alluvium and weathered rock are excavated upto the strong rock by earth cutting picks, chisels and hammers, without recourse to blasting. In most cases the thickness of such sub-soil varies from 3 m to 20 m. The excavated material is lifted to the surface through buckets hoised by manila rope in the same manner as practised for sinking ordinary water wells. A crane with a long jib or a grab is sometimes used for clearing the debris; a crane can be used upto a depth of 30 m.

During sinking the shaft sides are kept vertical and truly circular by a radius rod which is used to measure the radius from a plumb wire suspended from the surface in the centre of the shaft.

Temporary Lining

It is necessary to support the sides of the excavation to prevent their collapse. A heavy wooden frame or a frame of steel girders is built across the shaft top from which the first (topmost) ring of temporary lining is suspended. Alternatively the temporary lining may be suspended from strong iron spikes embedded on the surface round the periphery of the shaft. The temporary lining consists of skeleton rings (also called curb), hangers, planks of sal wood and tightening wedges. The skeleton rings are of mild steel, made in segments of 3 m in length and shaped to the circumference of the shaft. The segments are 100 mm × 25 mm in section and are joined together by fish plates or by lap joints as shown in Fig. 4.2 Before taking these rings in the excavation for support the segments are assembled at the surface and each segment numbered for assembling in its proper place in the excavation. The first skeleton ring to be inserted is suspended by chains from the steel girder frame or heavy wooden frame at the surface. The wooden planks are of sal, 2 m long, 215 mm wide and 38 mm thick and are securely held against the sides of the excavation by wedges driven between the rings and the planks (Fig. 4.5). Each ring is suspended from the ring above by hangers or S-shaped iron hooks of 25 mm diameter placed at intervals of about 1.2 m around the shaft circuference. The rings are hung at intervals of 1.2 to 1.5 m and every fourth or fifth ring is supported on plugs driven into holes drilled horizontally into the shaft sides. Friction with the ground keeps the planks in position and cavities behind the planks are packed with wood.

Blasting should be avoided in the area where temporary lining is essential.

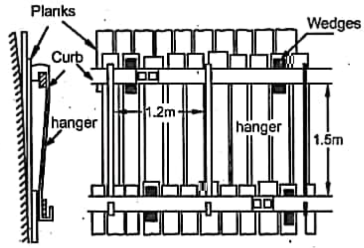


Fig. 4.5 Temporary Lining

Permanent Lining of shaft sides

When strong rock is reached, the excavation is reduced to the finished diameter of the shaft and continues thus for 3 to 4 m below. Arrangements are then made for construction of permanent lining which may be of brick, concrete or special steel tubbing. Brick walling is a common practice for ordinarily compact and moderately walling is a common practice for ordinarily compact and moderately wet strata. Ordinary bricks of first class and well burnt quality of a size 225 mm length × 115 mm breadth × 75 mm height are used. Usual thickness of brick lining varies from 0.4 m to 0.6 m.

The hard rock from where the permanent lining has to be commenced is made level only with picks and chisels (and not by explosives, to avoid shattering of the strata) with a projection inside the shaft side as shown in Fig. 4.7 at the ledge. Sinking is usually stopped when walling is in progress. A 150 mm layer of concrete is then laid to form a level bed, the inside edge of the concrete being the finished diameter of the shaft. A bricking curb (also called crib) made of cast iron is then placed on the hardened concrete floor. The curb is made in segments as shown in Fig. 4.3 shaped to conform to the finished diameter of the shaft. Before lowering the curb segments numbered. In the shaft the curb segments are assembled on the concrete, floor, correctly centred, levelled, and bolted together, each joint being wedged against the sides of the shaft to hold it in correct position. Brick walling is then started above the curb and the inner surface of the brick wall is kept vertical and true to the circumference of the shaft by plumb wires suspended from 4-5 reels at the surface. As the brick walling proceeds the temporary lining is dismantled in stages. The space between the brick lining and the excavation is filled with ash, sand or loose bricks. If water percolates from the strata which have been lined, the packing allows the water to percolate and this prevents build up of hydrostatic pressure behind the brick wall. Weep holes are left in the brick walls at the curb level during their construction for escape of such water which is collected in the water garlands at the curbs. The water is then piped down the shaft from the water garlands. The bricking curb comprising the water garland is of a special construction as shown in Fig. 4.6 and is called a "galand curb" which is required only where water percolates from the strata. It is made in segments and one or more of the segments are provided with an outlet hole into which is screwed a nipple for 50 mm diameter drain pipe.

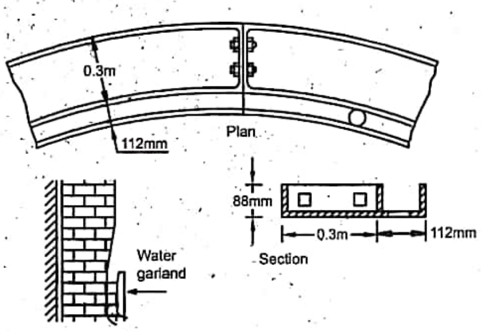


Fig. 4.6 Water garland curb

Permanent lining is generally not required where the shaft sides are of strong rock but the shafts sunk at Sudamdih and Monidih collieries and Jaduguda mine have permanent linings of concrete from the surface to the bottom of the shaft.

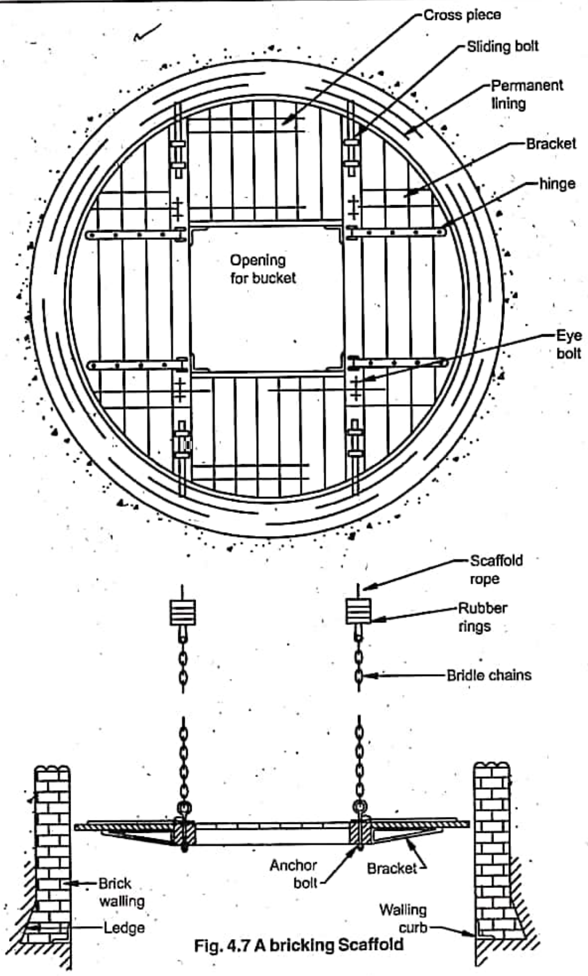


Fig. 4.7 A bricking Scaffold

Walling Scaffold

Construction of brick wall is carried out from a walling scaffold or platform (Fig. 4.7). this consists of a frame of sal wood having 0.3 m × 0.3 m square members covered with stout sal planks. It has an opening 2 m × 2m square for passage of the sinking bucket. The scaffold is suspended by chains from two ropes hanging in the shaft, one on each side of the winding rope, and it is raised or lowered by a double drum winch (or alternatively two winches) to which the scaffold ropes (locked coil) are taken. The diameter of the scaffold is slightly less than the finished diameter of the shaft. Four sliding bolts are used to keep the platform steady when in use, and the bolts are pushed on to the top of the brickwork or into vertical recesses cut in the brickwork. About 1.3 m of walling is completed from one position of the scaffold.

In the shaft sides buntons have to be fixed at intervals of 9 m to 16 m for support of cables, water delivery pipes, compressed air mains, etc. The position of buntons where they are to be fixed in the permanent lining is marked by plumb wires suspended from the surface and holes are left in the lining for fixing the buntons.

Walling scaffolds have been designed which allow sinking and walling operations at the same time. These were used in the shaft sinking at Sudamdih and Monidih collieries, described later.

Rider

It is not a common practice to use guide ropes in a sinking shaft, and to prevent undue swinging of the bucket during its travel, a rider is used in additoin to the use of a locked coil rope for winding (Fig. 4.4). The rider runs on the ropes supporting the walling scaffold and guides the bucket during its travel. The rider cone is so through it. When the bucket has to be lowered below the walling scaffold, the rider rests on the cappels of the scaffold ropes, and the rope passes down through the rider cone, guided by a loose guide sheave called the *spider*. The spider is so constructed as to collapse when passing through the detaching plate at the header in case of an overwind. The rider serves its purpose only between the bricking scaffold and the surface. It enables a bucket to be raised and lowered at a much greater speed and with greater safety than if no rider is used.

Drilling and Blasting in a Sinking Shaft :

The hard rock in a sinking pit is blasted with explosives after holes are drilled. The shot-holes are arranged as shown in the Fig. 4.8. As a thumb rule it may be stated that the number of holes in a ring is three times the diameter of the ring in metres. The holes are drilled by hand-held jack hammer drills operated by compressed air. A jack hammer operates at air pressure of nearly 6 kgf/cm². The supply of compressed air to the drill is by rubber hose pipe connected to the compressed air main which is nearly 100 mm diameter and supported at intervals in the shaft. Usually 2 to 3 drills, each consuming nearly 3 m³ of free air per minute work at a time in a shaft of 6 m diameter. The holes are 38 mm diameter and 1.2 to 1.5 m deep. A hole, after it is drilled, should be plugged with wooden plugs to prevent entry of sludge. Gelatinous high explosives like Ajax G are used in sand stones and shales and special gelatine may be used in very hard rock like that of a sill. A hole 1.2 to 1.5 m deep, may require 0.6 to 0.9 kg of explosive charge.

Low tension detonators are employed for blasting. The holes need no stemming as the water in each hole acts as a good stemming material, though sometimes the drill cuttings are utilised for more effective stemming.

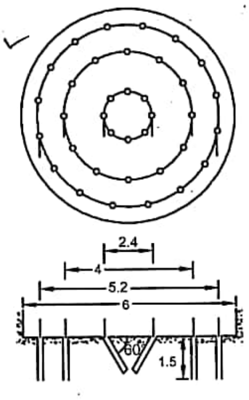


Fig. 4.8 Blast holes in a sinkin shaft (All the dimension are in metres)

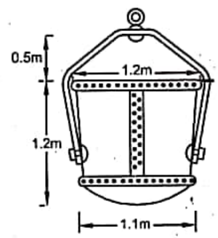


Fig. 4.9 Sinking bucket

Blasting of the inner ring is to be carried out first and all the shots in the ring are connected in series. Before blasting the equipment, lights, etc. which are likely to be damaged, are removed to the surface or raised high above the bottom of the shaft. The shots are connected to a shotfiring cable which is suspended from a reel at the surface. All men are withdrawn, the folding doors closed, and the shots are fired electrically from the surface by a hand operated heavy duty exploder. Blasting by tapping current from electric power lines is permitted by DGMS under certain conditions and was pracised in Sudamdih shaft sinking.

When the debris resulting from the blasting of inner ring is being cleared up, holes of outside sumper ring are charged and in this manner the blasting of all the rings is carried out. Blasting and clearing up debris of all the rings is carried out. Blasting and clearing up debris of all the rings gives a progress of nearly 1.2 m if the inside sumper holes are 1.5 m deep.

Clearing Up Debris

The debris is removed to the surface by a bucket (also called kibble, bowk or hoppit) (Fig. 4.9). A fork catch in the shape of a U keeps the bucket upright relative to the bow. The trunion aixs is low so that when the loaded bucket is suspended and the fork catch turned back the bucket tilts itself discharging the contents. The bucket is attached to the detaching hook of the winding rope through a clivy and bridle chains. Two buckets are generally in use, one at the pit bottom for getting loaded and the other which may be in transit, or at the pit top, getting unloaded.

The debris is cleared at the surface through a chute into which it is unloaded from the bucket. V doors ar sometimes used at shaft top so that the bucket, when hoisted slightly above the opened doors and then lowered to rest on the closed doors is automatically tilted if the fork catch is turned back. In another method a worker with a long hook stands on a platform built in the headgear over the chute. When the bucket comes to the surface, he catches the clivy with the hook and the winding rope is slightly slackened as the worker pulls the bucket with the hook towards the chute. In this case the chute is not above the shaft opening but to one side.

Shaft Centering Arrangement

The vertically and radius of the shaft are checked from time to time, usually once every day. At the time of such checking plumb wire is suspended from the surface in the shaft and the radius is measured by a light wooden radius rod. The usual arrangement is as shown in Fig. 4.10. At the decking level, riveted to the steel plates of the floor, is a channel with clamps in which a light section rail can slide on rollers. The clamps prevent overturning as well as lateral movement of the rail. At one end of the rail is a pulley over which piano wire from a reel can pass. On the rail and the channel there are two marks which coincide when the wire on the groove of the pulley attached to the rail is exactly in the centre of the shaft. When lowering the plumb wire, a light plumb bob is attached to it but it is replaced by a heavier one of nearly 19 kgf at the shaft bottom.

When checking of verticality and radius is completed, the plumb bob is raised to the surface and the rail withdrawn. The whole arrangement may be provide at the ground level where there may be no disturbance due to sinking operations.

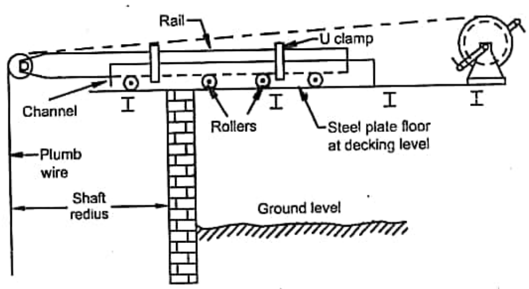


Fig. 4.10 Shaft centering arrangement

Dealing with Water

Where the make of water in the shaft exceeds about 100 litres per minute, pumps are used. For smaller make of water the bucket used for debris may be conveniet. It is filled by a small pump and then emptied at pit top. Centrifugal pumps are commonly used to dewater the shaft and pump may discharge water right at the surface. The common practice, however, is to instal a semi-permanent pump in an excavation in the shaft side (called *inset)* and the small pump at the shaft bottom delivers the pit water to it. This arrangement is adopted when some of the permanent lining of the shaft is completed as the delivery column of the main pump (installed in the shaft side) can then be supported on permanent buntons.

Sinking pumps which are suspended from the surface by power operated winches are also used. These are of the centrifugal type with a maximum of eight stages. In the initial phase of sinking only one or two stages are utilised and the rest replaced by dummy impellers. As the depth increases, further stages are added. For a deep shaft, the delivery column of a sinking pump is supported on buntons fixed permanently in the shaft sides and the bottom-most connection to the delivery range from the pump is made by a flexible hose pipe. The suction of the pump is flexible armoured hose fitted with a retaining valve and a strainer.

Ventilation

In a shaft exceeding nearly 25 m in depth, ventilation during sinking is produced by a mechanical ventilator which is commonly a forcing fan of 300 m^3 per minute capacity. The air tubes are of sheet iron, nearly 0.6 m diameter, suspended from the shaft side, as shown in Fig. 4.11. The bottom-most length of an air tubing is a canvas hose.

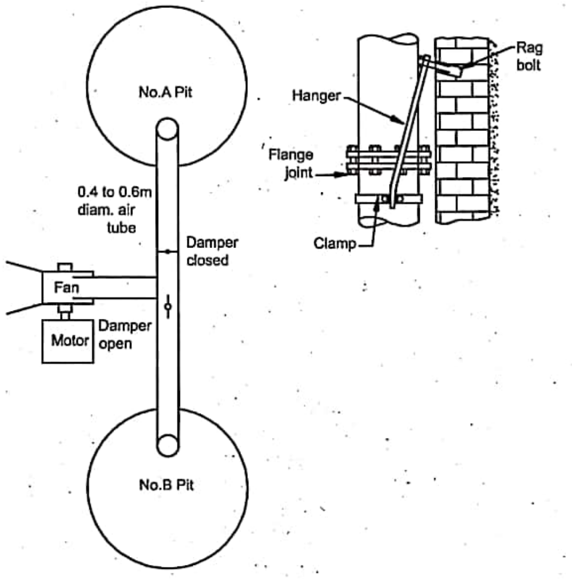


Fig. 4.11

Lighting

The workers use electric cap lamps, but the shaft bottom is illuminated by a cluster of 4 to 6 bulbs, each 100 watts and 110 V. The cluster is supplied power through an armoured cable suspended from a cable reel at the surface.

Permanent concrete lining

Concrete lining used for support of shaft sides is of two types :

1. Reinforced concrete lining, and
2. Monolithic concrete lining.

Reinforced concrete lining is costly and is used where high pressures have to be resisted. Concrete lining is stronger than brick work, offers less frictional resistance to air current, and can be erected rapidly. It is, however, difficult to repair and due to its rigidity may crack and collapse with slight earth movements. For monolithic concrete lining, concrete in the proportion of 1:2:4 (fresh cement : sand : coarse aggregate) is suitable for dry shafts. For wet shafts, richer mixture is preferable.

To construct monolithic concrete lining, the hard rock from where lining is to commerce, is dressed and levelled in the same manner as for brick lining to provide a base. When erecting the lining it is necessary to retain the wet or plastic concrete in position by a *shuttering*. This consists of segments of sheet steel nearly 1 m high, curved to suit the circumference of the shaft and having angle iron riveted to them for bolting together adjacent segments. The first ring of shuttering is carefully centered and levelled. Back side of the shuttering which will be in contact with concrete is geased for easy withdrawl after setting of concrete which is poured and ramded hard behind the shuttering.

Further shutterings are added and the concrete poured and rammed hard till the process builds up the lining upto the base of walling constructed higher up. Temporary lining is removed as the successive rings of shuttering are built up. The shuttering have to be left in position for several days until the concrete sets. Where the percolation of water from the strata is not insignificant, precautions have to be taken to prevent the water from washing the cement out of the concrete before it has time to set and this may be done by use of C.G.I. sheets as back sheeting. The C.G.I. sheets may be bolted or nailed on strips of wood about 40 mm thick which act as distance pieces to keep the sheets away from the shaft sides. The water percolating behind the back sheeting is taken through vertical pipes and discharged into the shaft. The space between the shaft sides and the back sheeting is filled with gravel. Once the concrete sets, holes are drilled into the gravel through pipes left in the concrete for the purpose. Liquid cement is then injected to seal off the water completely.

The concrete used for lining is sent down the shaft in a specially constructed bucket which has a bottom door for discharging the concrete on to a chute fixed over the walling scaffold. Through the chute the concrete gravitates into the place behind the shuttering.

Cast Iron Tubbing

A cast iron lining known as *tubbing* is used as a permanent watertight lining of shaft sides in case of water bearing strata containing water at high pressure. The water bearing strata may have several feeders of water at various depths and these have to be sealed independent lengths of tubbing, each made watertight at the top and bottom. It is also used as a permanent lining where running sand is encountered during sinking.

1. English tubing and
2. German tubbing (Tubbing plate shown in Fig. 4.12)

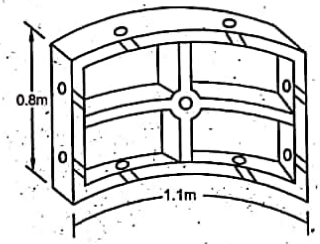


Fig. 4.12 German tubing plate

In both of these types of tubbing is built up of C.I. rings and each such ring consists of a number of flanged segements, shaped to suit the curvature of the shaft. A tubbing plate is strengthened with cast ribs and flanges. The German tubbing is preferred. The thickness of tubbing depends on the water pressure it has to withstand.

Construction of German tubbing upwards

A base for a wedging curb is prepared in good, strong levelled ground over which a cast iron wedging curb is carefully laid. It is carefully centered, levelled and wedged in position. The tubbing plates are then lowered, one at a time, by the winding rope, and slewed by the workmen into position on top of the curb. Each plate is bolted to the wedging curb. When a complete ring of plates is placed at site the position of each joint is measured from the centre line and the plates suitably adjusted to conform to the circumference of the shaft. The gap between the flanges is packed tighly with lead sheeting and the bolts of the flanges tightened. Quick setting concrete is poured and rammed behind the tubbing and another ring is built on top of the previous one. Additional curbs are laid, according to the nature of the strata, at intervals of nearly 20 m.

Construction of German tubbing downwards

German tubbing constructed from top downwards is known as "Underhung Tubbing". For building underhung tubbing an inverted weding curve or anchor ring is placed above the waterbearing strata and the tubbing suspended therefrom (Fig. 4.13). A ring of tubbing comprising the required number of tubbing plates to suit the circumference of the shaft is inserted as soon as sinking proceeds sufficiently below the wedging curb to accommodate the ring. As the sinking proceeds, the tubbing can be constantly maintained within 1.5 to 1.8 m of the shaft bottom. Each plate has a hole cast near the bottom (or at the centre) for pouring cement or concrete to fill up the space behind the next lower ring of tubbing. A horizontal retaining plate, made up in segments, is bolted to the lower flange to retain the cement or concrete in position until it sets. After the concrete sets behind the complete ring of tubbing a drill is inserted to clean out the hole which is then utilised for filling the space behind the next lower ring of tubbing. The process continues till the water bearing strata are passed through.

With underhung tubbing the advantages are :

1. Tubbing is done downwards. No temporary supports are, therefore, required.
2. No scaffold is required for its insertion.
3. Simultaneous sinking and strata support are possible.
4. Feeders of water can be promptly sealed by tubbing as they are met during sinking. This reduces pumping cost.

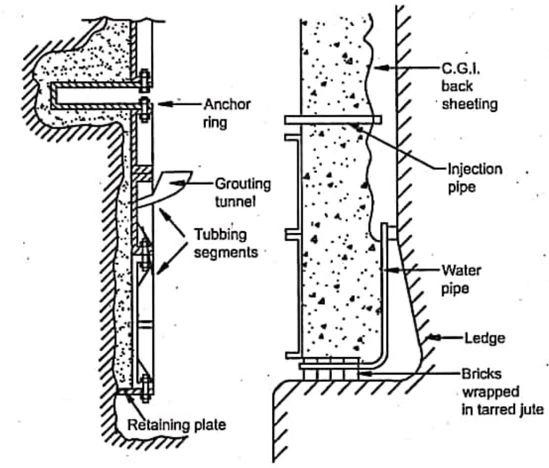


Fig. 4.13 Left - Underhung tubbing
Right – Concrete lining of a sinking shaft

The thickness of tubbing plates is increased after every 13 to 15 m. Tubbing plates of special construction are used for reception of buntons where they have to be fixed in the shaft sides.

Upward Drivage Of Shaft

It is sometimes necessary to drive a shaft upwards and such occasions are common in metal mining practice. Upward drivage may be carried out for a distance of upto 15 m and because of the difficulties involved, great care and experience are necessary to execute the work. A vertical shaft which joins two underground roadways but does not extend upto the surface is known as a *staple pit.*

At the proposed site of the pit a bypass is first driven (Fig. 4.13). The roof is then blasted down for a few metres and girders or wooden beams are placed across the gallery width to divide cross sectional area of the staple into three compartments. A_1, A_2 and A_3. The compartment A_1 which is larger than the others, serves the purpose of collecting rock dislodged after blasting and also as a platform on which the workers can stand. In the middle compartment A_1, thick wooden planks are attached vertically to the girders or wooden beams. Compartments A_1 and A_2 course the ventilating air current which is taken from one compartment, A_2 or A_3, across the top of the compartment A_1 and then to the other compartment.

In the compartment A_2 or A_3 a ladder of angle iron and mild steel rods is palced for the workers to ascend to or descend from their work site which is the top of compartment A_1 ; a winch is placed at the bottom of the staple for taking up materials to the top of the compartment A_1.

To maintain verticality of the shaft two iron plugs P_1 and P_2, each marked with punch, are driven into the floor. A light wooden rod, slightly shorter than the diameter of the staple, is similarly marked and the centre of the shaft, which corresponds to midpoint between P_1 and P_2 is also marked upon the wooden rod. When a pair of new wooden beams is placed in position the wodden bar is kept horizontally across them. Two plumbs lines are suspended over the plugs P_1, P_2 and the midpoint of the wooden rod gives the centre of the staple pit which is then marked on the roof.

Fig. 4.14 shows a staple in course of drivage. Holes 1 m deep are drilled by workers in the roof of the staple standing on the debris of compartment A_1. Before blasting, the top of the compartments A_2 and A_3 is protected by slanting timbers. Shots are fired electrically from a safe place in the roadway. The debris fills up the top of compartment A_1 and the surplus debris is thrown by workers down the compartments A_2, A_3 for further disposal. A pair of wooden beams and vertical planks are fixed in position on top of A_1 when the place is sufficiently advanced.

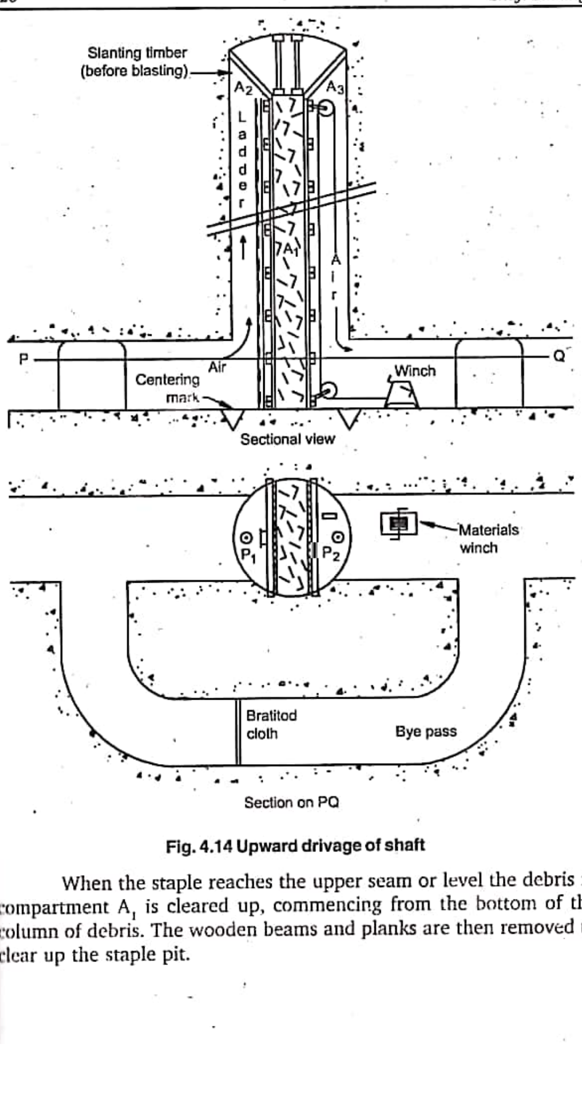


Fig. 4.14 Upward drivage of shaft

When the staple reaches the upper seam or level the debris in compartment A_1 is cleared up, commencing from the bottom of the column of debris. The wooden beams and planks are then removed to clear up the staple pit.

Special Methods Of Sinking

Ordinary methods of shaft sinking are not suitable in some cases and special methods have to be adopted under the following conditions.

1. Loose or unstable ground, such as sand, mud, etc.
2. Excessively watery strata.
3. A combination of the above two.

The special methods are as follows :

The Piling System

This method is known as simply "piling" or "sheet piling and is suited sinking through loose deposits of sand, mud, or alluvium *near* the surface upto a depth of 20 m. Interlocking steel piles, 6 m to 10 m long, are used and they are practically water-tight. Additional lengths may be available by welding or riveting two or three lengths of piles. At the surface, the piles are set up to form a ring and then they are hammered down in rotation, each member being driven a few metres at a time by a direct-acting steam piling hammer. (Fig. 4.15). As the piles descend in the loose gound, the latter, enclosed by the piles, is excavated and cleared up, but it should be remembered that the bottom ends of the piles are kept sufficiently ahead of the excavation to prevent inrush of water or loose sand. When the excavation reaches strong rock, permanent lining is constructed and the sinking then proceeds in the manner already described for normal conditions.

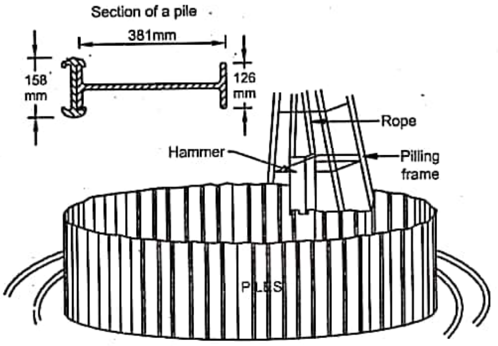


Fig. 4.15 Piling

Caisson Methods

The methods can be divided into three classes.

i) *Sinking drum process or open caisson method*

This consists of a cylindrical well of brick work, 0.3 m to 0.4 m in thickness over a m.s. ring having a steel cutting shoe. The shaft is excavated and the drum sinks down gradually by its own weight. As the drum sinks down, further brick work is added on the top. A compound sinking drum consisting of brick work surrounded by 13 mm thick steel plates is sometimes used to resist uncertain tensile stresses. Concrete sinking drums also can be used. Care must be taken to see that the drum descends vertically and with this object additional weights may be placed over the drum.

ii) *Forced drop shaft method*

This is commonly adopted where the strata consists of alternate tough and loose ground and also when the drop shaft refuses to sink further due to very high skin friction. In these cases sinking is carried out with the help of hydraulic rams which force down the cast iron drums. This method can be used for depth upto 60 m (Fig. 4.16).

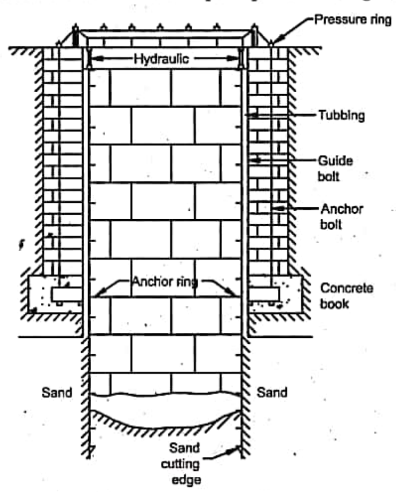


Fig. 4.16 Forced drop shaft

iii) ***Pneumatic caisson method***

This method is adopted when there is a danger of ground filing up the shaft or where there is considerable inrush of water under a small head. Compressed air is led into the chamber formed by means of a partition, 1.8 to 2 m above the cutting shoe compressed air keeps back the water and sand. An air lock is mounted on top of the partition as a passage for men material. The limit of the pressure of the air is 4 kgf/cm² beyond which persons cannot work. This method cannot be used for depths of more than 30 m.

These caisson methods are commonly adopted for the construction of foundations for bridges, tall buildings, etc.

Freezing Method

This method is used when the sinking is proceeding through an unstable or friable strata with heavy inrush of water, or sand connected with inflow of water and essentially involves the formation of a large block of frozen ground in the water-bearing strata. The frozen block prevents the influx of water into the shaft. The whole process can be divided into three operations.

1. The first operation consists of drilling holes, usually 150 mm diam. at 2.2 to 3 m intervals around the shaft from the surface or from a fore shaft. The holes, after drilling, are to be lined with special tubes and care should be taken to see that all the holes are vertical.
2. Inside the holes special small tubes are inserted to enable the cold brine (solution of $CaCl_2$) to be circulated. Cold brine, while circulating in the holes, extracts the heat from the surrounding strata and the circulation of brine is continued till a wall of ice of sufficient size is formed. Sinking and lining is carried out in the normal way after the formation of ice wall.
3. The third and final operation is thawing which consists in removing the ice wall by sending hot brine through the existing holes.

This method is very rarely used in India.

Cementation Process

This process can be used in all cases of shaft sinking, particularly in any fissured water-bearing strata except in running sand or loose ground. It can be successfully applied in sinking even when the inrush of water is heavy.

Treatment of ground around the shaft is carried out to achieve one or more of the following objectives : (1) To stabilise the collapsing ground, (2) To reduce the inflow of ground water, (3) To avoid flooding, (4) To prevent sand "boiling". The operation is usually carried out in 2 phases, one before the sinking and the other after shaft lining. Ground conditions usually dictate the pattern of treatment.

The pre-sinking treatment reduces the surprise-stoppages of the sinking due to unfavourable ground conditions. Further, by reducing the amount of water inflow it not only saves expenditure on the dewatering pumps but substantially enhances the rate of sinking and the quality of the work.

On occasions post-cementation treatment may be necessary to have improved working conditions in the mine. Otherwise humidity in the underground excavation would create serious ventilation and corrosion problems.

The method consists in drilling the holes as shown in Fig. 4.17 and then injecting a slurry of water and cement under pressure through the holes till they are completely sealed off. In the past injection was done at low pressures like 6 kgf/cm² but it has been proved that high pressure of the order of 300 kgf/cm² can be used successfully. The water cement ratio can be changed according to the requirements.

A process known as *pre-silicatisation,* which reduces the friction of the rock to the passage of cement is necessary in certain types of rocks. Extra holes are drilled for the purpose and are treated first with silicate of soda and then with aluminium sulphate. This process of treating the holes with the chemicals is known as *silicatisation.* The holes to be treated with chemicals are known as "product-hole" and their number is usually three times that of cementation holes.

After cementation of holes the shaft sinking proceeds in the usual manner.

Shaft Sinking At Sudamdih and Monidih Collieries

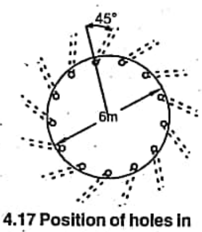
Sudamdih was the first mine in India where two shafts were sunk in the early sixties with the help of Polish engineers in a manner that was completely new for the mining industry in the country. The distinctive features adopted for the first time in shaft sinking at Sudamdih, and later at Monidih, were :

1. Sinking was carried out with the help of one head gear and 2 winders, diametrically opposite to each other on either side of the shaft.
2. A double deck platform was used for simultaneous sinking and concrete lining.
3. Delay detonators were used.
4. Shotfiring was carried out with the current from the electric power line at 550 V.
5. Grab loaders, suspended from the double deck platform, were used for loading of muck at shaft bottom.
6. High rates of sinking with completion of shaft lining were achieved. The average progress per month was 25 metres and the maximum was 50 metres.
7. Entire shaft depth was lined with concrete.

At Sudamdih No. 1 shaft is 450 m deep, finished dia. 7.2 m and the adjacent No. 2 shaft is 420 m deep, 6.5 finished dia. All the coal production comes from 400 meters horizon (400 m below M.S.L.) by means of skips in No. 1 pit. During sinking at shaft No. 1 a temporary head gear and two winders were installed but the headgear and winders were replaced later by permanent ones.

A general time-study of different operations :

1. Drilling 3 to 4 hours.
2. Charging and blasting 11/2 to 2 hours.
3. Clearance of smoke and cleaning of D.D. platform 1/2 to 1 hour.
4. Lowering of pipes, equipment and grab 1/2 to 1 hour.
5. Loading of much 18 to 21 hours.
6. Lowering the steel shuttering 45 min to 1 hour.
7. Placing and plumbing the shuttering 1 to 11/2 hours.
8. Extension of concrete pipe line (flexible portion 1/2 to 1 hour
9. Pouring concrete 3 to 4 hours.



4.17 Position of holes in cementation process

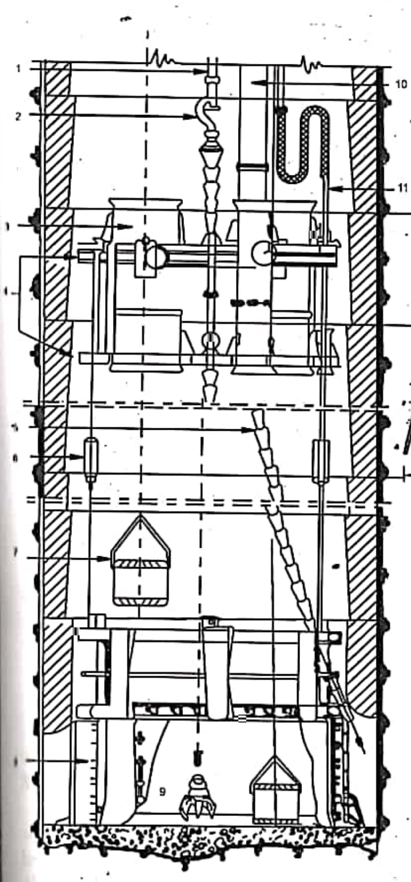


Fig. 4.18

1. Concrete pipe line
2. Kinetic energy damper
3. Opening for bucket
4. Two deck platform
5. Flexible portion of concrete pipe line
6. Pulley block for shuttering
7. Sinking bucket
8. Steel shuttering
9. Grab loader
10. Ventilation tube
11. Compressed air pipe

Drilled Shafts

The drilling of mine shafts by suitable machines is the most progressive method of shaft sinking adopted in Russia and other advanced countries of the west. The process is completely mechanised and all the sinking work can be carried out by sequence controlled automatic machines so that there is no need to keep any men at the shaft bottom. Basically a shaft borer is similar to a large tunnel-boring machine. A cutting head equipped with roller bits continously excavates the entire shaft cross-section at once. The removed fine muck is pumped as a slurry to the surface or dropped through a pilot hole for underground removal. In many cases sprayed or concrete is used as initial lining. The machines used in Russia for such shaft sinking are suitable for shaft diameters 6.2 m to 8 m (excavated diameter) upto a depth of 800 m through rocks of medium strength like those available in coal mining localities. The average speed is 100 m per month. The shaft is drilled in 3 stages; a pilot hole 1.2 m dia. is drilled to the full depth with a tricone bit; thereafter the reamer enlarges the hole to 3.6 m and then to 6.2 m. The reamer is rotated by the drill pipe through a square section pipe at the surface, operated by electric motors. During drilling, the hole and shaft are full of mud-water mixture (mud flush) which exerts a pressure within the shaft and takes the place of temporary lining. It also flushes the reamers, removing the broken rock to the surface with the help of an air-lift pump. The permanent shaft lining is built after the entire length of the shaft has been widened to its full diameter.

Acutal capital expenditure for boring per metre of shaft by the technique of shaft boring is higher than by the conventional drilling blasting-mucking method and other special methods of sinking. But the cost is likely to be reduced with more knowledge of the 'knowhow' in handling of rigs.

Such machines are not used in Indian mining practice.

The current trends in the design of the shaft boring rigs are :

1) Drilling of shot holes to a depth of 6 m. (2) Increasing the hoist speed for deep shafts to 15-18 m/sec. (3) Automation of sinking process.

The progress available with shaft sinking machines may be appreciated from the following performance of a machine manufactured by WIRTH, a leading manufacturer in West Germany. In early 1983, one 7 m diam. shaft was sunk in Alabama, U.S.A., completing the sinking of the shaft, 650 m depth, in 6 weeks. Sinking rate at one stage was 35 m/day. The period is inclusive of the time spent on shaft-lining.

The Company manufacturers machines which have achieved sinking rates of 7 m in a 7-hr. shift. The machine is adopted for cuttings-removal system and works in conjuction with the company's pilot hole machines for sinking of blind shafts. The cuttings are removed upwards.

✧ QUESTIONS ✧

1. A shaft 6 m dia. is to be sunk to a seam 150 m deep. The shaft is to pass through 12 m thick alluvium at the surface and strong sandstone. Describe briefly the methods adopted for support of sides.
2. What are the special methods of shaft sinking ? Under what circumstances are they followed ? State their limitations.
3. A shaft 5m dia. is being sunk through rock consiting of shale and sandstone. Describe the arrangement of shot holes, blasting and disposal of debris.
4. Describe a method of driving a staple pit from a bottom seam to a top seam, 10 m above.
5. Describe a method of simultaneous shaft sinking and shaft lining.

✧ ✧ ✧

Chapter -5

Opencast Mining

Opencast mining or quarrying of minerals is easier than mining by underground methods. During quarrying the alluvium and rocks below which the minerals lie, are removed and dumped, in the initial stages, in a place which is not required in future for quarrying, residential or other purposes. The mineral exposed is completely extracted. Opencast mining is also known as open-put mining, open-cut mining, surface mining and also as *strip mining,* the later term being commonly used in the U.S.A. for opencast mining of coal. The overburden and the mineral, coal, are excavated in long strips of a few metres thickness and hence the operations are termed strip mining. The operation of removing overburden and extracting mineral is done by one of the following methods.

1. Manual quarrying : In this case manual labour is employed. Small drilling machines, drilling 1.2 m tc 1.8 m deep holes, 37 mm diameter, are used and the holes are blasted with gun-power or other explosives. The overburden and mineral are manually loaded into tubs which are hauled by rope haulages or locomotives. Tipping trucks are also sometimes employed and manually loaded.

2. Mechanical opencast working : In this method heavy earth moving machinery like draglines, power shovels, rear-dumping trucks (common type being Haulpaks), well-hole drills *etc.* are used. The blast holes are 6 m to 18 m deep, and 125 mm to 250 mm diameter. The rock is blasted by liquid oxygen, open cast gelignite or other high explosives. Mineral and overburden are transported by locomotives, belt conveyors or large trucks known as dumpers. Bucket wheel excavators are used in some mines for soft rocks *e.g.* at Neyveli lignite project.

A method of surface mining known as *placer mining* involves mining and washing together of generally unconsolidated or semiconsolidated rock near the ground surface and the method is normally not treated as opencast mining but a variation of it.

Glory hole mining is a method where the mineral is excavated in small open pits but is transporated to the surface through underground excavations and transport system.

Nearly 70% of the mineral production in the world comes by opencast mining and in India this method of mining accounts for nearly *75% of our mineral output.*

Quarriable Limit

The cost of removing overburden to extract mineral lying below it goes up as the quarrrying operations extend to the dip side of the property and the thickness of overburden increases. The stripping ratio, thickness of overburden; thickness of mineral deposit therefore decides the economic working limit of quarrying, *i.e.* the quarriable limit. The softer the rocks, the less is the expense of oberburden removal and higher is the stripping ratio. The wages of labour, the selling price of mineral and the margin of profit and the major considerations in deciding the limiting ratio which is as follows in coal mines :

i.	Manual quarrying	1.5 : 1
ii.	Semi-mechanised quarrying	2 : 1
iii.	Mechanical quarrying :	
	With dipper-shovel, dumper combination :	4 to 5 : 1
	With draglines :	8 to 10 : 1
	With bucket wheel excavators :	3 to 4 : 1

The maximum depth from the surface in existing mines in our country is 120 m but future mines are planned to reach a depth of nearly 480 m.

Advantages Of Quarrying

There is no problem of roof control or ventilation. Full extraction of mineral is possible. No mineral is blocked in shaft pillars, support of main roadways, etc. as in underground mining. Quick return on capital and early extraction are possible without the need to wait for a long period of development work or unproductive work like shaft sinking, etc. Large ouput is available from a small area and supervision is easy owing to concentrated work. Artificial lights are necessary only after dark.

Dangers and hazards are less as compared to underground mining. There is no risk of gas explosion. Very few stringent mining regulations are applicable to quarries. For example, compared to underground mining less number of competent persons are to be appointed and less number of statutory inspections are necessary.

Better sanitary conditions can be maintained.

High efficiency of mine workers; ease in loading the tubs in unconfined space and natural light gives high O.M.S. (output per manshift).

Female labour can be employed. All the members of a family are often employed and the accommodation problem is simplified.

Once the mineral is exposed, output can be easily varied to meety wagon supplies, or consumer demand.

Training of operatives is easier.

Large scale mechanisation is possible as there is no restriction on the dimensions of machines to be used. Unlike in underground mines, machinery working at high voltage can be employed.

Disadvantages

Among the disadvantages of quarrying are :

Work is affected by weather. During winter nights, and summer mid-days efficiency of workers is very low. During rainy season unless effective steps are taken to dewater the mine mineral which is at the lower levels, cannot be worked and mining comes pratically to a standstill.

Surface land is destroyed and is rendered unfit for agriculture and residential purposes.

Mining lease gives only underground rights ; surface rights have to be acquired for quarrying.

The method is uneconomic for working mineral beds at depth.

Where quarrying aims at quick return on capitcal, outcrop mineral is also mined. As it is inferior in quality due to weathering and percolating of water, the overall quality available to the consumers is affected in earlier phases of mining.

The quarried area and the OB heaps present an unpleasant sight. In some foreign countries the mining law requires that the quarried area should be filled up with overburden and restored to the prequarry state fit for agriculture. Marshy land, after extraction of underlying mineral and restoration of surface has, in a few cases, resulted in a good agricultural area. Such low requiring surface restoration to pre-quarry stage does not exist in India.

The overall O.M.S. is low due to a large labour force engaged in OB removal in manual quarrying.

A property with extensive area on the strike and containing a thick mineral bed moderately inclined, lying at shallow depth, is ideal for quarrying. In India coal seams with inclination as steep as 1 in 3 have been worked by mechanised opencast mining methods in Karanpura field. Seams at shallow depths which are actively gassy, liable to spontaneous heating or with bad roof should preferably be extracted by quarrying as the mining legislation for underground working of such seams is stringent.

Before the quarrying operations are undertaken it is necessary to vacate buildings and divert electric overhead lines, aerial ropeways, water mains, telephone lines, roads, railway lines, streams, etc. from the area which has to be quarried. The trees have to be cut. Sufficient space for dumping of overburden, not far from the quarry, has to be considered. Where mechanised opencast mining is to be adopted, plans should be prepared to show the contour lines and the thickness of coal and overburden and good roads without steep inclinations and sharp curves should receive attention. Dumpting yard for OB should be so selected that wind does not carry the dust to residential colony. By dumping of OB if the ponds, paddy fields, mango/coconut groves, not under the ownership of the mine owner, are likely to be affected, the compensation payable should not be ignored.

If heavy explosive charges have to be blasted as in mechanised quarries, the quarry site has to be far away from residential area (beyond 300 m). Such heavy blasting may cause cracks in old buildings resulting in demand for compensation.

On the surface referecne lines have to be marked on a square pattern, every 30 m apart, for monthly measurement of the excavation and they should extend 50 m beyond the limits of the proposed quarry. Junctions of the squares should be marked by permanent pegs in brick pillars.

Formation of Benches :

The overburden and mineral deposit can be extracted by formation of benches or by keeping the surface sloped so that the angle of slope does not exceed 45° from the horizontal. A bench has two elements, the floor and the face (high wall). The width of the floor should not be less than the height of the face (high wall) and the heights of benches are as follows :

1. *Loose material :* In alluvium, morrum, loose earth, etc. which is likely to slide the bench height should not exceed 1.5 m. During the rainy season, there is possibility of land slide of the loose debris and it is desirable to keep the width of the floor much larger than the height of the bench.
2. *Coal :* The height of coal bench, *i.e.* the coal face, should not exceed 3 metres.
3. *Sand stone and hard rock :* Benches in sand stone and hard rocks are rarely vertical but generally sloping at a small angle with the vertical. The mining regulations do not stipulate definite bench heights in hard rocks except that the bench width has to be more than the bench height. In manual quarry the bench height is usually 3 metres to 4.5 metres and in mechanised quarries, more than 5.5 metres and depends upon the height of the boom of shovel above the bench floor. Suitable bench height for a 2 m^3 shovel is 6 to 8 m and for a 3.5 m^3 shovel, about 12 m. The slope of the high wall is usually 20° off vertical and depends upon the travel of the bucket during loading. The width of bench floors in mechanised quarries is usually 15 metres and preferably more for movement of dumpers, tractor loaders and other equipment.

   Gradients of roads in quarries for tyred vehicular traffic should not exceed 1 to 10.

Manual Opencast Working in Coal

The quarriable area is divided into sections along the strike so that overburden extraction takes place in some sections and coal extraction in others which have the coal already exposed after overburden removal. Small pillars of the rocks excavated are left for measurement. These are called *witness* or 'Sakhi' in Hindi. They are removed after measurement of excavated area, usually once a week, is over. The height of such 'sakhi' should not exceed 2.5 m and where the height of such pillar exceeds 1.25 m its base should not be less than 1.5 in diameter.

Removal Of Overburden

The soft material like earth and weathered rock is cut by earth cutting picks. A team of workers consits of 3 or 4 members, one cutting and two loading. As female workers are allowed in quarries, a team of 3 workers usually includes one or two females members who are normally permitted to work only between 5 a.m. and 7 p.m. under the Mines Act. Each team is allowed a small plot usually 4.5 m × 4.5 m. The average output per worker in soft rock or earth is about 2.8 m^3 (in situ) per day.

In the hard rocks which need blasting holes are drilled.

(a) manually with the help of hexagonal steel rod with chisel end

or

(b) by compressed air hammer drill (jack hammer).

Method (a) is used in a small quarry where the output is only 50 to 100 te of coal per day, electricity or compressed air is not available and labour is cheap. A hole is 1.2 m to 1.5 m deep and one man can drill 5 to 8 holes, each 1.2 m deep in a shift (8 hrs.) Method (b) is now-a-days commonly employed. Compressed air is supplied by steel pipes 50 to 100 mm dia, up to central places and branch pipes supply air to drills through hose pipes. Two workers (one driller and one helper) drill 40 to 50 holes, each 1.5 m deep, in one shift. The holes one placed 1.2 to 1.5 m apart and are blasted with gun powder or other suitable explosives. Blasting is done during the rest interval of the workers to prevent frequent interruption of work.

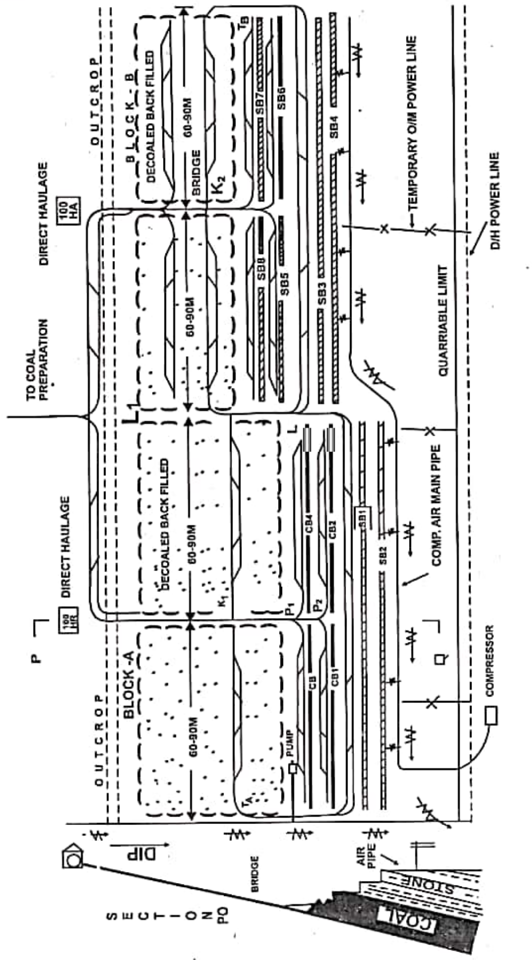


Fig. 5.1 Layout of a manual quarry (coal). W with indicates water drains

The overburden is loaded into tipping tubs (0.73 m^3 capacity) which are hauled by direct or endless haulages. In seams of mild gradients locomotives may be used. The haulage track is taken to each bench or alternate benches. Blasted overburden of higher bench is sometimes dropped on the lower bench for loading into tubs. Overburden is dumped to the rise of the outcrop or beyond the quarriable limit but as the coal extraction proceeds, the overburden may be dumped in the area from which coal is extracted - an operation known as **back filling.** The overburden should be so dumped that it does not roll down at the coal benches when it assumes its angle of repose, nor should it choke water courses or damage paddy fields, other agricultural area or water reservoirs.

Extraction Of Coal :

The coal which is exposed after removal of overburden is blasted after drilling holes. The same drills which are used for stone may be utilised for coal also but if the compressor has a limited capacity electrically operated drills are used for coal due to their relative lightness and better performance. The spacing of holes in coal is 1.5 m to 2.2 m, the depth varying from 1.2 m to 2 m. The coal available per kg of high explosive like special gelatine (60 to 80%) is generally 10-12 tonnes. With blasted coal, the average loading performance per loader is nearly 4 tubs (1.1 m^3 capacity).

Fig. 5.1 shows the layout of a large manual quarry, after it has advanced some distance along the dip. The entrance to the working places is by steps and inclined roads. The permanent installations like haulages, compressors, power transmission lines, etc. are installed in such places that their frequent shifting is not necessary when the stone benches or coal benches advance. On a level track hand pushing of empty tubs having pedestal bearing is not uncommon for a distance of 100 m. If the same track is used for loads as well as empties, the gradient of the track should be nearly 1 in 80 in favour of loads. These factors limit the length of benches.

The property is divided into blocks, each 120 to 180 m long along the strike. Each block has a direct or endless haulage in the middle and the benches are formed along the strike on either side of the haulage. Block A which is shown more advanced than Block B, raises coal from the coal benches CB1, CB2, CB3, CB4. Each bench is 3 m high and has a haulage track on its floor. From the clipping point P1 to the junction P2 the haulage track is either level or slightly rising inbye at 1 in 80 or so. The stone benches SB1, SB2, are ahead of the coal benches so that the exposed coal lasts for 2-4 weeks. Stone from the higher bench SB2, is dropped on the lower bench SB1 which has haulage tracks on its floor. The stone is taken in tipping tubs along a level track TA by hand pushing for dumping in the de-coaled area. The track TA is along the barrier. The tubs cross the direct haulage track over a bridge K_1 to fully utilise the decoaled area for dumping. As the stone benches advance the position of the bridge K_1 has to be shifted to the dip side. Another level haulage track is taken from the stone bench SB_1 to the decoaled area over a ledge LL_1. This ledge is 3m wide at the top and is a solid barrier of coal and stone left between blocks A and B. The coal of the ledge is not recoverable.

In block B, there is emphasis on removal of overburden. The direct haulage track is along the floor of the coal seam and more stone benches than in block A are provided for employment of a large number of workers on overburden removal. The stone benches SB_5, SB_6, SB_8 are served by the central direct haulage. Stone benches SB_3 and SB_4 are worked by a level track T_B and another level track passing over the ledge LL_1 stone is taken over these level tracks in tipping tubs for dumping in the decoaled area in the same manner as described for block A. The bridge K_2 serves the same purpose as bridge K_1 in block A and is advanced towards the dip at intervals.

During rainy season coal raising may be suspended in block A which is on the dip side, and only overburden removal may take place from the higher stone benches which are free from water. From block B, only coal raising may take place during monsoon and the block can be kept free from water and making a through connection 1.8 m high in the ledge LL_1 for the water to gravitate to block A.

Drains for water, as shown in the figure, called garland drains are cut to minimise inflow of surface water into the quarry. Pumps are installed at possible places of heavy accumulation of water. The installation of compressed air pipes and the overhead power lines is as shown in the figure.

MECHANISED OPENCAST MINING

Opencast mining is the oldest method of excavating minerals but the mining operations have been mechanised by the use of heavy earth moving machinery during the last 50 years resulting in excavations on a scale which was unthinkable half a century ago.

Some of the coal mines planned for large production (million tonnes per year) are : Kusmunda - 6.0, Mukunda - 15.1, Gevra - 14.0, Nighai-12, Rajmahal-10.5, Jayant/Dhudhichuva / Khadia-10.0 each,

Dipka Expansion/Anant-8.0

For mechanised quarries, employing heavy earth moving machinery the DGMS makes byelaws coverings bench sizes, roads, etc. These have to be studied before planning a mechanised quarry. Mechanised opencast mining is preferred when there is a thick mineral bed of mild inlcination, practically continuous and not in pockets, at a low depth and the reserves are plentiful. For coal, a seam of less than 6 m thickness and with less than one million tonnes of quarriable reserves will not justify the heavy capital expenditure, large amount of interest on it and the depreciation charges.

The overburden may be removed by a combination of dozers and scrapers if the rocks are soft. If they are hard, blast holes are drilled by wagon drills or well hole drills and blasted with explosives. The blasted rock is loaded into dumpers by dipper shovels or tractor shovels. Draglines are also used where the overburden is alluvium sand or soft rock, but if it consits of hard rock it is loosened by sprase blasting for loading.

The equipment commonly used in a mechanised quarry for drilling, loading and trasporting is briefly described below.

Crawler Chain Vs Pneumatic Tyred Equipment

The machines employed in a mechanised quarry are mounted on pneumatic tyred wheels if they have to be towed from one place to another e.g. compressors, wagon drills, etc. Self propelling units having their own engines (line tractors, dozers, shovels, cranes, etc.) may however be mounted on pneumatic tyres or crawler chains. Table 1 shows the approximate rolling resistance of various road surfaces to pneumatic tyres and crawlers.

Table 1 Rolling resistance

Types of surface	Pull in kgf per 1,000 kgf of gross weight	
	Low pressure tyres	Crawler chains
Smooth concrete	18	28
Good macadem	30	35
Earth roads, dry, dusty	40	43
Unplough earth	60	55
Earth road, rutted, uneven	90	70
Loose sand and gravel	110	85
Construction haul roads or roads in loose soil	145	112

A resistance of 9 kgf per 1,000 kgf is roughly equivalent to that offered by an upgradient of 1% i.e. a road rising 1 m vertical through 100 m horizontal.

It will be seen that the advantage of pneumatic tyres receders rapidly as the ground conditons worsen and for rough roads a crawler unit is at an advantage. Machines using crawler chains are not allowed to cross public roads which are damaged by the cleats of grouser plates and the latter should be replaced by plain plates preferably with rubber soles, for such crossing. Speed of crawler chains is slower than that of tyres. Where public roads have to be crossed often, or where the flitting is frequent, it is an advantage to use pneumatic tyred equipment. In the field crawler mounted equipment scores over the tyred one for climbing steep gradients and has better traction on rough roads and better gripping capacity on bench floor when the machine has to dig hard, *e.g.* when extracting toe. Crawlers can negotiate sharp turns, a big advantage in a quarry of limited size. The quantity of mineral and rock that remains frozen in the roadways of a quarry where tyre mounted dozers and tractor shovels are used is much larger compared to that when similar equipment on crawlers is deployed.

Diesel-Driven Vs. Electrical Equipment

The equipment used in quarries may be diesel operated or electrically operated. Machines which have to move frequently from place to place are operated by diesel engines *e.g.* bulldozers, scrapers, graders, dumpers, tractors dozers. But the equipment which has to work from one site over long periods in a shift may be electrically operated or diesel operated e.g. dipper shovels, draglines, compressores, well hole drills, bucket wheel excavators. Permanent or semi-permanent machines are always powered electrically *e.g.* large centrally located compressors, pumps, *etc.*

An electric shovel avoids fuelling problem, is comparatively quiet, simple and maintenance cost is cheaper than for the diesel one. In electric shovel motors can be placed so as to eliminate complex gear trains and chain drives, and controls are easy to operate. Initial cost of electric shovel is however higher than for a diesel unit. Ward-Leonard system is the standard method of drive and control on large H.P. shovels and draglines. Dipper shovels of smaller size (upto 3.5 m^3) are usually diesel driven, but electric drive is preferred for larger machines. The electric shovel, due to limited length of the trailing cable, operates only within a restricted area and the trailing cable being heavy needs men to handle it during movement from one place to another. Some shovels are diesel-electric. In this type of drive a diesel engine mounted on the shovel itself drives a generator that supplies electricity to motors which to the heavy work of the machine and the various controls on the shovel. Such drive is found in medium large shovels.

The voltage of small machines is usually medium, 400 or 500, though H.T. voltage (3300-6600) is often essential for large machines like dipper shovels which have to move very little during a shift. The bucket wheel excavators at Neyveli are supplied power at 11 kV through T.R.S. cables and transformers in the machine step down the voltage to 3.3 kV for some motors and to 400 V for some other motors. Overhead power transmission lines bring the power to a convenient point near the quarry and from there the pliable armoured cables which are sufficiently flexible carry it to the machines. In mechanised quarries it is difficult to construct the power lines entirely outside the blasting zone as trailing cable lengths are restricted by the Electricity Rules and such overhead lines and the insulators sometimes break due to flying stones of blasting.

Buldozer

A buldozer is often referred to simply as a **dozer.** It is a tractor with a pusher blades attached to the front portion. The tractor is the diesel-operated power unit equipped with either crawler chains or rubber tyred wheels for liftting. The pusher blade can be raised or lowered or tilted through small angles horizontally by rams operated through hydraulic pressure or by ropes. The dozer blade is used for pushing loose material or for digging in earth, sand and soft weathered rock. The machine is also engaged for levelling or spreading earth, for levelling of rock spoil in the dumping yard, grading and compacting temporary roads, pushing mineral into sub-ground level bunkers through grizzly, for towing dumpers, etc. It also serves the purpose of pushing boulders, pulling down trees, and is an essential equipment to push scrapers. A dozer equipped with a fork like attachment is known as **ripper** and operates like a plough to loosen moderately hard rock. The loosened rock may be loaded by a scraper. A dozer can dig 1.2 m to 1.5 m below ground in earth or weathered rock. (Fig. 5.2).

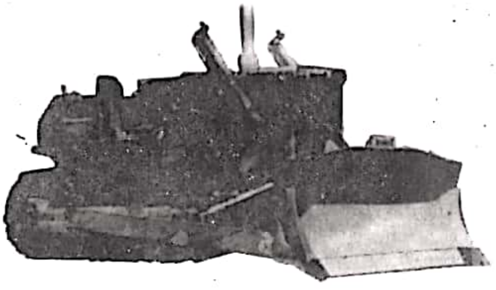


Fig. 5.2 Bulldozer

Scraper

This machine is diesel-operated with pneumatic tyred wheels and has at the centre a bowl fitted with a cutting blade at bottom. The blade is reversible and can be replaced when blunt. Its working may be compared to that of a lawn power. As a scraper is pushed forward by a dozer, its blade cuts a thin slice of earth usually between 75 mm and 225 mm thick over a distance of nearly 30 m. The earth is automatically collected in a central bowl whose capacity ranges from

3 m^3 to 22 m^3 and it takes nearly one minute for loading. When the scraper is fully loaded its bottom opening is closed by the operator through manipulation of a cable (rope) and the loaded scraper, with the bowl lifted, travels to the dumping yard on its own power. At the dumping yard, as the scraper moves, the bottom opening of the bowl is opened and the contents are unloaded in a layer 150 mm to 250 mm thick, over a distance of 30 to 70 m. The bowl is always bottom discharging. Scrapers are unsuitable in soils with stumps, large boulders and hard rocks. When the ground is hard, it is necessary to rip the surface with the help of a ripper before loading by a scraper. Sandy soil is best for a scraper which has to be stopped during rains, if engaged in alluminum. (Fig. 5.3).

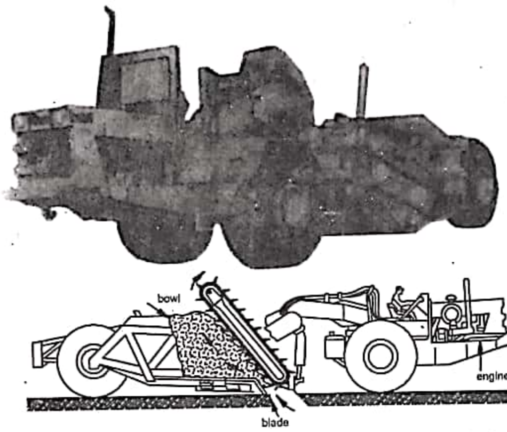


Fig. 5.3 A scraper showing the bowl (a cutaway view) getting loaded.

Scrapers are used in coal mines for cutting and transporting weathered sandstone as well as coal. The coal excavated by it is however smaller in size. A Scraper may take 5 to 6 minutes for a complete cycel of loading and unloading if the total up-and-down distance of a trip is nearly 300 m. One-way traffic of loaded and empty scrapers is desirable for good results. One dozer is normally sufficient for every two scrapers used.

The scraper manufactured by BEML has the following **main specifications.**

- Flywheel H.P. of engine 332 at 2100 rpm;
- Capacity : payload 23000 kg; struck 11.5 m^3, heaped 16 m
- Max. travel speed (forward) 44 km/hr
- Overall dimensions mm : length 12600 ; width 3470; height 3890.
- Net weight (no load) 26584 kg.

Ripper

A ripper is a machine which cuts, as it travels, 0.6 to 1 m deep furrows in the ground, and it can be well compared with the farmer's plough. The ripper is essentially a crawler mounted heavy duty diesel tractor with a ripper attachment as shown in Fig. 5.4. Like a farmer's plough, the ripper with the ripping tool thrust into the ground by hydraulic pressure, travels along close paths, 1.2 to 1.5 m apart and during the travel rips open the ground. The broken ground or rock can be dozed to form a stockpile for convenience of loading or can be loaded by a scraper. If the overburden or mineral is suitable for ripping its breaking is possible with the help of a ripper and the process of drilling and blasting can be dispensed with. Soft rocks and medium hard rocks, below hardness 5 on Moh's scale, which are laminated and stratified, provide suitable material for ripping. The alluvial surface deposits, weathered sandstones and shales underlying them in the coalfields can be easily ripped and the relative rippability of the rocks can be known with the help of an instrument known as *Refraction Seismograph.*

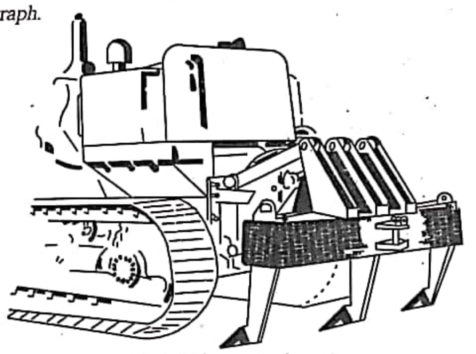


Fig. 5.4 Ripper attachment

The Refraction Seismograph operates on the principle that "Sound waves travel subsurface material at different velocities, depending upon the degree of consolidation of the material". It is believed that the same factors that affect consolidation also affect rippability. Thus poorly consolidated material with low seimic wave velocities could be ripped easily, while highly consolidated material with high velocities would be difficult to rip.

Equipment needed for seismic analysis includes a source of a sound or shock wave, a receiver, an electric counter, and a set of cables. The main items are :

1. *Refraction Seismograph :* An electronic counter that determines the time interval between the strike of the hammer and the arrival of the seismic wave at the geophone.
2. *Geophone :* Receiver of sound waves. A *geophone* is a velocity gauge suitable for detecting frequencies in the range of 1-100 Hz. The geophone converts the mechanical vibrations into its electrical analogue. The electrical signal is then amplified and transmitted to the monitoring station.
3. *Sledge Hammer and Impact Plate :* Source of sound wave transmits sound through earth and also through an impact switch having direct connection with seismograph, through a connecting wire.
4. *A 30 m Tape :* For measuring distances between the geophone and various impact points (wave sources).

The seismic wave is produced by a sledge hammer striking a steel plate at various distances from a geophone receiver. Immediately upon impact, a wave "front" composed of innumerable seismic waves travels in all directions away from the point of impact, or source. Some of the waves are refracted into the layers of sub-surface materials and the angle of refraction is determined by Shell's law which gives the following relationships.

$$\frac{\text{Sine of angle of incidence}}{\text{Wave velocity in upper layer}} = \frac{\text{Sine of angle of refraction}}{\text{Wave velocity in lower layer}}$$

The geophone receiver is sensitive only to the first seismic wave that reaches it. Thus, either the wave which travels the shortest distance, or one which travels a longer path but which includes a high velocity segment, arrives first at the geophone.

In addition to determining the degree of consolidation or rippability of each layer, it is also possible to determine the depth of each layer.

In iron ore areas of Goa the practical results obtained with seismograph were as follows : Seismic velocity in overburden (practically laterite) was 600 to 1,200 m per second. In iron ore it was 1,050 to 1,500 m per second, but in some cases velocities as high as 1,800 to 2,100 m per second were also recorded.

When selecting a tractor for ripping purposes, it is necessary to consider (1) the down pressure on the tooth to determine whether penetration can be accomplished, (2) the tractor H.P. which should be capable of advancing the tooth through the rock and break it, (3) tractor weight which provides traction for full use of the H.P. in advancing the tooth.

The tractor speed is 0.8 to 2.5 km/hr during ripping. If the rock is soft it is advisable not to increase the speed but to add one ore more ripping teeth. The distance between adjacent furrows during ripping may be 1 to 2 m and the harder the rock, the closer are the furrows.

In some rock formations ripping is possible after sparse blasting of widely spaced charges.

Tractor Shovel

It is essentially a diesel operated tractor with a bucket as the front attachment and is called a front-end loader or payloader. It may be on pneumatic tyres or crawler chains. The tractor shovel attachments consists of a push frame and a bucket that can be raised, lowered and dumped hydraulically or mechanicaly. The shovel usually has a pusher fan so that the dirt falling form the bucket will not be sucked back towards the operator and engine air intake (Fig. 5.5).

For digging, the complete tractor shovel has to move forward toward the bench and for unloading the contents of the bucket the entire unit has to come back and position itself conveniently to empty the bucket on to a dumper. Its rate of digging and loading cannot, therefore, be as fast as with a revolving shovel, described later.

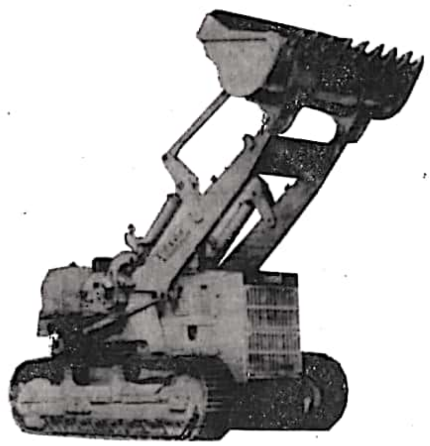


Fig. 5.5 Tractor shovel

Tractor shovels have been employed in some mines to load the stacked mineral at the siding into railway wagons or to push it into ground-level bunkers. Capacities of the buckets are from 0.57 m³ upwards. Heavy rock buckets for handling blasted rock carry teeth as a standard equipment though the buckets used for coal handling need not be so equiped.

Main specifications of two wheel-loaders (B.E.M.L.)

	WA 200-1	Model-3035
Bucket capacity m³	1.5	3.3
Heaped 2:1 (SAE) m³	1.7	3.8
Flywheel H.P.	108 at 2,400 RPM	300 at 2,100 RPM
Max. Travel Speed		
Forward, km/hr	37	31.6
Dumping Reach, mm	1,025	1,300
Turning Radius, mm	5,620	6,300
Overall Height, mm	3,190	4,075
Length, bucket on ground, mm	6,715	9,200

An excavator, technically speaking, is any machine which excavates the rock or earth and swings or transports it, within narrow limits, to an adjacent place or dumps it on to a receptacle like a dumper or railway wagon. In this sense, a tractor shovel which cuts or digs to some extent below the flow on which it stands, may well be considered an excavator - the name *traxcavator* for the tractor shovel manufactured by one company aply conveys the meaning-but, in earthmoving terminology the term *excavator* covers machines of the following type :

1. Power shovels like dipper shovels, stripper shovels and back-hose or pull shovels.
2. Draglines
3. Bucket wheel excavators

A power shovel is a shovel using electric or diesel motive power for its operation, as distinct from a hand-operated shovel. The functions of power shovels are very simple. Basically, these machines lift fragmented rock, and swing it to a different location such as dumpers or spoil heaps.

The **main components** are :

1. Propelling arrangements consisting of either crawler chains or pneumatic tyres.
2. A deck or cab mounted on a turn table and housing the prime mover, all the controls for operation, cable (wire rope) drums and the operator's seat. The deck or cab can swing through 360° independently of the propelling crawler chains or tyres.
3. Deck swinging mechanism. When the deck swings, all the equipment mounted on it, including the boom and the bucket, also swings.

A shovels is made in three structural divisions. An automatically, the top or revolving unit is the head and torso, the mounting or travel unit is the legs, and the various attachments are the arms and hands. A revolving and a travel unit together make up a basic shovel which may be fitted with any of the five front attachments shown in Fig. 5.6. The machine may thus become any one of the following : a crane, a clam shell, a dipper shovel, a drag line, a pull shovel or a back-hoe.

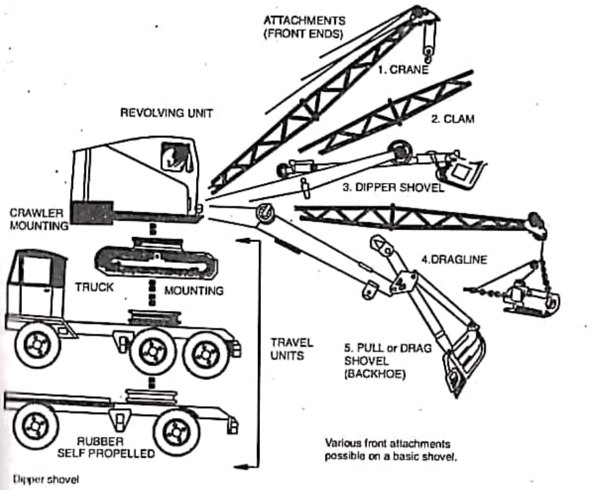


Fig. 5.6 Various front attachments possible on a basic shovel.

Dipper Shovel

This is a machine employed for excavating soft rock or loading fragmented rock from a bench and is very commonly used in mines. It is usually mounted on crawler chains. Fig. 5.7 B shows the principle parts and controls. The cab carries the power unit which may be an electrical motor at 3300 V, supplied with power from an external source through a flexible electric cable, or a diesel engine. The bucket (also called dipper) commonly used may be of 1 m^3 to 4.5 m^3 capacity. It is used for loading dumpers and for this purpose it has to stand on the floor of the bench. Watery conditions in the quarry are not suitable for efficient operation of this machine, as dumpers have to move inside the quarry. During operation, the crawlers are stationary within 3 to 5m of the toe of the bench. To load the bucket, the operator crowds it into the fragmented rock with the dipper stick and hoists it. As it moves through an arc in the rock pile, it gets loaded Fig. 5.6(a). It is then retracted and the cab, along with all the machinery mounted on it, the boom and the bucket, is swung horizontally through nearly 90° to poistion the bucket over the dumper. The bucket is bottom discharging and its door is opended by the trip cable. Normally five buckets are required to load a dumper. The teeth of the bucket wear out fast and when worn out, have to the built up to size by welding. The trip cable lasts for nearly 35 hours and the hoist cable, for nearly 100 hours. In one shift a shovel loads 450 to 500 buckets. Where the dumping yard is away from the quarry a dipper shovel loading into dumpers is advantageous.

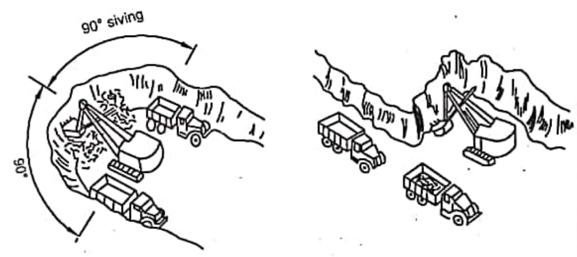


Fig. 5.6 A Dipper shovel in operation

Hydraulic shovels which eliminate use of wire ropes (cables) have become popular in recent years. The electric motor or diesel engine mounted on the shovel drives the hydraulic pump and the pressure developed is utilised for varous operations of the shovel. Hydraulic motors are of low speed, high torque with hydrostatic braking. One example of such hydraulic excavator is Poclain shovel of Larsen Toubro Ltd.

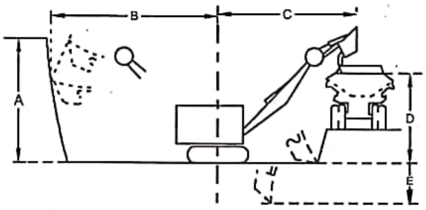


Fig. 5.7 (A) Working ranges of a dipper shovel (maximum).

Figures in brackets show the working ranges in m of Tata dipper shovel, model 1055 B, at a boom angle of 50°. Boom length is 8.53 m.

A – Cutting height (10.83)B – Cutting radius, (11.96) C – Dumping radius (9.51) D – Dumping height (7.25) E – Cutting depth below crawler level (2.51)

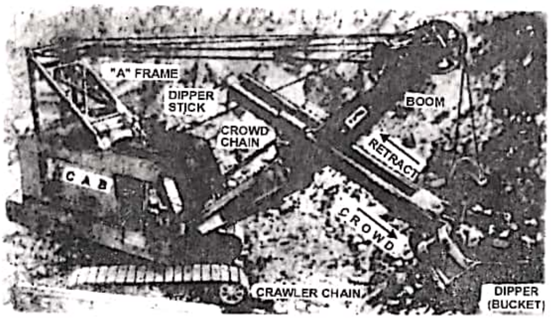


Fig. 5.7 (B) A dipper shovel, model TATA 1050 B

Dipper shovels commonly employed in our mines are of 2 m^3 - 4.6 m^3 bucket capacities. Only a few mines employ shovels of 8.3 m^3 or 10 m^3 capacity, *e.g.* Malanjkhand Copper Project employs 10 m^3 dipper shovels.

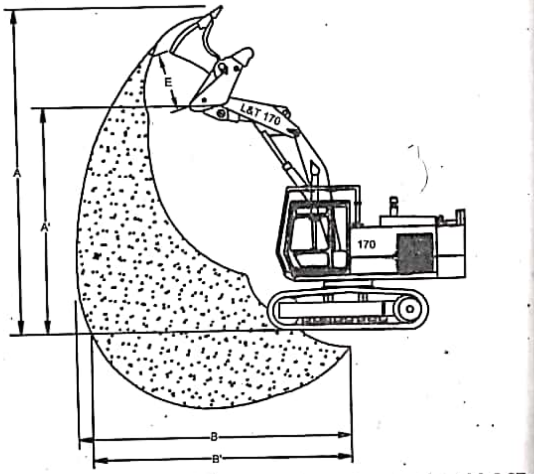


Fig. 5.8 Working ranges of L & T 170 CK-II bottom dumping shovel A-8.67m, A'–5.63 m, B–7.45, B'–7.06 (in m)

Hydraulic excavator. Gross power 240 H.P.

Under conditions existing in India the loading capacity of 2 m^3 shovel in good condition and well fragemented rock is as follows :

Per hour	80 passes of bucket.
Per shift (8-hours)	500 passes
Per day (2-shifts)	950 passes or 190 dumper loads or 1070 m^3 solid
Per week	5,800 m^3
Per month, dry	23,000 m^3
Per year	2,53,000 m^3

Loading capacities or performance of other shovels may be considered as follows :

3.5 m^3 capacity	400,000 m^3 per year.
4.5 m^3	550,000 m^3 per year.

Main specifications of some shovels of B.E.M.L.

	Pull shovel (Backhoe) PC220-3	Loading shovel PC 650-3 diesel	Dipper shovel 182 M-HR-17 (Elec.)
Bucket capacity SAE-Heaped, m^3	1.0	3.8	10
Operating Weight kg	22,000	67,000	3,27,500
Flywheel Horsepower (H.P.)	148 at 2,100 RPM	404 at 1,800	RPM 600
Boom Length, mm	5,850	8,400	12,240
Arm Length, mm	3,045	3580	8,230
Max. Travel Speed km/hr	3.4	4.1	1.61
Operating Data Max. digging/ cutting height,m	9.18	10.68	12.70
Max. dumping height, m	6.36	7.71	7.65
Max. digging depth, m	6.70	3.43	–
Max. digging reach, m	10.18	10.00	17.42
Overall length,mm	10,000	9,540	N.A.

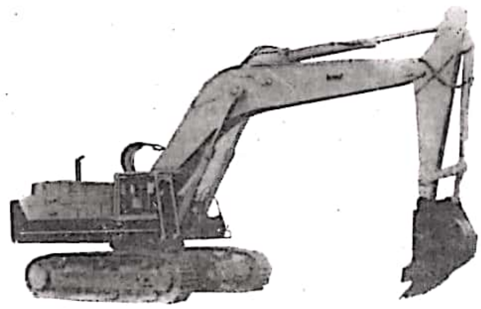


Fig. 5.9 A pull shovel or back hoe

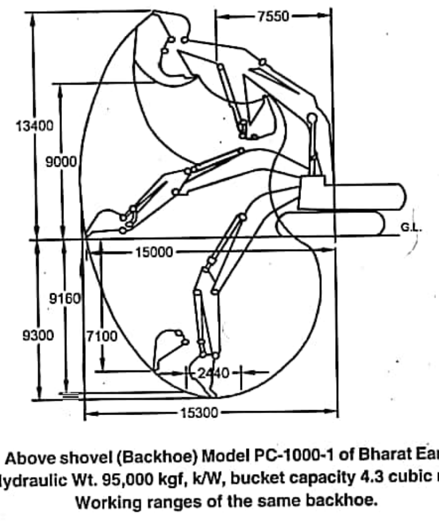


Fig. 5.10 Above shovel (Backhoe) Model PC-1000-1 of Bharat Earth Movers Ltd. Hydraulic Wt. 95,000 kgf, k/W, bucket capacity 4.3 cubic metres. Working ranges of the same backhoe.

Stripper Shovel

A stripper shovel is only a modification of dipper shovel with a long boom and is used for casting fragemented rock or earth into a dump of overburden. It is mostly deployed for overburden.

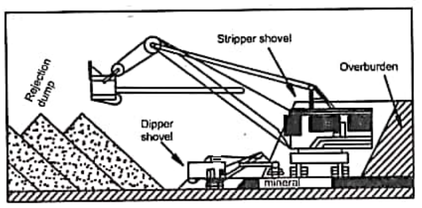


Fig. 5.10 Quarry working with stripper and dipper shovels

Pull Shovel or Hoe

Fig. 5.9 shows a pull shovel, which is also known as a hoe, back hoe, drag shovel. It is used for loading dumpers and its best application is for digging below the level on which it stands. The shovel and the dumpers can stand at a higher level free from water and mud of the quarry floor. As the attachments to the bucket are by dipper stick and not by cables, the bucket is under positive control of the operator and therefore suitable for hard digging.

The shovel is used for stripping top soil, and making shallow cuts and trenches upto a depth of 3.5 to 6 m. Compared to dipper shovel, the hoe is slower in digging and less efficient for loading trunks.

In deep digging, the face should be kept fairly straight and the shovel should be as far back from the edge as possible, otherwise there may be danger of caving of the edge.

Dumpers or Tippers

These are heavy duty trucks with a container-body of steel open at the top for receiving material loaded mechanically by tractor shovel, dipper shovel, dragline, *etc.* All dumpers/tippers are provided with arragements to lift the loaded body by utilising hydraulic pressure to force a ram out. The body swings from its horizontal position round a fulcrum through nearly 70° to dump its load and the hydraulic system also functions to pull the body back on its seat *i.e.* the chassis. Fig. 5.11 shows a typical hydraulic system layout for the tipping gear of a dumper. From an oil tank oil flows by gravity to hydraulic pump. When the driver engages the power take off (P.T.O.) control lever, power from the engine is transmitted from the transmission countershaft to the power take off which drives the pump. The oil under high pressure from the pump goes to the control valve whose lever can be manipulated for 4 different possition.

(a) ***Raise position :*** High pressure oil goes through the hose pipes to the bottom of the hoist cylinder and the ram is then forced out. Oil at the top of the hoist travel back to the control valve through the hose connected to the piston rod.

(b) ***Hold position :*** Both the passages between the control valve and hoist are closed so that oil at the bottom and top of the hoist is at a stand still and the latter is unable to move in either direction.

(c) ***Float position :*** Both hose passages between the control valve and hoist are open so that oil at either end of the hoist can flow either way. The hoist can then travel in either direction depending upon the direction in which the force is applied.

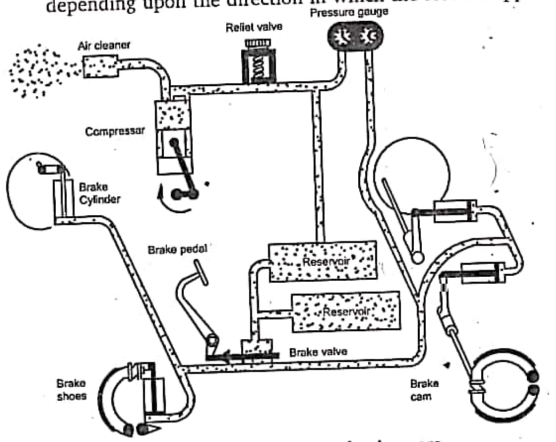


Fig. 5.11 The tipping gear for dumpers

(d) *Lower position :* High pressure oil goes to the top of the hoist which then telescopes itself by the oil pressure and the oil at the bottom of the hoist travels back to the tank via the control valve. The body is thereby lowered on to the chassis.

Steering on all the heavy duty dumpers is mechanical but assisted by hydraulic power, generated by the engine. The dumper operator's exertion in thereby considerably reduced. Mechanical transmission from the engine to the rear wheels is the standard practice now-a-days, though for some years the rear wheels were driven by individual electric motors controlled from operator's cabin. Medium sized mechanised quarries employ dumpers of 25-50 te carrying capacity. 50-60 te coal haulers are on the manufacturing line of B.E.M.L. and Hindusthan Motors. Future planning of large projects is for employment of 100-150 te dumpers which will be fed by shovel of 8-10 m³ capacity. Bottom discharging coal haulers of 55 te payload 43 m³ struck capacity (model GB 60C) are manufactured by BEML.

Brakes on dumpers are operated by compressed air (Fig. 5.12). Some dumpers are equipped with *hydraulic retarder* (hydrotarder). This is a device used on some trucks and dumpers to prevent the speed from exceeding certain limits when travelling a steep down-slope and also to produce a breaking action on the vehicle. In a way, it acts as a governor. It uses the hydraulic friction to produce the breaking action. Unline the regular brakes, the hydrotarder will not completely stop the vehicle but will slow it down preparatory to stoppings with the familiar friction brakes, operated by compressed air or hydraulic pressure. The retarder essentially consists of a vane type rotor turned by the driven shaft, a fixed casing or stator fitted with vanes and an oil circulation system.

The machines deployed in the opencast mines, at the crushing and ore preparation plant have to be of matching capacities. At **Kudremukh** Iron Ore Project, one of the largest opencast mines in India, the capacities of some of the machines are : shovels 10.7 m³, production trucks 108 te, front end loaders 10 m³ , electric drills for 310 mm dia. blast holes, gyratory crushers 4000 te/hr.

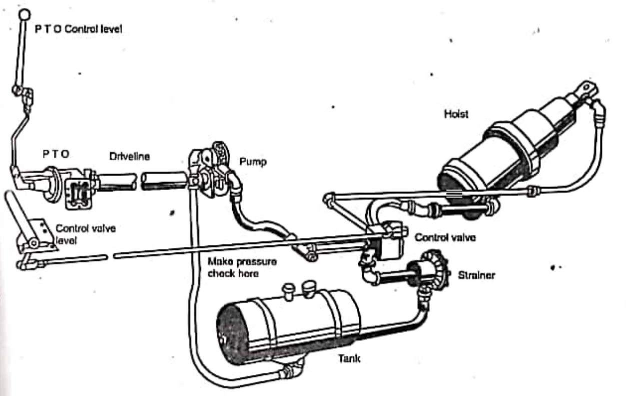


Fig. 5.12 Schematic arrangement of compressed air brake

Drage Line

A drag line is a machine used for excavating earth, sand or soft rock and consists essentially of a revolving deck, a long light boom, crawler chains, and a special type of bucket held in position and controlled by cables (Fig. 5.13). The bucket, when it has to be loaded, is lowered in the earth or loose rock by manipulation of the cables and is dragged by them. As it is dragged it gets loaded. Hence the name **dragline.** A dragline is operated by diesel engine or a motor which is supplied power at high voltage from external source through a trailing cable. The depth to which a dragline digs is limied by the capacity of the drums to hold the hoist cable. When digging, the bucket, after it is loaded, is hoisted up, the boom given a swing through 90° and the contents then unloaded by manipulation of the cable.

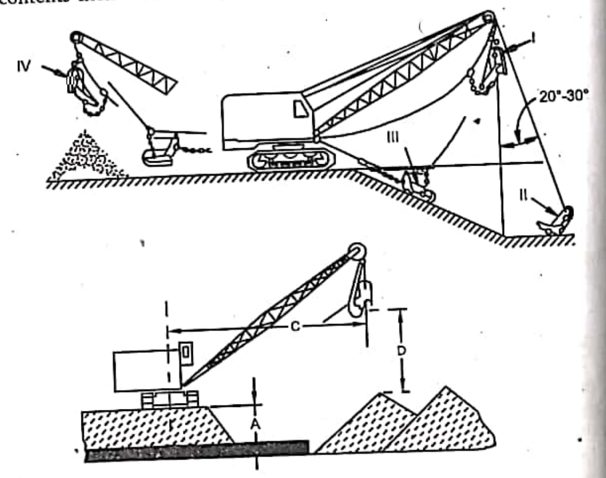


Fig. 5.13 Working ranges of a dragline

A – Digging depth C – Dumping and digging radii D – Dumping height

(Above) A dragline showing the bucket in various stages. I-Bucket empty, II-Bucket starts digging, III-Bucket being dragged and getting loaded, IV-After a swing of the boom bucket unloaded on a spoil heap.

A dragline is suitable for digging alluvium, sandy soil, unconsolidated rock or blasted coal/rock. It digs below the level, at which it stands and from position can dig over a wide working place and cast the earth over a wide area within the reach of the boom. It is generally not employed to load dumpers as the accurate positioning of the dragline bucket over a limited area of the dumper delays the cycle of operation and the common application is for dumping overburden. It is suitable for working a quarry with watery conditions as the dragline works from a higher and, therefore, dry position.

Loading capacity of a dragline

A dragline is capable of dealing with the following quantites of rock/earth (solid) in a year (12 to 14 hours work daily).

Bucket size	Million m^3
4.5 to 7.5 m^3 (6 to 10 cyd)	0.25 to 0.75
11.5 to 15 m^3 (15 to 20 cyd)	1.5 to 1.7
23 to 30 m^3 (30 to 40 cyd)	3 to 4.5

A drag line may be crawler mounted or of waling type. Crawler mounted machines have travelling speeds of 0.25 to 5 km/hr. A walking dragline has a travelling speed of 0.18 to 0.6 km/hr.

Road Grader

This is a machine for levelling the road surface by smoothening out the ups and downs and for casting aside the boulders on the road. It is always pneumatic tyre-mounted with only rearwheel drive and the front wheels are small. The grading blade is attached to a circle that is hung from the overhead frame and pulled by a drawbar fastened to the front of the frame. The blade is usually 3.5 to 4 long having replaceble edges on the sides and bottom. Steering is direct-connecting mechanical by a hand wheel though a hydraulic booster is fitted on some models (Fig. 5.14).

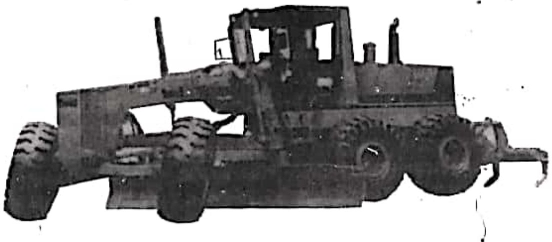


Fig. 5.14 Road grader

The motor grader (Mode GD 605 R-2) of B.E.M.L. has the following main specifications : Engine flywheel HP 145 at 1800 RPM; operating weight 12,650 kg; Max. Drawbar pull 7,280 kg; Max. speed forward - 43.6 kmph; steering - full hydraulic; overall length-8415 mm; width 2375 mm; height-3200 mm, minimum turning radius 10.4 m.

Rock Drills

Rock drill is the term applied to all machines using compressed air for drilling holes into rock by combined percussive and rotary action. The hole diameter is normally upto 100 mm.

The rock drills are classified mainly as follows :

1. Jack hammers (also called *Sinkers*)
2. Drifters.
3. Stopers.
4. Wagon drills

A jack hammer, so familiar to mine workers, is a hand-held and unmounted drill used for vertically downward drilling. It weighs from 15 to 25 kgf and is used for drilling upto a depth of 2 m (rarely 3 m); hole dia. is generally 30 to 37 mm and rarely 50 mm. In a few cases a jack hammer may be mounted on an air leg. Though ordinary used for dry drilling, it can be adapted for wet drilling as well.

A drifter is a mounted drill, generally designed for horizontal drilling. It is heavier than the Jack hammer and is used in quarries and for tunnel driving. The widely used mounting is the column and arm and the drill may be used for wet as well as dry drilling. Its working is like a jack hammer.

A stoper is a drill for drilling upward and derives its name from its widespread use in mine stopes. It is used normally for wet drilling.

A wagon drill is essentially a drifter type drill capable of movement up and down a vertical guide and mounted on a portable frame fitted with wheels. The hole dia. is from 50 to 100 mm and the depth drilled ranges from 3 to 15 m.

Compressed air was the motive power for wagon drills till recently but now-a-days some wagon drills are operated by hydraulic power, as hydraulic power is more efficient than compressed air power.

Jack Hammer Drill

It is a compressed air operated drill to which air is supplied from external compressors through hose pipes at a pressure of about 6 kgf/cm^2. The drill weighs 15 to 25 kgf and drills holes of dia. 30 mm to 38 mm (rarely upto 50 mm) upto 3 m depth. The drill rod is hexagonal in cross-section, suitably shaped at one end to form the shank and the other end is so shaped as to form a non-detachable single chisel bit with a tungsten carbide insert. Drill rods may also be equipped with detachable X type tungsten carbide drill bits. In a shift of 8 hrs, two workers who hold the drill can drill 60 holes, each 1.2 to 1.5 m deep in sand stone, laterite, etc. When hand-held, the machine drills vertically downward holes only but if mounted on air legs, it may be used for drilling inclined holes. An oil bottle placed between the drill and the air receiver, and connected by hose pipes to both, provides lubrication to the drill when working. For dust suppression a jack hammer can be adapted to wet drilling by some modifications so that the drill cuttings mixed with water come out of the hole in the form of a slugde. The air consumption is generally 2-2.5 m^3 of free air/min. (See vol 3 for more description).

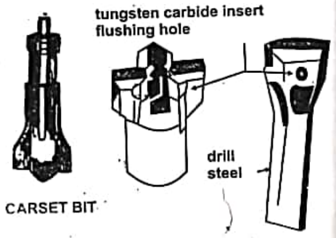


Fig. 5.15 Common Types of drill bits

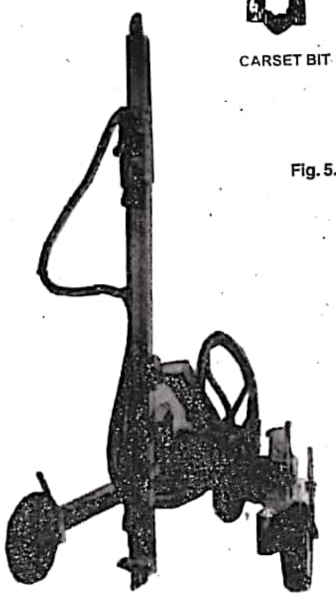


Fig. 5.16 Wagon drill Model BVB 25 of Atlas Copco suitable for drilling holes of dia. 50 to 115 mm. It may be fitted with a heavy duty rock drill (drifter) or with a down-the-hole-hammer.

Wagon Drill

A compressed air operated drifter mounted on a mobile frame and capable of travel up and down a mast is known as wagon drill. The frame is usually tyred wheel mounted though crawler chain mounting is provided in a few models. Tyred wheel mounted wagon drills can be pulled by the operator and his helper to the hole sites on a level ground. A wagon drill, as stated earlier, is used to drill holes of dia. varying from 50 mm to 100 mm for depth of 3 m to 15 m. The mast for the drifter is usually 3 m long providing for nearly 3 m vertical travel of the latter. This travel is possible with the help of a compressed. air driven feed motor through chain (known as chain feed). The drifer provides the rotary motion as well as the percussive action to the drill rods, and in turn, to the drill bit. The drill bit is detachable X type with tungsten carbide insert. Compressed air fed through the hollow drill rods blows away the cuttings to the surface. Total meterage drilled in an 8-hours shift is 60-70 m in rocks like sandstone, coal, etc. including the time spent on shifting the drill from hole to hole. The mast is capable of swivelling from vertical to a horizontal position and it can be kept fixed at any angle between the horizontal and the vertical, thereby facilitating vertical, horizontal or inclined drilling upto 40°. The drill is not self propelling, and receives air from external compressor (Fig. 5.16).

The maximum air consumption is 8 to 19 m³/s min. of free air at 6 kgf/cm² including air blowing for drill bit of 60-70 mm dia.

Though a detachable X-bit is the drill bit on most of the wagon drills, some wagon drills used for 100 m dia. holes used down the hole hammers. Such down-the-hole hammers are used for larger dia. holes also and are described a little later.

Down-The-Hole Hammers

In a large size wagon drill using a drifter a considerable portion of the drifter's energy is utilised in overcoming the inertia of the drill string making up the column of the drill rods and in rotating them. Such loss of the drill energy increases with depth. This waste of energy is considerably reduced by the use of the down-the-hole hammer. The drill bit used may be a carset bit (a X-bit with little modification) or a button bit which is fitted in the hammer. The compressed air going down the hollow drill rods forces the piston which directly hammers the drill bit without any drill rod in-between. The number of blows is from 500 to 2400 per min. When using down-the hole hammer the drifter is replaced by a rotary head placed at the top of the drill string and driven by a built-in piston type air motor. The rotational speed of the drill rods is neary 15-25 r.p.m. The rotary head is also used to tighten and loosen threaded joints on rods. The up and down travel of the drillrods is by a chain feed. Fig. 5.17 shows down-the-hole hammer, type 100 ASS used on HALCO drills for holes of 100 mm to 125 mm dia. Its specifications: Outside dia. 89 mm, length without bit 94 cms; weight without bit 31 kg.

Air consumption at 7 kgf/cm² is 5.5 m³/min.

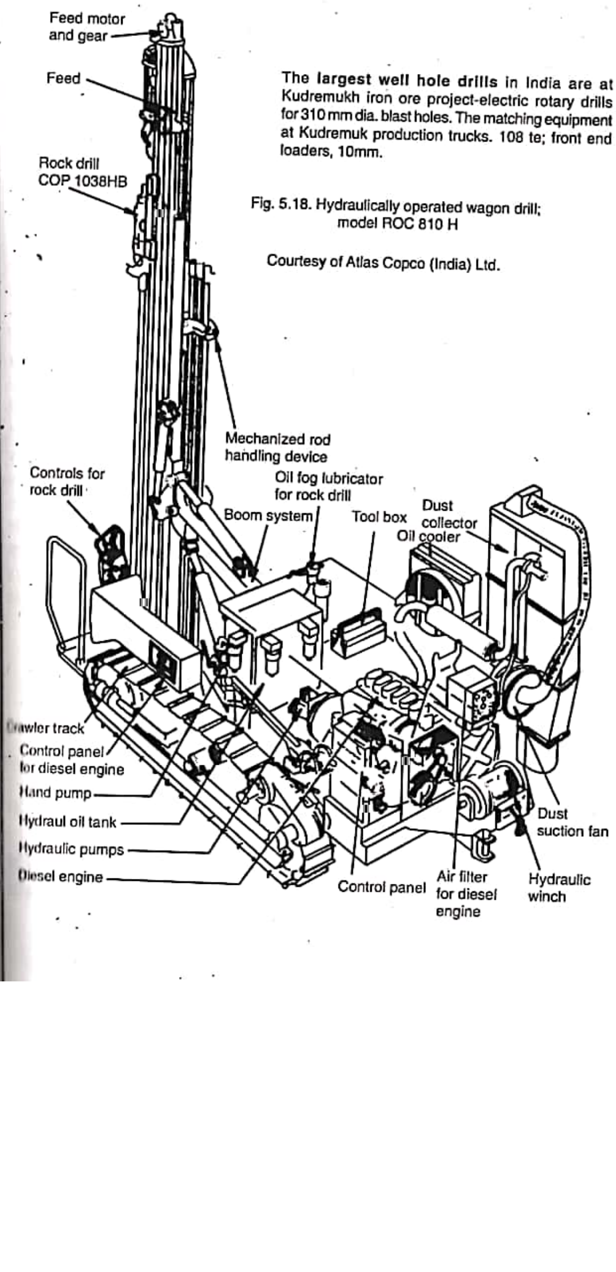
Hydraulic Wagon Drill

Some of the heavy duty wagon drills are powered by hydraulic pressure system. Fig. 5.17 shows a hydraulic wagon drill, model ROC 810 H manufactured by Atlas Copco (India) Ltd. It is equiped with a rock drill model COP 1308 HB manufactured by the same company. In the drill, compressed air is replaced by hydraulic pressure and the prime mover for the hydraulic power pack is an air cooled diesel engine. The absence of exhaust air results in a much lower noise level when compared with air-powered rockdrill. It can drill holes of dia. 65 mm to 127 mm and can therefore be used as a well hole drill for 127 mm dia. holes for depth upto about 12 m. The hole is flushed with compressed air at 10 kgf/cm². The rate of penetration in hard rock is generally 1 m/min using 90 mm dia bit. The rock drill 1038 HB is equiped with a hydraulic system incorporates indicators rock condition. The hydraulic system incorporates indicators which point out any fault or malfunction in the system. The boom system is operated by hydraulic pressure.

Automatic disc brakes contribute towards increased safety for the operation when travelling along steep inclines.

Well Hole Drill

This is usually a crawler mounted drill operated by a diesel engine or by an electric motor which is supplied power from an external source through a trailing cable. It drills holes of 125 mm to 300 mm diameter, depth varying from 6 m to 18 m. It has a long mast, 3 m to 6 m, to accommodate the length of the drill rod. The mast is collapsible and the drill should not be moved over an appreciable distance with



The largest well hole drills in India are at Kudremukh iron ore project-electric rotary drills for 310 mm dia. blast holes. The matching equipment at Kudremuk production trucks. 108 te; front end loaders, 10mm.

Fig. 5.18. Hydraulically operated wagon drill; model ROC 810 H

Courtesy of Atlas Copco (India) Ltd.

the mast raised. The drills are of percussive as well as rotary type but the latter is common in coalmining areas. The drilling tool of rotary drill is a tricone bit on most of the drills but on the machines which are known as "down-the-hole percussive drills" (sometimes called "down-the-hole hammer drill"), the drilling tool is a cross bit (carset bit), or a button bit. In down-the-hole hammer drill the assembly of the drill and its short length pipe is called *down-the-hole hammer* (see Fig. F.18).

In the rotary drill the string is rotated by the prime mover through suitable gearing. The tricone bit attached at the end of the drill string is thus rotated and it is kept pressed against the rock by hydraulic or pneumatic pressure.

In down-the-hole precussive drill the rotation of the hollow drill rods is provided by a rotary placed at the top of the drill string and driven by a built-in air motor. The air motor is also used to tighten and loosen threaded joints on rods and bits. The up or down travel of the drill rods is by a chain, operated by a reversible piston type air motor (Chain Feed). A compressor mounted on a well hole drill helps to clean the hole as it is drilled. During drilling, the meachine is levelled with the help of 3 hydraulic jacks. Normally twenty holes, each 9 m deep, can be drilled in one shift in sand stone, shale and coal. Only vertically downward drilling is possible on most models though holes 20° off vertical can be drilled by a few machines. On some machines the drill-rods and the tools are at one end and on others, in the middle of the machine. The latter arrangement is permissible where the burden of blast hole is large and the ground at the quarry edge strong enough to support the weight of the machine; but where this is not practicable drills rigs with the drill rods and tool at one end have to be used.

Fig. 5.18 Down-the-hole-hammer

A well hole drill appears like a wagon drill suitable for larg diam. holes, *e.g.* the wagon drill shown in Fig. 5.17.

A rotary well hole drill can drill in a shift of 8 hours nearly 2 holes, 200 mm dia. each 9 m deep, in sandstone, coal, shale and simil rocks.

Inclined Drilling

Where the overburden consists of soft rock which can be conveniently removed by ripper and scraper-dozer combination an alternative to ripper and scraper-dozer combination an alternative to ripper is the method of drilling nearly horizontal blast holes and blasting them. Vertical (or nearly-vertical) blast holes have to be drilled where the overburden consists of hard rock like sandstone, laterite, etc.

30° off vertical may be considered to be the limit for inclined drilling of nearly-vertical holes on a bench. Larger angle increases the length of the hole, difficulties in charging it with explosives of fixed shaped cartridges, proportion of stemmed section of the hole and gives face inclination unsuitable for travel of the shovel bucket.

The toe of a bench can be removed by extra drilling of short length horizontal holes only in the toe and blasting them, or by resort to inclined drilling of the main (nearly vertical) blast holes. In vertical as well as inclined blast holes for the face, it is always essential to extend the hole slightly beyond the level of bench floor to secure proper fragmentation of toe if the hole is terminating in hard rock.

Layout of a mechanised quarry

The layout of a quarry depends primarily on

1. shape, size and dip of the deposit.
2. proposed depth upto which mining activitiy is planned to extend.
3. thickness of the overburden,
4. surface topography,
5. desired production,
6. transport system for mineral and OB,
7. arrangement for disposal of debris,
8. type of mechanisation and finance available for it.

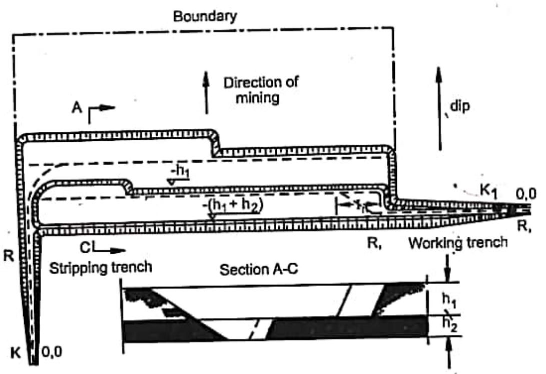


Fig. 5.19 Layout of a mechanised quarry with separate trenches

The layout of a quarry should depict the following :

1. Position of OB benches and mineral benches.
2. Access to the benches and exit roads for the dumpers.
3. Position of back filling area.
4. Location of machinery which operates from one site over a long period such a sovel, dragline, bucket wheel excavator, in relation to the benches.

Fig. 5.19 shows the layout of a mechanised quarry using dipper shovels, dumpers and well hole drills, the common equipment in most of our mines. The property is divided into areas along the strike, each area being nearly 100 to 300 m long. In the initial stages a box cut (trench) RK is made with the help of scraper, dozers or small capacity (2 m^3) shovels to suit the depth of the trench. Such trench is essential for a shovel which stands on the floor of the quarry and operates on benches. Some amount of blasting may be necessary in making a trench in hard rocks. In the advanced stage of the quarry two trenches RK and R_1K_1 are in use, one for the OB and the other for the mineral which goes to the crushing plant. The OB bench and the mineral bench are usually parallel to each other and advance towards the quarriable limit of the mine.

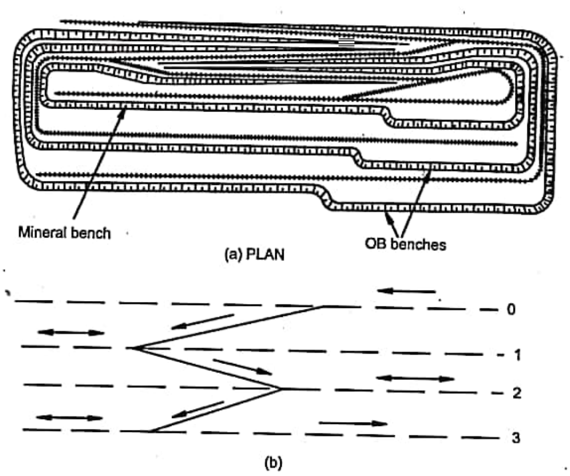
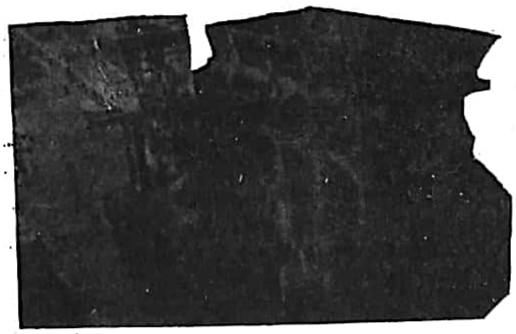


Fig. 5.20 Switch back layout (Vertical section exaggerated)

In the figure, h_1 and h_2 are OB bench and mineral bench respectively. If the OB and mineral are of considerable thickness, there may be two or more benches for OB as well as for the mineral. In such cases ramps have to be provided between adjacent benches. Provision of two trenches, one for OB and the other for mineral, avoid congestion of dumpers on the haul roads. This layout can be modified to have a crusher and outgoing belt conveyor on the lower end of mineral trench. In-pit crusher with feeder breaker is the trend in open cast mines these days. Mineral is then transported by conveyor belts to coal handling plant or mineral processing plant. Some of the mechanised opencast mines in South Eastern Coalfields Ltd., are now having in-pit crushers.



A mobile crusher in a mechanised quarry receiving finishing touches in the manufacturing shop
Courtesy of Simplex Engg. & Foundry Works ltd. Bhilai

Such layout of two trenches is not suitable if the mineral is at much depth as formation of trenches would involve removal of large volume of earth and rock. Layout of one trench for both mineral and OB may then be adopted.

For deeper quarries switchback system of track layout is advantageous. Fig. 5.10 b shows the tracks as they would appear in a vertical projection (exaggerated) for a locomotive-minecar combination of transport system. Numbers 0,1,2,3, mark the levels of the floors. Thus to get from level 0 to working floor 3 a train must follow the route marked by the arrows. Switchback entails a big loss of time for shunting operations. Each switchback should be sufficiently long to hold a train and railway switches, or else to allow enough room for minecars to turn round.

Fig. 5.21 shows a modified layout depicting the position of various coal and OB benches in a quarry (shovel-dumper combination) at an advanced stage if OB is backfilled in the decoaled area. The seam is considered to be 18 m thick dipping at 1 in 5.

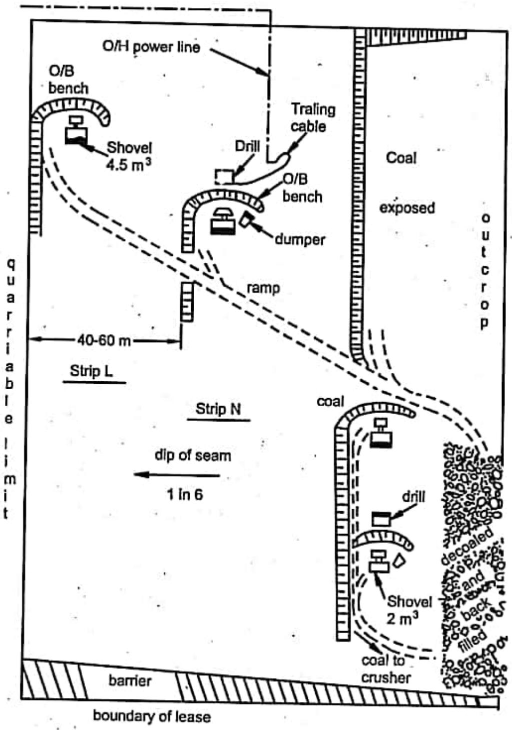


Fig. 5.21 Quarry at an advanced stage with dumper-dipper shovel combination

The benches formed are sometimes designated as Bench No. 1, Bench No. 2, etc. or by the reduced level of the floor of the bench e.g. 500 metre bench, 510 metre bench, 520 metre bench, etc.

Pumps are installed in the quarry which has to be kept nearly dry for operation of shovel and dumpers. The usual arrangement is to have two or three pumps as a semi permanent installation fed by small portable pumps.

The area may be divided into strips of 45 to 60 m width along dip rise. A width of 45 to 60 m permits easy movement of dumpers, the minimum width required being 18 metres. Each strip provides a bench, which is equipped with a shovel and a well hole drill. Daily progress of shovel in the strips L, N, is along dip rise direction, but the advance of the bench is along the strike when considered over a quarter or half year. The quarry excavation of course advances towards the dip to the quarriable limit.

The shovels S_1 and S_3 work on stone benches and the others (2 m^3) work on coal benches. The overburden may be taken to dumping yard away from the quarry, or dumped in the decoaled area. Suitably graded roads, not steeper than 1 in 10, with ramps between the strips, are provided as shown in the figure. Shortage of shovels and ancillary equipment may require the management to work only a few benches at a time and not as many as shown in the figure.

Fig. 5.22 shows a spiral layout for dumper transport. As the haulraod runs along the periphery of the quarry, excavation should commence from the boundary of the quarry and proceed downward forming one bench at a time. Such layout is an advantage in hill-top deposit which may be worked in horizontal slices.

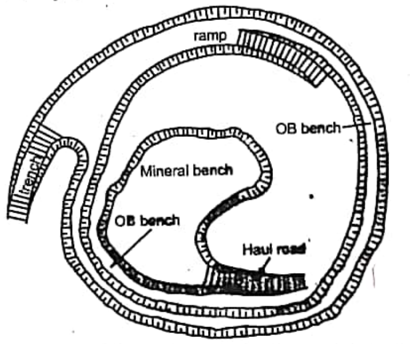


Fig. 5.22 A spiral layout for shovel-dumper combination

LIGNITE MINING AT NEYVELI AND BUCKET WHEEL EXCAVATORS

In India lignite is excavated on a large scale at Neyveli Lignite Project by mechanised opencast mining employing bucket wheel excavators and other heavy duty matching equipment. India is among the four countries after West and East Germany and Austrialia to engage in lignite mining on a large scale and Neyveli mines are the largest lignite mines in Asia worked by Neyveli Lignite Corporation.

The Neyveli Lignite Project is located at Neyveli in South Arcot Distric of Tamil Nadu. Situated in far South, having no coal mines, the project fulfils to a great extent, the requirements of solid mineral fuel in that region and has, therefore, a special significance. Lignite based two pit-head thermal power stations, one of 600 MW and the other of 1470 MW capacity contribute nearly 40% of electricity consumption of Tamilnadu. Lignite is a low calorific fossil fuel with a gross calorific value of 2600 to 3000 kcl/kg (as received samples) with the following proximate analysis :

Moisture 45-60%; ash 3-6% (in some cases even less than 3%, V.M. 22-26%; F.C. 20-26% (in a few cases even less than 18%;) gridability index 108 to 127.

The average bulk density (in-situ) is 1.15 gm/cc or 1.15 te/m^3. Total on lignite (in 1995).

1. Opencast mining project, No. 1 Mine, to extract 6.5 million tonnes/year of lignite proposed to be expanded to 8.5 MTe/yr in the second expansion scheme. Ultimate planned capacity 10 Mte/yr.
2. No. 2 Mine Project. It has an installed capacity for producing 10.5 million tonnes/yr lignite. This mine caters to the requirements of thermal power station II which has been expanded from initial installed capacity of 3×210 MW to 8×210 MW.
3. Two thermal power stations, one of 600 MW and the other of 1470 MW capacity, as stated above.
4. Fertiliser plant to manufacture urea; capacity 1,30,000 te/yr.
5. Briquetting and L.T. carbonisation plant producing carbonised briquettes called LECO for domestic use. Capacity 2,62,000 te/yr. Certain carbochemicals are produced as byproducts.
6. Clay washing plant, capacity 6000 te/yr of washed clay.

Reserves of lignite at and around Neyveli spread over 400 km^2 are estimated to be around 3300 million te. Out of these, mineable reserves within the parameters of not less than 8 m lignite thickness and not more than 110 m overburden thickness work to be 1500 million te.

No. 1 mine is spread over an area of about 15 km^2 with deposits of 230 million te of lignite. Average thickness of lignite seam is about 15 m at a depth of 70 m and the lignite bed dips at 1 in 100. The overburden thickness increases from 50 to 100 m at the rate of 1 in 100 towards south-south west direction of the mine field. The lignite thickness ranges from 11 to 25 m. No 1 mine is placed to produce ultimately 10 M te of lignite per year.

The ratio of overburden to lignite is nearly 5:1 and quite favourable for extraction by bucket wheel excavators which remove, without blasting, the overburden as it consists mostly of soft rocks. Lignite being soft, needs no blasting and it also is excavated by bucket wheel excavators.

The project is situated near the sea and control of artesian water under high pressure below lignite is a serious problem. The artesian water exerts an upward thrust of nearly 6-8 kgf/cm^2 on the base of lignite and there is risk or water bursting through the lignite seam on removal of overburden. Nearly 1,35,000 lit/min of water is pumped out with the help of nearly 50 bore hole pumps, most of them located around the area being worked out, to maintain pressure of water below the lignite at a safe level. Water table is kept below the lignite floor by such peripheral bore hole pumping and 15-20 te of water has to be pumped out for each te of lignite mined.

The **second mine** is planned for an initial capacity of 4.7 million te/yr of lignite (1st phase) and the final capacity in 2nd phase will be 10.5 million te/yr. This second mine will extract lignite from 26 km^2 area south of the first mine and the deposits of extractable lignite are 390 millions te.

Excavation And Transportation :

Presently (10.5 Mte/year stage) the overburden is removed in 4 benches (Surface, Top, Middle & Bottom). The Top two benches (Surface and Top) are equipped with two 1400 litres bucket wheel excavators (BWE), a system of 2400 mm width conveyors and a spreader of 20,000 TPH capacity each. The third bench (Middle bench) comprises one 1400 litres BWE, a system of 2000 mm width conveyors

and a spreader of 11,000 TPH capacity. The fourth bench (Bottom Bench) is equipped with two 700 litres BWEs, a system of 2000 mm width conveyors and a spreader of 11,000 TPH capacity. The lignite bench has three 700 litres BWEs and a system of 2000 mm width conveyors. The excavated lignite is transported to a storage bunker on the surface of the mine. From there, lignite is reclaimed to the Thermal Power Station II by two Reclaimers of 2000 TPH capacity each and a system of 1800 mm width belt conveyors.

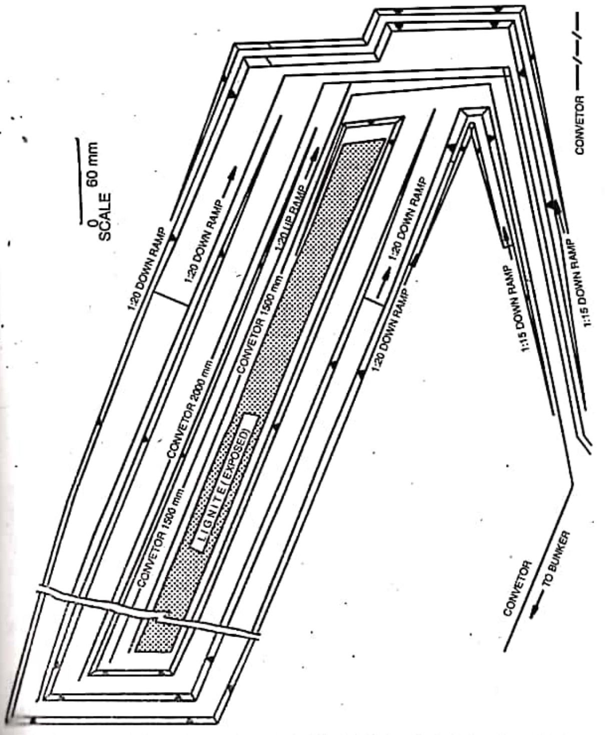


Fig. 5.23 Plan of second mine cut at Neyveli Lignite Corporation Ltd.

Dewatering Arrangements :

The principle of controlling the ground water is depressurisation by dewatering with large scale continuous pumping from a series of pump wells situated at hydrological calculated distances around the mine. The pumpage of ground water is presently at the rate of about 150 Cu. m/minute from about 35 to 40 pump well in the Second Mine. By the large scale pumping operation all that is achieved is keeping the pressure under control locally below lignite.

Storm Water Control : Neyveli, being close to the eastern coastal area and being in the cyclonic belt, has a normal annual rainfall of 1200 mm to 2000 mm. The rain water during the monsoon months (October to December) posses many problems in the operation of the mine. This storm water is coursed to the deepest point of the mine by a pattern of toe and cross drains, where pumps mounted on pontoons are deployed in these sumps and the pontoon pumps are shifting according to the configuration of the lignite floor. Storm water from the sumps can be pumped directly to the surface with high head pumps mounted on pontoons.

Drilling and Blasting : The overburden in the Northern half of the mine consists of hard Cuddalore Sandstone which needs blasting before excavation. Presently 30% of the total overburden is to be blasted in the Surface and Top benches.

Communication : Due to enormous size and multifarious activities special communication facilities are provided through wireless between various machines, conveyors, drive head/train end stations and service yards to the Control Room for smooth co-ordination of all activities in the mines. Separate frequencies/channels have been provided so that there exists no interference in communications amongst various machines and yards. Paging system is also planned to be introduced for quicker contacts.

Exploration work is going on for the **third mine** of proposed **10 million te/yr** capacity. The third mine, area 25 km2, will be located to south of the second mine.

Bucket Wheel Excavator

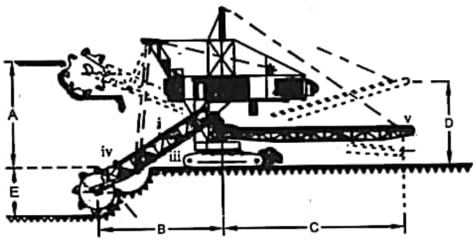


Fig. 5.24 General arrangement of a bucket wheel excavator

(I) Boom which can be raised and lowered (ii) Superstructure with machinery houses, ballast and winches (iii) Under-carriage with crawler travel gear control equipment, cabins for switch gear, workshops and crew (iv) Operation cabin (v) Discharge end.

Working Ranges (Figures) in bracket relate to 700 lit. B.W.E. at Neyveli)

A - Cutting height (20 m) B - Cutting radius C - Dumping radius
D - Dumping height E - Cutting depth (3m)

Neyveli lignite project is the only mining field in India where bucket wheel excavators are used.

A bucket wheel excavator is perhaps the most spectacular of all the excavating machines. It is used for the excavation and removal of soft or unconsolidated overburden, coal and soft ores in benches of upto 90 metres.

The bucket wheel excavator essentially consists of 3 units :

1. The bucket wheel,
2. The connecting belt to the main crawler mounted carriage, and
3. The loading belt which disposes of the excavated material continuously into rail wagons or conveyor belts through hopper cars.

The bucket wheel may be compared with the Persian wheel used in the past on large wells and is a large wheel known as bucket wheel with 6 to 12 buckets. The bucket has a lip with manganes esteel teeth for cutting the rock and these are faced with tungesten carbide for hard rocks. The maximum life of teeth with t.c. inserts is 250 hours in soft sandstone at Neyveli. The wheel is mounted at the end of a boom which can be (a) pushed out or pulled back (b) lowered or raised (c) swung through 90° in a horizontal plane depending upon design. The dumps/min by the buckets vary from 70 in small models to 27 in large models. The hourly capacity of an excavator is equal to the total volume of all buckets x wheel speed per hour. The hourly capacities of small lmodels is only 100 m³, but in large models, it is nearly 4000 m³ (3720 m³ solid for 1400 lit. bucket capacity B.W.E.) hr. The excavators are crawler mounted. Fig. 5.25 shows relative positions of the various units of a bucket wheel excavator.

The excavated rock is transporated by a system of belt conveyors mounted on the machine to hopper cars, belt conveyors, or rail wagons.

Power is supplied to bucket wheel excavators at A.C. voltage upto 25 k V though motors operate at 3300 V and 500 V. At Neyveli the B.W. Excavators are supplied electric power at 11 kV through T.R.S. cables and transformers in the machine step down the voltage to 3.3 kV for some motors and to 400 V for some others.

Methods of working with a bucket wheel excavator

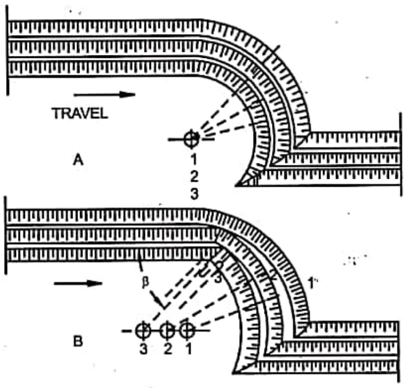
A bucket wheel excavator is employed to operate on

1. full block method, or 2. the lateral block method.

Full Block Method

This is commonly adopted and is shown in Fig. 5.26. The position of the excavator is shown by the numerals 1, 2, 3. Fig. 5.26 A shows the full block method is practised with a bucket wheel excavator having a boom that can be pushed out and pulled back, *i.e.,* capable of thrust forward. Fig. 5.26B shows the full block method as practised with an excavator having a bucket wheel boom that can be lifted and lowered but has no thrust movement.

Where selective wining of minerals is not necessary, full block method is employed so that the bucket wheel cuts in full blocks and in parallel operation with cuts which are made as wide as possible. The blocks removed by giant bucket wheel excavators have a width of 45 m to 100 m. This large width of cut which can be maintained along the whole face, not only reduces the travel way for the crawlers, and consequently, the time required for this displacement, but also the shifting work required for the transport system used to dispose of the mineral. The method is therefore simple and economical.



Full block method.

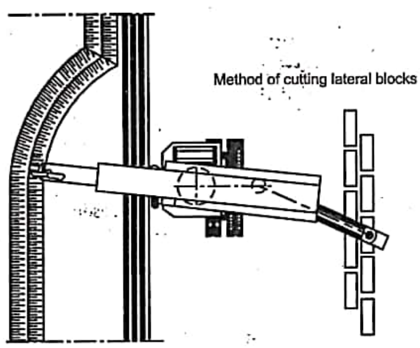


Fig. 5.25

During cutting, the excavator cuts its own ramp of 1:20 into the new block and finally excavates the ramp also.

Lateral Block Method

In order to remove continuously waste partings or mineral seams of minor thickness selectively, the excavator adopts the method of cutting lateral blcoks. Both bucket wheel excavators with thrust or without thrust-can dig the necessary lateral slopes. The width of the block depends, however, always on the length of bucket wheel boom (Fig. 5.26). Bucket wheel excavators with a short bucket-wheel boom are not suitable for cutting lateral blocks. Should the level of the selected seam lie high above the travel-way of the machine, and taking into account the necessary slope angle, a "super" bucket wheel would be required, i.e., with a diameter larger than that which would actually be needed for the output required.

In any case, the amount of crawler travel increases with decreasing width of blocks. The travel-way is reduced if an excavator with thrust is employed.

The excavated rock is transported by a system of belt conveyors mounted on the machine to hopper cars and then to belt conveyors or rail wagons. The loading belt of the machine is also mounted on a boom.

Reclamation

Reclamation is the act of restoring the quarried area to the prequarry state and render it useful for different purposes. In USA and other foreign countries, the mining laws require that the quarried area is restored to the pre-quarry state. As already stated, such law area is restored to the pre-quarry state. As already stated, such law does not exist in our country and the abandoned quarries present unsightly appearances of hillocks of overburden and vast ponds of stagnant water. Only in Neyveli Lignite Complex the quarried area has been reclaimed and turned into pleasant parks, mini-forests, zoos and recreational centres. In fact, with a little foresight and planning, the quarried area can be rendered more useful and valuable than in the original pre-quarry state. Thus vast marshland, after the underlying mineral is excavated, can be turned into a profitable agricultural land.

When starting the quarrying activity in an area, if reclamation is one of the final objects, the top soil is removed by rippers and scrapers and collected at a suitable place where it will not be carried away by wind and will not prove a nuisance to nearby localities and dwellings. In suitable cases such soil can be sold away, thus doing away the need to stack it over a long period of the life of the quarry. Small grass is allowed to grow over it so not raised and carried away by the wind.

After the mineral is excavated and the quarried area backfilled, it is compacted by rollers. Tamping or *sheepsfoot rollers* are the standard tools for compacting fills. They consists of steel drums fitted with projecting "feet" and towed by means of box trames. The portions earmarked for roads and houses are further compacted, but the areas reserved for agriculture, farm land or plantation of trees are covered with 1-1.5 m of soil transported from other places. Such soil is often mixed with appropriate type of manure to suit the agriculture or vegetation planned.

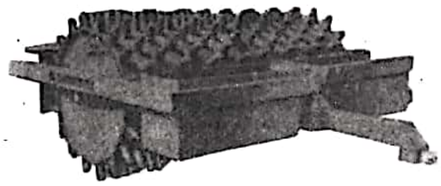


Fig. 5.26 A sheepfoot roller for compaction

Drains are provide all around the reclaimed area. In the areas earmarked for houses, and roads compaction by rollers is carried out for 2 or 3 seasons after the rains with a view to consolidate the back filled debris nearly as compact as in the pre-quarry state.

✧ QUESTIONS ✧

1. What is the scope and what are the limitations of working a seam by opencast mining ? What is quarriable limit ?

2. What is the common type of equipment used in a mechanised quarry? State the purpose for which it is used.

3. Under what conditions will the following equipments be used ?

   (a) *A dipper shovel*

   (b) *A scraper*

   (c) *A dragline*

   (d) *A wagon drill*

4. What is the reasonable performacne of a 2 m³ shovel in (a) an 8 hours shift (b) a week (c) an year.

✧ ✧ ✧

Chapter -6

Access to a Mineral Deposit & Pit-Top, Pit-Bottom Layouts

Access to a mineral deposit is afforded by

1. Haul roads and steps – in the case of an opencast mine
2. Incline and adit – in the case of shallow underground mine.
3. Pit of shaft – in the case of deep underground mine.

Incline And Adit

A mineral bed which is at a shallow depth can be approached by an incline or adit. An incline is a sloping road driven from the surface to the deposit through the alluvium and the rocks overlying the mineral deposit. unless it is for locomotive, an incline is usually steeply dipping at 1 in 4 or 1 in 5. The normal practice is to have an incline along the true dip of the deposit though the shape of the property may sometimes necessitate its drivage along an apparent dip. In the case of steep beds, usually with gradients of 1 in 4 and steeper, a haulage incline is along the apparent dip to have milder inclination of the haulage road. Inclines which touch a coal seam at a vertical depth of 30 m are common in Singareni group of mines. The trends now-a-days is to approach mineral bed upto a depth (vertical) of 200 m by inclines.

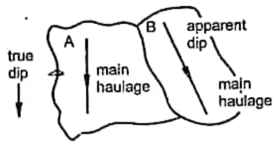


Fig. 6.1 Property A has an incline along the dip but shape of property B dictates incline to be along an apparent dip to have equal area on either side.

If a deposit is at a higher level than the general ground level and outcrops, the access to it is by a level or a slightly rising roadway from the ground level. This type of nearly even roadway is known as *Adit*. Kuraisa, Chirimiri and some other underground mines are approached by adits.

The size of an incline is usually 4.2 m wide × 2 m high if it is for endless haulage or a belt conveyor with rope haulage beside it. Inclines for travelling only may be as small as 2 m × 2 m.

Pit or Shaft

Access to a mineral deposit at depth is by a well, known as *pi* or *shaft*. Deposits at depths of 30 m and more have been entered by shafts in this country. If the strata overlying the vein or seam are soft, and excavation cost is not high, an incline may be driven to enter deposits upto about 30 m depth. Such long inclines at steep gradients of 1 in 4 or 1 in 3 are common at Kolar gold field and Mosabani mines and are known as *inclined shafts*. In Indian coal mining practice, the term *inclined shaft* is not used and the word "shaft" invariably connects a vertical shaft.

Depending upon their purpose shafts are referred to as main shaft or auxiliary shaft. Main shafts are designed for hoisting mineral to the ground surface. Auxiliary shafts are generally for man-winding, ventilation, material transport, stowing or filling material, etc. though the main shaft may, in some cases, serves some of these purposes.

An inclined shaft in metal mining practice is rarely in the ore body proper which normally does not have uniform angle of dip. It is a standard practice to drive an inclined shaft nearly parallel to the dip of the ore body but in the country rock of the footwall as the hangwall collapses after stoping (Fig. 6.3). In Kolar Goldfield, vertical shafts are located at a distance varying between 100-150 m from the lode in the footwall with the intention that they should not be affected by the stresses or strains which develops in the rock when the ore is excavated by stoping. The stresses may affect the vertically of the shaft, and in the extreme case, may result in its collapse.

In horizon mining the coal seams are approached by long level stone drifts starting from centrally situated shafts. (Fig 6.4) for example, at Sudamdih Colliery (BCCL), some mines with steep seams in North Eastern Coalfields Ltd. Horizon mining is a standard practice in many metals mines where the lodes are steep.

Where two or more deposits have to be worked by underground mining methods, one deposit is entered by an inclined or pit from the surface. The same incline or pit may extend to the lower deposit. Sometimes the lower deposit is entered from underground workings by (a) A staple pit (sometimes called secondary shaft), or (b) A cross measure drift.

A *staple pit or blind* (shaft) is a vertical pit form one underground working place or road to another underground working place or road but does not extend up to the surface.

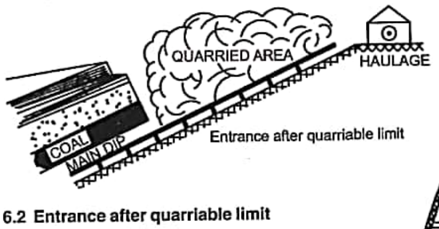


Fig. 6.2 Entrance after quarriable limit

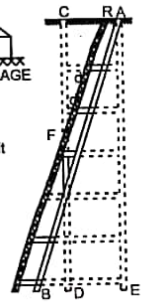


Fig. 6.3 Access to a steep vein

AE and CD - vertical shafts ; AB - Inclined shaft in footwall.
Cross-cuts at intervals connect the shaft with the vein.

A cross measure drift (also called *drift* for brevity) in a coal mine is an underground roadway through stone connecting two or more coal seams. It may be horizontal or inclined. The term *drift* in mineral mining practice has a different meaning, as explained in later chapter. Cross measure drifts in coal mines are usually of moderate length but in metal mines, some cross measure drifts are quite long. The noteworthy example is Bullen's incline at Kolar Goldfield, serving the lower levels of Orgaum section of Champion Reef Mine. It is 1520 m long and extends from 40th to 88th level. It is equipped for raising men and large tonnage of ore at high speed.

It is desirable to plan sinking of 2 shafts at least upto the first level in a metalliferous mine or to first horizon in coal mining whereby production can start from one shaft while the other one is being deepened.

Shaft Vs. Incline

Incline drivage is cheaper and quicker than pit sinking as it does not need skilled men, costly headgear and heavy capital cost. Men can travel by the incline to surface; not so in case of a breakdown of winding arrangements for a pit. If the mine is stopped for one or two shifts in a day, or for some short period, say a couple of weeks or months, and only pump drivers or other few essential men have to go underground for inspection or maintenance, the winding arrangement has to be kept in readiness for all the time and the banksman, the onsetter and the winding engineman have to be on duty. In case of steam winders the boiler plant has also to be kept working. An incline in such case has obvious advantages.

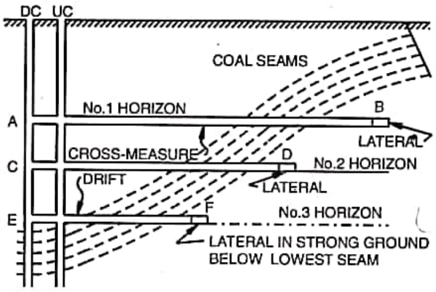


Fig. 6.4 Approach by shafts in horizon mining (coal)

Inclines are safer than shafts; maintenance and repairs of inclines are easy and cheap.

Mining regulations are stringent on inspection of shafts, the equipment used for winding and other winding gear. Winding equipment requires a number of safety devices some of which have to be imported.

For increasing the production or for any other reason if it is necessary to increase the width or height of an incline, it is not a serious problem. But increasing the size of a shaft often dislocates raising arrangements over a long period.

An incline is ideal for trackless mining. By installation of conveyors from working face upto the surface, filling of mineral into tubs, mine care or skips can be avoided and degradation of mineral reduced.

In inclines it is possible to take locomotives from surface to underground working face in a nearly flat seam *e.g.* at West Bokaro Colliery.

An incline has the disadvantage of heavy make of water during rains due to large catchment area if the deposit is worked by quarry on the outcrop side.

The choice of site of an incline is limited to the outcrop region, while a shaft may be located at any convenient place in the property.

Under CMR 66 and MMR 68 every mine should have at least 2 shafts or inclines before any underground production work is undertaken.

Selecting Site Of An Incline Or Pit

A number of factors have to be considered before selecting the site of a shaft or incline. In mountaineous regions, inclines and shafts should be always from possible path of land slides. The site should be above the highest flood level of a river or lake in the area, and the slope of the ground away from the shaft to facilitate drainage and movement of mineral-loaded tubs to the crusher and tippler. Usually the crusher and tippler are located at a higher level for flow of mineral by gravity. A shaft or incline should be situated close to efficient transport facilities, like railway, road or canal to avoid the need for construction of long railway line as also curves and heavy cutting or filling for the line. At the same time surface transport of mineral from incline or pit to processing plant should be minimum. If the railway has to cross a big river like the Damodar, the cost involved in bridge construction may rule out the possibility of constructing a siding. Installation of an aerial ropeway for transport of mineral provides an alternative *e.g.* at Sasti Colliery (Wardha Valley til 1982). Broken ground like goalf area and stoped area and a site near geological disturbances like faults, dykes and sills should be avoided so that un-productive work of long stone drift through dyke of fault should not be necessary soon after touching the mineral. A plentiful supply of water, not far from the shaft or incline, is essential for bilers, pithead baths, washeries, etc. Costly land should be avoided as a large number of buildings and other installations require an extensive area. The space near the shaft should be adequate for dumping the debris during sinking and for the service buildings like office, lamp cabin, pithead bath, winding engine rooms, creche, canteen, screening plant, repair shops, *etc.* In the case of metal mining if ore preparation mill or plant is to be located near the shaft, mill tailings occupy a large surface. The site should be level or nearly so over a wide stretch.

In some mines large dumping space near the shaft is required to dump underground stone or shale bands or to dump washery rejects if a washery is planned. Sufficient space should be earmarked for further expansion programme.

For a coal seam of moderate inclination the shaft may be in the middle (dip-rise direction), but where the seam has more inclination, it is convenient to so place the shaft as to have nearly one-third area on the dip side and two-third on the rise. This facilitates underground transport and drainage of rise areas towards the shaft. Shafts for sand stowing pipes are however on the rise side of the property. Ventilation shafts used mainly for upcast air where exhaust fans are used are also the rise side.

If the surface of the mine leasehold is not hilly and the land is nearly flat or uniformly sloping, it is convenient to have production shafts in the middle of the mine area as transport of the mineral on the surface can be arranged by locomotives, belt conveyors or by rope haulage to coal handling plant or mineral bunkers. Maintenance and supervision of surface transport arrangements is more efficient than underground.

As stated earlier, an incline has to be situated near the outcrop and its site does not offer as much choice as a pit. The section of shaft site is, however, a major decision. Shaft sinking being a costly process, nature of strata through which shaft has to be sunk should be ascertained by a proving bore hole near the proposed shaft site. Trial pits 3 m to 5 deep, are sunk to test bearing pressure of the soil for structures associated with winding and other heavy installations.

The incline in a coal mine should have practically equal area upto a maximum of 600 m on the strike, on either side, for development by rope haulage. This limit may be considered at 1000 m for belt conveyors. Larger area on the strike requires more inclines. The shaft should have nearly equal width on either side along the strike for development.

Shape, Size And Number Of Shafts

As stated in chapter 4 shafts are circular in shape; rectangular shafts are rare in this country, the exceptional cases being some of the shafts in metal mines. The finished diameter of a shaft varies from 4.2 m to 6.7 m.

Where a shaft accommodates a pair of single cages (a cage capable of a accommodating a single tube only), the diameter may be nearly 4.2 m; where a pair of tandem changes (a cage capable of accommodating two tubs) is used, the finished diameter may be 5 m to 6 m. At Chinakuri colliery employing two single deck cages, each cage accommodating one 31/2 te mine car, the coal raising shaft is 6.0 m finished diameter for a planned production of 50,000 te per month. At Sudamdih colliery, the main coal winding shaft has 7.2 m finished diameter. The main shaft for mine entry at Jaduguda mine is circular, 5 m finished diameter, fully lined. It is equipped with 2 multirope friction winders, one winder for cages, and the other for a skip having a counterbalacing weight. Each cage has 2 decks and is for a payload of 3 1/2 te or 50 men. The skin has a capacity of 5 te payload. The cages run on rigid guides whereas the skip and its balancing counterweights run on rope guides.

Under the mining Regulations each seam should have a least two outlets to surface separated by a minimum of 13.5 m at any point. The general arrangement is to sink two shafts close to each other, separated by 30 to 60 m, for facility of a quicker intercommunication. Unless the two shafts are connected by underground tunnel no development work inside the mine is permitted under the law. One of the shafts serves for mineral winding, and the other for ventilation, man riding and material transport. The upcast shaft, having air locks, may be used for small quantity of raising, which may be from a separate seam, or form a different level. This helps in grade control.

The nearness of the two shafts permits common boiler plant, pit-tip arrangement, screening plant, crusher plant and siding. The pit-bottom arrangements also may be so laid out that loads of one shaft may be directed to the other, if desired. An extensive property, say 2.5 km^2 or more may have two or more pairs of shafts, situated far apart far mineral winding only if deposit is at a shallow depth. Geological disturbances such as faults or dykes may require divisions of the area into convenient zones and shafts may be sunk to serve each zone. Such division into zones may sometimes be desirable if a river, railway or major trunk road (e.g. G.T. road in Bengal and Bihar coalfields) exists on the surface as each of these features requires solid block of mineral to the left-in-situ. In each zone the mineral winding shaft may be so situated as to have 50% to 60% area on the rise side. Moreover the extent served by a mineral winding shaft should be such that the manager of the pit has not more than 2,000 to 2,500 men including surface workers in his unit for ease of supervisor and control.

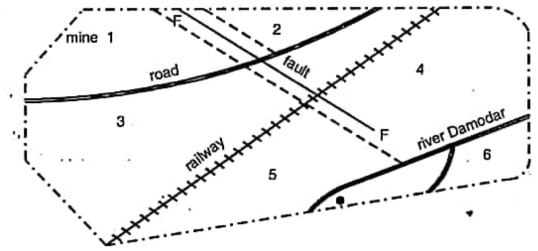


Fig. 6.5 A mine leasehold has to be divided into a few mines due to natural and man-made features

Depth Of Shaft

The shaft should reach nearly 5 m below the lowest bed to be worked to accommodate cheese weights of the guide ropes. If guide ropes are clamped to the girders at underground decking level, shaft may extend only 2 m below the girder level. Where skip winding is adopted the shaft should be sunk deeper by a minimum of 6 m below the underground decking level, depending mainly upon the height of the skip.

In an incline mine in coal upcast ventilation pit should be so such that the pit bottom is in coal which is comparatively stronger than near the outcrop. This reduces the cost on supporting sides and roof of the gallery leading to the ventilation pit-bottom. The return air is warm and laden with moisture, gases, etc. and has a weathering effect. Because of this consideration the upcast shaft should touch the coal seam in 3rd or 4th level if the mine is laid out on bord and pillar system.

The deepest shaft in coal mine in this country (600 m depth) is at Chinakuri Colliery.

Shaft Pillar

Since the inclines and shafts serve as means of access to underground mines throughout their life they should be in strata which are not likely to subside or collapse. In a mine a solid block of rock known as *shaft pillar* should be kept on all sides of the shaft, and only essential rods should be driven through it. Any attempt to extract mineral from the shaft pillar on its periphery will weaken it, and verticality of shaft may be affected, or it may collapse.

On the surface some buildings and other installations have to be situated very close to the shaft for technical reasons, *e.g.* winding engine rooms, boiler plant (in case of steam winders) and fan house. Some other buildings, for the sake of convenience and easy supervision, are also installed close to shaft *e.g.* workshop, electrical sub-station, lamp cabin, pit head bath, office, store, etc. Screening plant, coal bunkers and coal handling plant, crushers are usually situated near the shaft to reduce cost on surface transport though benefit of centralisation for a number of mines may sometimes dictate otherwise. Such buildings and installations last for the whole life of the mine and the shaft pillar should be of such a size as to support them.

The Mines Regulations do not prescribe any specific size of shaft pillar. It may be said in broad terms that in coal mines one side of a square shaft pillar should be equal to depth of the seam and the essential surface buildings or installations likely to last the whole life of the colliery, should be elocated within the area of the shaft pillar. In most of the mines in our country where mineral is raised through vertical shafts, the standard arrangement is to bring the mineral loaded tubs to the shaft bottom and push them into the cage for hoisting to the surface. The cage loading arrangements are in shaft level which is a level road in coal on either side of the shaft. Such shaft level in coal and the other roadways in coal leading to the shaft level, as also essential excavations like pit bottom sumps, underground sub-station, etc. in coal itself demand generous dimensions of the shaft pillar. If however, the shaft level is in stronger rock like stone, as in Chinakuri mine. The shaft pillar in stone need be of comparatively much smaller size.

In the case of skip winding loading of the skip is by a chute at the bottom of a coal bunker which is in stone below the coal gallery housing the tippler. In coal mines worked by horizon mining and in the case of metalliferous mines the pit bottom/plate are in stone. In all such cases where the pit bottom loading arrangements are in stone, the shaft pillar is in stone and of smaller size compared to what it would have been in coal.

Some thumb rules based on experience have been advanced for the side of a shaft pillar in coal.

Considerating D depth of shaft in m,

t = thickness of the seam in m,

R = radius of shaft pillar in m.

1. DRON'S rule : area of the shaft pillar = area to be supported + D/6 on all sides.

2. *Foster's rule :* $R = 3\sqrt{Dt}$

3. *Wadin's rule :*

   For shafts upto 100 m depth, size should not be less than 36.5 × 36.5 m. Thereafter for every 36.5m depth, increase size by 9 m.

4. *Mining Engineer's Rule :* For shallow shaft, Minimum radius for shaft pillar is 18 m.

   For deeper shaft

$$R = 18.3 + \frac{D\sqrt{t}}{32.8}$$

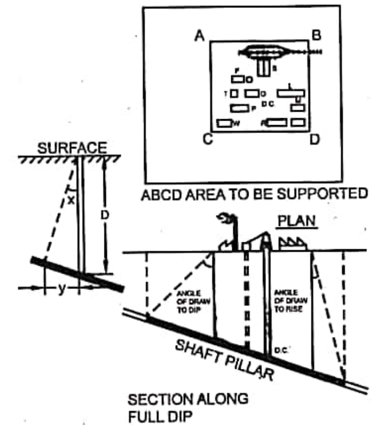


Fig. 6.6 Illustrating shaft pillar

(5) *O'Donahue's formula for inclined seams :*

If D = depth of shaft ; x = angle of dip of coal seam, then y will be D sin x cos x (See Fig. 6.6).

S = Margin of safety, usually equals to 5% to 10% of the depth;

Then width of pillar on the rise side = S + D/7 + 2y/3

Do dip side = S + D/7 − y/3

Do along strike = S + D/7

To calculate the size of a shaft pillar the scientific approach is based on the "angle of draw". Precise observations and scientific studies have not been made in this country to ascertain the angle of draw for the various coalfields but the Central Mining Research Station has conducted experiments in a few cases.

In Fig. 6.6 is the surface area which needs to be suppored. If the angle of a draw is known the shaft pillar may be set out as shown in the figure. It should be noted that angle draw to the dip of the workings and to the rise of the workings have different values.

In the case of an incline or adit there is not such thing as "shaft pillars". In a coal mine developed on bord and pillar method two pillars of coal are left unextracted on either side of the incline for its support and essential buildings and installations can be located to the rise side of the incline mouth. In a metalliferios mine also a solid block of rock is left in-situ on either side of the incline and its dimensions depend upon the strength of the rock.

After the deposit has been entered through an incline or pit, arrangements have to be made at the top and bottom of the pit/incline to deal with the planned output of the mine in a efficient manner.

PIT-TOP & PIT-BOTTOM LAYOUTS

Haulage incline top layout

Fig. 6.7 shows an incline top layout where a direct rope haulage and tubs varying from 0.8 to 1.1 m^3 capacity with pedestal bearing are used. In a coal mine the layout is capable of handling nearly 600 tubs per day of three coal rising shifts and is typical of the mines having little mechanisation and utilising manual labour for tramming of tubs at the incline top.

The tubs are hoisted in set each consisting of 3 to 6 depending upon gradient of the track and horse of haulage engine. The loaded set is lowered and fed by gravity long suitably graded track to the side-on-tipple T_1 which is situated at a high level on gantry to permit of arrangements of screening plant and passage of railway wagons or turcks below. The empties are puhsed manually to the point P, where are attached in a set to the haulage rope and then lowered into the incline.

If tippler T_1 cannot be used due to stoppage of screening plant or non-availability of wagons or trucks, an end-on tippler T_2 is used for tipping the mineral into a dumping yard. Dumping space is, however, limited by the height of the tippler and angle of repose of the mineral. A travelling tippler is sometimes used for facility of more dumping space. Additional space is provided along a branch line L which is utilised

by manual emptying of the tubs after side-on-tipping. The dumping space is utilised fully by extending the track on the mineral dumps and supporting the long protruding raily on wooden props.

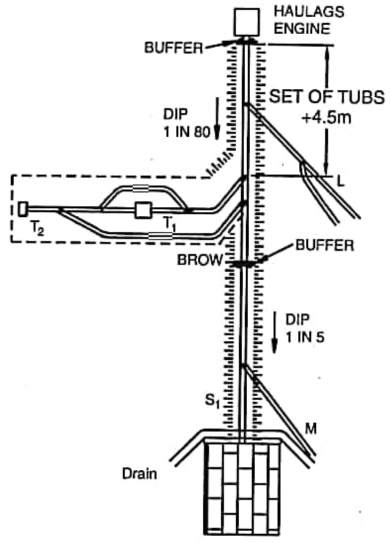


Fig. 6.7 Incline top layout with direct rope haulage

A track M is provided for material which have to be supplied to them and also to carry to workshop the motors, pumps, etc. which are brought from underground for repairs. It also serves to bring to the main line empty tubs which sometimes roll down the mineral dump when tipping manually.

An inter-connected stop-block and runaway switch SS_1 is required at all incline mouths. The distance SS_1 is nearly equal to a length of a set of tubs plus 4.5 m.

The gradient from the brow to the incline mouth and in the incline is steep, nearly 1 in 5, and from the haulage engine to the brow the gradient is dipping at nearly 1 in 80.

Repairs to tubs are effected on the branch line L, which may be utilised for standage of tubs temporarily out of use.

Inclines or drifts from surface to the mine may also be equipped with endless haulage, balanced direct haulage, belt conveyor or a locomotive as the main transporting medium where high outputs are to be handled. The locomotive, due to the nearly flat gradient required for its operation, finds a limited application.

Canveyor incline top layout

With a velt conveyor, the incline top layout may take the shape as shwon in Fig. 6.8 for a coal mine. Main belt discharges the output into an overhead bunker from which coal is tapped into the railway wagons or trucks. Alternatively, a small cross belt may take the coal from the main belt to a ground bunker, which is easier and cheaper to construct and which can augment its capacity by allowing an almost unlimited ground stock to be spread over and about it. The ground stocks can subsequently be pushed into the bunker by dozers. At the bottom of the ground bunker are chutes which discharge coal on to a high-speed belt delivering into railway wagons or trucks through another chute. This arrangement will allow the raising of the mine to continue uninterrupted throughout the day and enable wagons to be loaded when supplied, within the free time allowed by the railways.

When access to the deposit is by a deep shaft, the raising pit may be fitted with a cage winding installation or a skip winding installation.

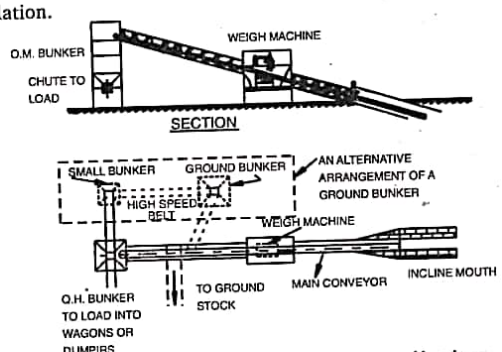


Fig. 6.8 Incline-top layout with a belt conveyor and bunkers.

Pit-top Layouts For Cage Winding Pits

In the case of pits the winding of men and material is by cages, or by skips. Winding by single deck cages, either a single cage or a tandem cage, is the common practice in this country. Multiple deck cages are common in foreing countries and are also employed at modern mechanised mines like Sudamdih, Jaduguda and others. For deeper shafts skips are preferred.

The term "tub" is in common use in the coalfield and the term "mine car" normally is used for tubs of 1.25 te and higher capacity, without brakes it is a standard practice to have roller or ball bearings for the wheels of a mine car.

The raising capacity of a mine depends on the shaft capacity which in turn, depends on the manner in which tubs or mine cars are handled at the pit-top and pit bottom. The design of pit-top and pit bottom layout is done with the following objects in view.

1. Use of the shaft to its full capacity.
2. Use of minimum number of tubs in the circuit.
3. Use of minimum number of operatives.
4. Maintaining steady flow of tubs.
5. Minimum decking time.
6. Lowering of materials.
7. Handling of ores or coals of different grades.

In any pit top arrangement it is essential that the loaded tub or mine car, raised from the pit, discharges mineral close to the shaft and returns to the cage, so as to require the least number of tubs in circuit. It is also imperative that mine cars are not allowed to run freely under gravity over long distances.

Run round arrangement

A run-round arrangement is shown in Fig. 6.9. From the decking level, the loaded tubs are taken to the tippler T via weighbridge W and empties travel by gravity to a creeper (which elevates them to a little above the decking level) and gravitate to the other side of the cage. A creeper on a load side is not desirable and the usual arrangement therefore is to have the decking level 4 to 6 m above the ground level on gantry. A weight-bridge for all the mine cars raised from the pit is a good practice but is uncommon in our mines. If the quality of mineral raised from the pit is not uniform, sometimes due to working of two or more seams of coal (or ore from 2 different levels) by the same pit, two or more tipppliers have to be provided for the different seams, each raised by a separate pit. One tippler T_1 has been provided for the loaded tubs containing shale or stone, which may be diposed of by a belt conveyor. Provision may be made for alternative arrangement to unload the coal tubs, when the usual tipplers cannot be used due to breakdown or stoppage of screening plant. Such arrangement consists in providing one or two travelling tipplers, depending upon the output for tipping the coal into a dumping yard.

When the decking level is above the ground level, the materials are lowered into the mine by loading them into the cage at ground level and an opening in the shaft walling, equipped with a gate and a track, is provided for this purpose; alternatively, a hoist is used for taking material to decking level.

The main disadvantage of a run-round arrangement is the large space required and the long circuit which the tubs have to pass, specially with long wheel-base mine cars which require large radius curves. (For other pit top layouts see Vol III of this book).

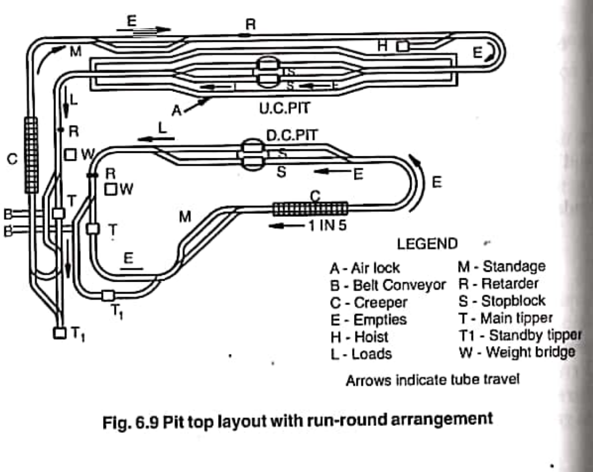


Fig. 6.9 Pit top layout with run-round arrangement

Pit-Bottom Layouts For Cage Winding

The type of pit layout to be followed depends upon the type of transport system used in the vicinity of the pit bottom and the method of winding, whether skip winding or cage winding. The pit bottom layout lasts the whole life of the pit, and has to be designed to meet the maximum production likely to be handled by the pit, as rearrangement of pit bottom is expensive and may involve costly excavations in stone over a wide area, resulting thereby in weakning of the shaft pillar. The rearrangement takes a long time and hampers normal production.

Though a pit bottom layout essentially depicts the transport arrangements near the pit bottom to deal with a targeted output, ventilation, drainage and support arragements have to be considered in designing it.

Rope haulage layout

A common layout, typical of many Indian coal mines, using rope haulage and cage winding, is shown in Fig. 6.8 considering ventilation by exhaust fan, which is the normal practice. The points to note in the layout are :

1. Winding of coal is from only one level which is in the coal seam so that pit bottom decking arrangement is in coal.
2. Shaft on the dip side is D.C. and the main coal winding shaft.
3. The two shafts are connected by a roadway provided with an air-lock and a track for materials. Materials are generally lowered from the surface by the U.C. shaft which has little output, if any, to handle. The air lock consists of three doors, opening towards the D.C. shaft against air pressure. The third door, opening in opposite direction, is required in an emergency which may necessitate reversal of ventilation system. Distance between the two doors should be not less than the longest timber or rail likely to be transporated through the air lock.
4. D.C. shaft carries high voltage cable from surface to the sub-station near the pit bottom. U.C. shaft being warmer is generally not preferred for power cables. D.C. shaft carries telephone cables also.
5. Main delivery column from pit-bottom pump is taken to surface by the D.C. or the U.C. shaft according to convenience.

6. A by-pass is provided near the pit-bottom for persons to go from one side of the pit bottom to another without crossing the pit.

7. The shaft levels in East and West directions are not exactly levelled, but slightly dipping towards shaft-bottom at a gradient of 1 in 80 to 1 in 120, depending upon the type of tubs or mine cars used. This facilitates drainage also.

8. The shaft levels are equipped with double tracks, one for emptiy and the other for loads, on either side of the shaft. A stope block on each track near the cage is essential and diamond crossings are provided to give clear sapce of only 2 to 4 tubs near the cage. Tub retraders are also usd in the shaft level near the pit bottom. Ordinary tubs, with 1.1 m^3 capacity, can often be slowed or even stopped by the use of sprags and may obviate the need for retarders.

9. A third track or siding 10 to 15 m long has to be provided for standage of material transporting trolleys.

10. Standage for loads has to be provided on East and West shaft levels for occasional stoppages of winding system. The standage generally provides for half an hour's output during peak raising periods. Considering a raising of 30,000 te per month or nearly 1200 te per day with three working shifts or 400 te per shift, and assuming that this is raised in six hours of an eight-hours shift, half an hour's output is nearly 35 tubs, each of one te capacity. Shaft level on either side of the pit should have standage for about 20 tubs *i.e.* clear space of nearly 40 m.

11. The endless haulages are situated near the shaft bottom. The dip districts are served by direct haulage and the rise districts by tail rope haulages. Crossing of direct and endless is avoided by either of the arrangements shown at A_1 and A_2. At A_1 the endless rope passes above the direct rope. This reduces the span of the bridge which has the further advantage of supporting only slow moving traffic of endless haulage. At A_2 the loads are taken over a brick ramp which is raised in the middle for movement of loads, when detached, by gravity to the shaft level. For empty, the track is suitably graded from the shaft level to the clipping point at A_2 for supply to the dip district.

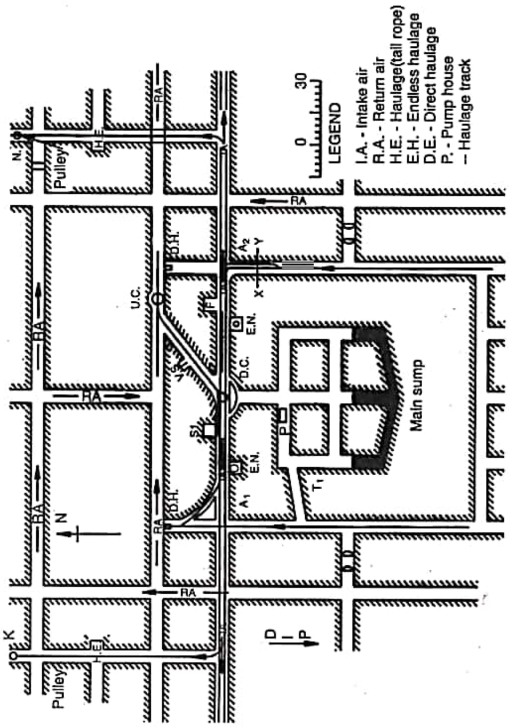


Fig. 6.10 Rope haulage layout at put bottom

12. The shaft pillar occupies an area KLMN in which only essential excavations and galleries are made for haulage, drainage, ventilation, etc. These include a main sump with overflow gallery T_1, main underground sub-station S_1, free fighting F, an underground office S, an underground store and ambulance

room V. The office, stores and ambulance rooms should be away from tub traffic; for high output which involves heavy mechanisation an underground workshop is desirable.

This type of layout will serve upto 30,000 te/month and the haulage arrangements will work on gradients from 1 to 4 upto 1 in 12. For gradients milder than 1 in 12, endless haulage should replace direct haulage.

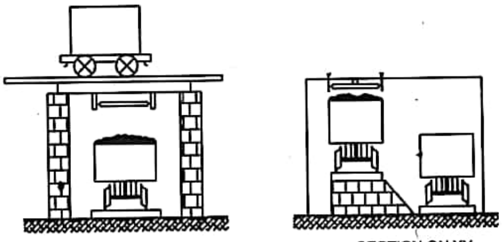


Fig. 6.11

Underground crusher level layout in a metalliferous mine is shown in Fig. 6.13. The figure shows the layout between level 7 and 8 at Mochia Main Mines, Zawar (Hindusthan Zinc Limited) in Rajasthan. The shaft is equipped with tower mounted koepe winder for skip (400 H.P.) and another tower mounted koepe winder for cage (200 H.P.) The cross-section of the shaft, rectangular in section, is 5.2 m × 3.8 m. It has a capacity to handle 1200 te of ore per shift of 8 hours. The layout includes an underground crusher and orebin which is a feature of some metalliferous mines but rare in coal mines. The pump chamber and sump are located at 7th level.

Midest Landing Arrangement

When mineral has to be raised from two or more seams or 2 levels by the same pit, the arrangement that has to be made in the middle or intermediate seam is known as shaft *midset landing* arrangement. Where guide ropes are used there are oscillations of the cage during winding and it is not desirable to instal fixed receivers at the midset landing to receive the cage. A fixed platform at the midset landing is also not desirable as the cage, because of its oscillations, may strike it. Rigid guides of rail or wood are quite convenient for the cage if midset landing is provided.

The arrangement at midset landing with rope guides is shown in Fig. 6.14 and the main requirements are as follows :

i) *Hinged platform* are used. These are operated by levers manually handled. The two platforms on either side of the shaft at midset are interconnected by rods and links. The platforms bridge the space between the midset and the cage at each side of the shaft.

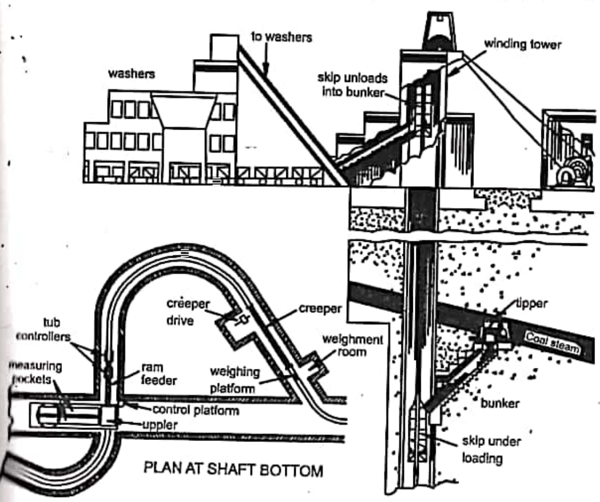


Fig. 6.12 General arrangement at the pit top and pit bottom loading point

When the platform is lowered it is supported by buntons fixed well clear of the cages and comes in level with the floor of the cage. Counterbalancing weights are used to manipulate the platform.

ii) *Safety gates* are provided at each side of the mideset landing. These are of the sliding type or collapsible type. They may be so interlocked with the lever or the hinged platform that unless the platform is lowered in position, the gates cannot be opened.

iii) *Stope blocks* are essential to control tubs at the mideset.

Telephones and arrangements for coded signalling to pit top are required. A separate and distinct code of shaft signals should be used specially for the midset landing only. It is desirable to have red and green light signals which will automatically operate with the manipulation of hand lever of hinged platform. The pit top banksman can then know whether the midset is free for cage winding (green light on).

When two seams or two levels are being worked by the same shaft and both have practically the same output, it is convenient to make such arrangements of winding that when one cage is at midset the other is at pit bottom decking level. This can be arranged by use of differential drums, *i.e.* two drums of different diameters on the same shaft of winding engine. Fixed platforms can then be used at midset.

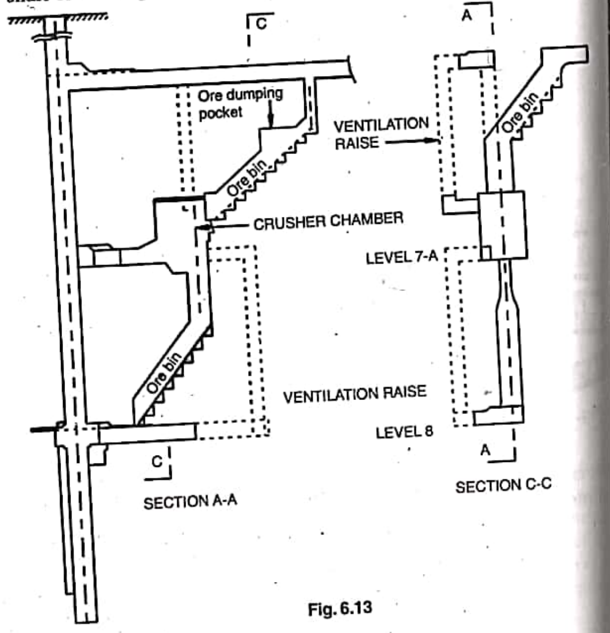


Fig. 6.13

Drainage Of Shaft Bottoms And The Main Sump

If a shaft carries guide ropes, these are loaded with cheeseweights at the bottom end to keep the guide rope tigh, permitting only little oscillations. Shafts are, therefore, sunk 4.5 m to 6 m deeper than the pit-bottom decking level to accommodate the cheese weights. To avoid water of the pit bottom over-flowing into the shaft level and also to keep the cheese weights clear of water, provision has to be made for drainage of the shaft bottom. The arrangement may employ one of the following methods.

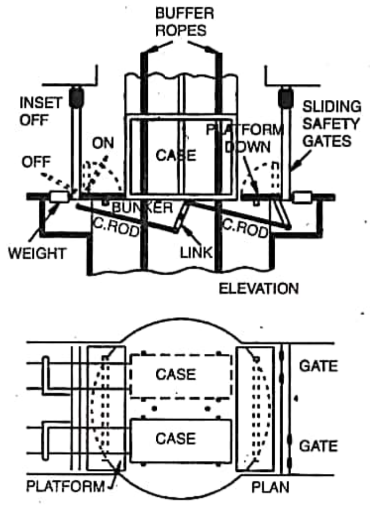


Fig. 6.14 Midset landing arrangement

1. One pump is installed in the shaft level and suction pipe is placed in the shaft bottom below the decking level. The pump is worked as and when necessary to pump out water accumulation below the decking level. A ladder from the decking floor to the pit bottom gives access to the foot valve and strainer of the suction pipe. This method is applicable where pit bottom is only 3 m to 5 m below the decking level and guide ropes are

preferably clamped to avoid congestion by guide ropes, cheese weights and pump-suction below the decking level.

2. Water is allowed to gravitate to the main sump by connecting the pit bottoms of both shafts to the sump by a drift dipping towards the sump. Fig. 6.15 shows the D.C. and U.C. shafts connected at the bottom end by a level gallery in stone. To prvent short circuit of intake air to U.C. through the connection, a constant water seal is maintained by the proper design of the gallery as shown in the figure and the muck is cleared by an auxiliary winch and bucket at intervals.

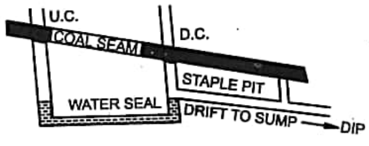


Fig. 6.15 Drainage of upcast and downcast

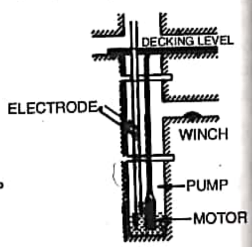


Fig. 6.16 Submersible borehole pump for shaft bottom

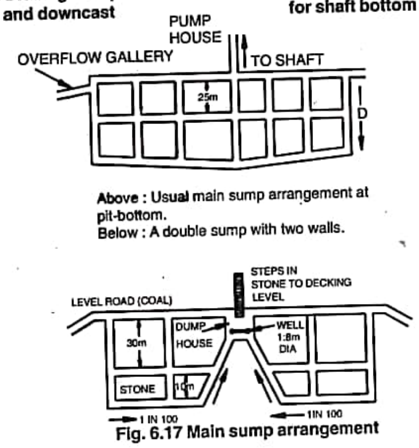


Fig. 6.17 Main sump arrangement

A staple pit with a ladder, or a stone drift from the coal seam, gives access to the drainage drift. This arrangement is commonly adopted.

3. Submersible bore hole pumps may be used in the bottom. The submersible bore hole pump has its bottom at least 400 mm clear from the pit bottom. The pump can be raised or lowered by a winch placed in a roadway driven in stone below decking level. The roadway has access from the shaft level by a staple pit or by a small drift. Three electrodes are suspended from the shaft and reconnected to an electrical system. The continuity of electrical circuit is through the water in the pit bottom; if the water level falls below the middle electrode, continuity is broken and a relay trips the current to the motor. The pump is self priming and is arranged to start and stope automatically. Access from winch gallery to the pump is by a ladder.

Main Sump

Water from all the districts of a mine is collected at a main sump situated on the dip side of the pit bottom, the usual arrangement in a coal mine being as shown in Fig. 6.17. The galleries are driven through coal to store the water. The esssential features of the main sump are :

1. The dipmost "level" galleries should dip towards the junction to facilitate water flow towards a common point. The junct on is the suitable place for strainer and foot valve. To keep the latter always under water, it is helpful in flat seams to make a small pit. say 1.8 m dia. × 1.2 m deep, at the junction.
2. The suction head including friction should not exceed 6 m for smooth and trouble-free operation of the pump; if necessary, level galleries of the sump may be at shorter distance than that prescribed in the Regulations with prior permission from the D.G.M.S.
3. The sump should be able to store water for 24 hours during rains in case of pump failure. More storage capacity will require more number of galleries in the main sump.
4. An overflow arrangement should be provided. The overflow gallery should be connected to a dip gallery which should not

have a haulage track, otherwise the overflow will wash away the tram line packing. Where this is not practicable the dip haulage gallery should have a drain for over flow water.

5. The pump house should have standy motor-pump unit and it should be so situated that it is not affected by overflow water of sump. It should be well ventilated by intake air. Track should be laid from shaft to pump house for transport of pumps, motors, pipes, etc. and extended upto foot valve when the sump is to be cleaned by loading muck on tipping tubs. The pump house should be spacious to accommodate pumps, motor and switchgear. It should be high enough to permit provision of lifting tackles over the pump and motor.

Main Sump With Two Wells

Another type of main sump with double sump arrangement is preferred in a mine where standstowing is practised. Water from sand stowing districts invariably carries and which is pumped to the main sump by the inbye district pumps. The main pumps therfore wear fast and need frequent repairs and overhauls when pumping gritty water mixed with abrasive sand. It therefore becomes necessary to water mixed with abrasive sand. It therefore becomes necesary to

1. Clean the main sump at intervals.
2. Provide additional sump when one sump is being cleaned, settling tank or similar arrangement where sand can settle down.

One such arrangement is shown in Fig. 6.17. The pump house is common for both the sumps. Each pump has its galleries so arranged that the water gravitates to the well which carries the suction pipe, foot valve and strainer of the pump. The well may be nearly 6 m deep and 1.5 m dia. The two wells are connected 0. 6 m below the top by a 200 mm dia. pipe fitted with a sluice valve so that the water may be shared by both the wells when one of the wells is nearly filled to the brim. Excessive overflow in water is stored in the drift and the levels which form the sump. Each well is provided with arrangement for lifting the suction pipes.

By arranging discharge of the district pumps to feed only one sump and the well serving it, suitable arrangements can be made to keep the other sump and well out of use and free for cleaning.

Chinakuri colliery is provided with arrangement of double sumps.

✧ QUESTIONS ✧

1. What are the considerations in selecting the site of a shaft ?
2. What is a shaft pillar and state the formulae for deciding its size ?
3. Give a suitable pit bottom layout in a seam 3 m thick dipping at 1 in 8 for an output of 25,000 te per month with 3-shift working.
4. What are requirements of a main sump in a pit mine where sand stowing is adopted ? Give a layout of a double sump.
5. Give an arrangement of midset landing if mineral has to be raised from two seams by a vertical shaft.

✧ ✧ ✧

Chapter -7

Drivage of Roads in Coal & Stone

After a shaft reaches about 2 m below the coal seam to be worked or below the particular horizon or level in a metalliferous mine, the formation of roads is carried out according to a worked out plan. Where a shaft touches a coal seam, essential roads in coal are driven to reach the limits of shaft pillar for ventilation, transport, drainage, stowing and distribution of electric/compressed air power. Beyond the shaft pillar the drivage of roads and extraction of coal depends upon the method of work to be adopted to win the coal.

Whenever any work is being done in a mine the following points should be borne in mind as they are the cardinal points for any mining activity, safety dominating all other considerations.

1. Safety and mining regulation.
2. Support of roof and sides and surface features.
3. Ventilation.
4. Drainage.
5. Transport.

Needless to emphasize, economics must always be kept in mind when considering any of the cardinal points stated above.

Drivage of roadway in coal in a mine so that their network reaches a predetermined boundary is known as *Development* of the mine. The boundary may be the lease boundary of the mine, a fault plane, dyke or any other artificial boundary.

In a metalliferous mine, a producing underground mine requires a carefully planned network of shafts, drifts and raises. The preparation of this network is known as *development*.

Development of a mine by a method of working known as the *bord and pillar* method-the normal method of working in this country-consists of driving a series of narrow roads, separated by blocks of

solid coal, parallel to one another and connecting them by similar set of narrow parallel roadways driven nearly at right angles to the first set. Fig. 7.1 shows the plan of a general scheme of developing a mine by bord and pillar where the entrance is by inclines.

Definitions

The terms used in coal mining practice are given below :

A seam is assumed to be *thin* if normal thickness is less than 1.5 m; *moderate* if between 1.5 m and 4.5 m; *thick* if between 4.5 m and 9.0 m and *very thick* if beyond 9 m.

The stratum is referred to as *flat* if general inclination does not exceed 5° to the horizontal; *inclined* if it is between 5° and 18°; steep if it is between 18° and 40° and *very steep* if inclination is beyond 40°.

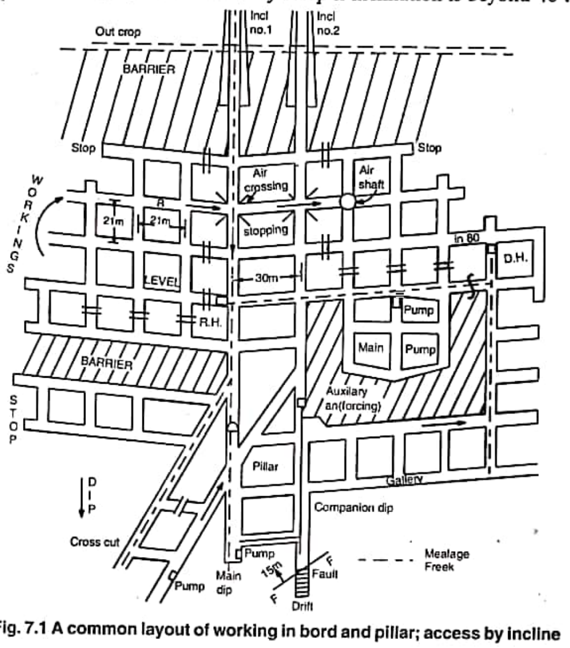


Fig. 7.1 A common layout of working in bord and pillar; access by incline

The workings are assumed as *shallow* if their average depth is below 200 m; *moderately deep* if between 200 m and 450 m and *deep* if beyond 450 m.

The gassiness of a coal seam is laid down by a circular of the D.G.M.S. in 1967 and the mines are divided into three categories of gassiness as shown in the table below :

Gassines degree	% of inflammable gas in general body of air	rate of emission of gas m³/te of coal raised
I	below 0.1 and	below 1
II	above 0.1 and/or	1-10
III		above 10

A road in a coal seam proper is called a *gallery*.

A road which is driven along the dip of the seam is called a *dip gallery* and sometimes only a *dip*. A road driven along the strike of a seam is called a *level gallery* or simply a *level*. A dip gallery may be along the true dip of the seam or along an apparent dip. A level gallery may not be truly along the strike of the seam but may slightly rising inbye, *i.e.* towards the working face, for considerations of haulage and drainage.

A roadway in stone connecting two ore more coal seams is called a *drift*. It is sometimes referred to as *stone gallery* if only one of the ends of the road is in coal and the other is a blind end.

A solid block of coal surrounded on all sides (or nearly all sides) by galleries is known as *pillar*. It forms the natural support of the roof in a mine.

Where the galleries in a seam are generally along the dip-rise and strike forming square or rectangular pillars, a gallery which cuts across the pillars, due to its drivage along an apparent dip, is called a *crosscut*. Such crosscuts are sometimes required for facility of ventilation, drainage, haulage or stowing.

A gallery in the process of being driven is called a *heading*. The moving front of any working place on the inbye end of any gallery, roadway or drift is called a *face*. A face in coal is also called a *working place* or simply a *working*.

When a set of dip or rise headings is driven, one of hte headings is called a *main dip or main rise* repsectively and the other headings on either side are known as *companion dips or companion rises*. The main dip or rise usually carries haulage arrangement and sometimes auxilliary fans. The main dip is equipped with face pump in addition.

A *district* is an area in a mine having a number of working places. The word "Section" is also sometimes used to denote a district, e.g. north section, west section, etc. But the use of the term *section* is not recommended where a seam is being worked in two or more sections like top section, bottom section, etc. The term *panel* is also sometimes used to denote a district which is separated from other district by an artificial barrier of brick walls or by a natural barrier of coal.

Cover is the vertical depth of a place in the mine from surface.

Metal Mining Terminology

The **terms** used in **metal mining practice** are as follows (Fig. 7.2).

Level : A level roadway in the ore body or be in which follows the strike. In metal mines a level is rarely in a straight line (unline in coal mines).

Cross-cut - A level tunnel or roadway which leads from the shaft or level and passes through the country rock in order to cut across the lode at an angle to the strike.

Drive or drift - A horizontal tunnel or roadway parallel to strike of the lode or vein but it can be located in the country rock either on the footwall side of the lode or on the hangwall side. It is called a footwall drive in the former case and hangwall drive in the latter case. The term *level* is sometimes used for drive, but the distinction should be noted.

Cross drift or Cross drive : It is horizontal underground roadway driven within the ore-body between the hanging wall and the footwall. It is usually at right angles to the drive or drift.

A *reef drive* is in the vein itself or partly in the reef and partly in the wall rock, usually the footwall.

Level interval : The vertical distance between two adjacent main levels, main horizons or main drives.

Sub level or *intermediate level :* a level or drive situated between the main levels or main drives.

Raise : A connection between two levels in an orebody driven in an upward direction.

Raises intended for passing ore from an upper level to lower one under its own weight are called ore *chutes*. Short ore chutes intended for drawing broken ore from the blocks are called *Draw holes.*

Ore pass : An ore pass is a vertical or steeply inclined underground passageway for downward movement of ore by gravity. This term is not used in coal mining.

Draw point : A spot on the floor from where gravity fed ore of a higher level is loaded into tubs or minecars.

Winze : A dipping connection in the orebody joining two levels. A raise or winze is located mostly near the footwall of the orebody.

Plat or station : It is the excavation adjoining the shaft at each of the different levels where men and materials are removed or delivered.

Stope : An area from where ore (and in some cases, a little country rock in the hangwall and footwall) has been extracted and the hangwall allowed to cover or supported by filling of some material like sand, mill tailings, blocks of grainite, etc.

Stoping - Extraction of ore from block or pillar formed during development. As a rule stoping is started on is started on each side of a raise-winze connection.

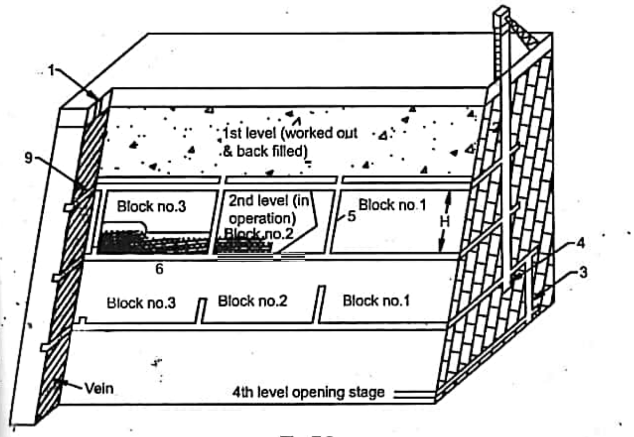


Fig. 7.2

1. Auxiliary shaft, 2. Main shaft, 3. Staple pit, 4. Sump, 5. Raise, 6. Short ore chute or draw hole, 7. Cross cut, 8. Drift along strike of ore body, 9. Cross drift.

In metal mining terminology development of a mine means formation of a network of shafts, drifts and raises, in the early stages of mine work, with a veiw to

(a) prospect the orebody from underground for more details,

(b) arrange for drainage, ventilation and transport,

(c) divide the mine area into convenient sections for future working, and

(d) enable the mine planners to decide upon the methods of stoping.

Sizes Of Pillars And Galleries (Coal)

The width of the road varies from 3 m to 4.8 m, 4m width being common where coal cutting machines are used. The height varies from 2 m to 3 m, nature of immediate roof being the deciding factor. Under the coal mining Regulations the gallery in a seam should not be more than 4.8 m wide and not more than 3 m high.

The roads used for travelling purpose should be separate from the haulage roads having rope haulage, belt conveyor, locomotive haulage, belt conveyor, locomotive haulage or gravity haulage. The travelling road should be not less than 1.8 m high. Men are not permitted to travel on mine cars, tubs or conveyors.

The minimum size of pillars and the maximum width of galleries to be formed in a mine during development are prescribed by the Regulations. For the same depth the wider the gallery, the larger should be the pillar size. Similarly for the same gallery width, the size of a pillar increases with depth. The following table gives the pillar as laid down under the Regulations.

Depth of seam from surface	Maximum gallery width			
	3 m	3.6 m	4.2 m	4.8 m
	Minimum distance of pillar centres (m)			
60 m	12.0	15.0	18.0	19.5
61-90 m	13.5	16.5	19.5	21.0
91-150 m	16.5	19.5	22.5	25.5
151-240 m	22.5	25.5	30.0	34.5
241-360 m	28.5	34.5	39.0	45.0
exceeding 360 m	39.0	42.0	45.0	48.0

Pillars should be of larger sizes near a fault plane which is generally a weaker section, near a crushed zone and in the area where the coal is soft or friable. In hilly areas the pillar has to be large at local places under the hills as the depth from the surface naturally increases. Where the roof or floor has a tendency to creep or heave in the galleries, pillars should be large and the galleries small in width. If the pillars formed during development are expected to be depillared long after their formation, they should be of larger size. In steep seams the pillars are rectangular with the longer side along the strike. This provides the advantages of opening levels comparatively early.

The pattern on which the galleries are to be driven is decided by the manager and marked on the plan which is known as the

projection plan. The direction in which the galleries are to be driven is therefore known to the surveyeory and he has to mark the centre lines of the galleries on the roof in the coal seam and extend them upto the face daily or on alternate days depending upon advance of faces.

DRIVAGE OF GALLERIES IN COAL

The methods commonly employed for drivage of galleries in coal are :

(a) mechanical method of drivage with coal cutting machines and drills.

(b) mechanical method of drivage with heading machines which cuts and load the coal, like the continuous miner, coal auger, and road header.

The floor of the gallery should be parallel to the floor of the seam. This is important when the galleries are being driven in the middle or top section of the seam. Some distinctive band of shale, stone or bright coal in the seam helps to give an idea of the floor of the gallery, and serves as a guide.

Mechanical Methods Of Drivage

Galleries are usually driven in pairs or sets of 3,4, or 5. From considerations of ventilation drivage of minimum two galleries is essential unless the gallery has to go only a small distance. Drivage of galleries by coal cutting machines and coal drills is a common practice for quick progress of headings, large output and fast development of the area. When coal cutting machines are used the operations at the face require.

i. giving a cut by the coal cutting machine,

ii. drilling of shot holes.

iii. charging with explosive and blasting,

iv. dressing of roof and sides to render them safe.

v. loading of coal into tubs, mine cars or conveyors.

vi. dressing of sides, roof and floor to make the face ready for the next cut.

Blasting-off-the solid (known popularly as *solid blasting*) was introduced in Indian Coal Mines in the early sixties with the appearace of P-5 explosives in the market. Prior to that the Coal Mines Regulations required that before blasting at a coal face a *free face* has to be created and this necessiated use of coal cutting machines for giving undercuts or overcuts at the coal face.

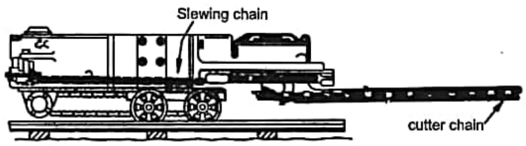
Skid mounted coal cutting machines were used in bord and pillar method of mining (mosly under-cutting machines) for a cut at the face to provide a free face before blasting. The introduction of explosives for "solid blasting" eliminated the need to have a "free face" resulting in gradual disappearance of coal cutting machines from the mines. In recent years, MAMC is manufacturing arcwall coal cutting machines for longwall and bord-and-pillar mining and it seems the coal cutting machines are likely to stage comeback.

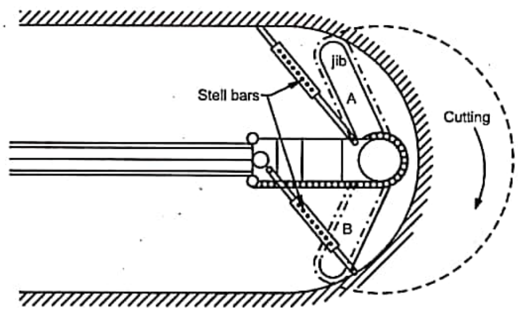
As the skid mounted machine has to pass in the space between the haulage track and the coal pillar, the galleries should be at least 4.0 m wide if double track (0.6 m gauge) is laid in the gallery and the delivery range of face pump should be as close to the pillar as possible and preferably placed on large iron pegs in the coal pillar at about 1.2 m above floor.

The gate end boxes in the level gallery should be on the rise side to avoid damage to them by accidental slipping of the machine during flitting, an occurance to be expected in seams steepr than 1 in 6. Length of the trailing cable used for machines is restricted to 100 m by the Electricity Rules and this limits the range of operation of the machine from the gate and box.

The electric coal drills are normally of 1.25 h.p., helped by two workers in hand during drilling. The hole sizes is 30 mm dia. and the depth may be 1.1 m to 1.5 m depending upon the depth of the cut as the hole must be at least 150 mm shorter than the cut under the Regulations if P-5 explosives are not used. In an 8-hour shift a crew of 2 workers can drill 50 to 60 holes, each 1.5 m deep.

The position, direction, depth and number of holes at a face are governed by factors detailed in the chapter on explosives and blasting.





Arcwalling a wide heading

Side view of an arcwall coal cutter

An arewall coal cutter (a) jib ready for cutting (b) jib at the end of a cut

Fig. 7.3

Support

The coal pillars formed during drivage of galleries form the natural support. Where the roof is bad wide junctions should be avoided and level galleries on either side of a main dip should be staggered. Props should be avoided as far as practicable in roads having tub movements on gradients as they get dislodged by run-away or derailed tubs. In a thin seam a bar to support the roof in working places reduces the effective height and workers find it difficult to carry loaded baskets on head for tub loading. Roof bolting or roof stitching provides a good alternative for such situation.

Drainage

Drainage of water in the mine is effected by pumps which are designated as follows depending on the location of their installation.

1. Face pumps, 2. Stage pumps, 3. Main pumps.

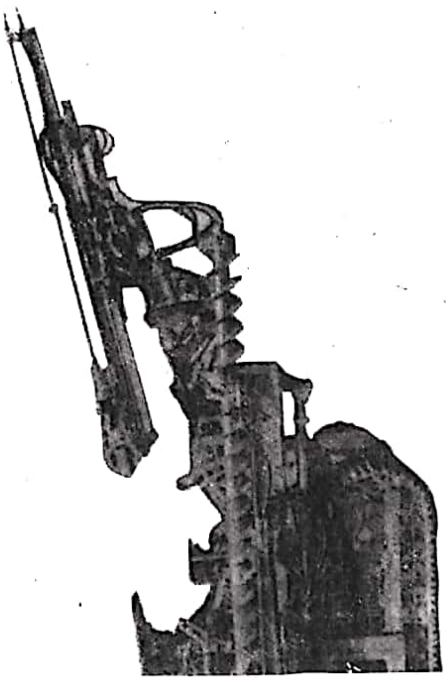


Fig. 7.4

Fig. 7.5 and 7.6 show the general arrangement for dewatering the dip faces. As the face advances the pump at the face also has to advance. For this reason the pump is usually small, of 5 to 15 h. p., 50 mm suction and delivery pipes, with capacity of 250 to 450 mm capable of devloping only 15 to 30 m head, and is mounted on a also should preferably be mounted on the pump trolley. Centrifugal pumps are generally used but Roto pumps have greatly replaced them in recent years.

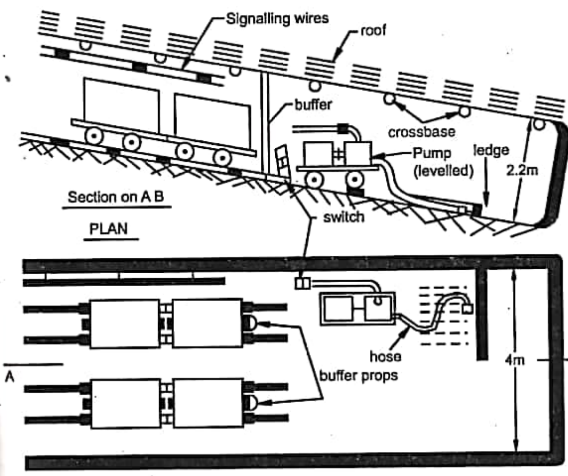


Fig. 7.5 Sectional elevation and plan of a dip face

The face sump is of a temporary nature and has a small capacity as the face has to progress daily. The sump serves one week where the face advances 9 m to 12 m per week. If the thickness of coal seam permits, the face sump may be formed by leaving in the floor a small ledge (called *adkha* in Hindi) 0.4 to 0.6 m high on one side of the gallery and 4.5 to 6 m outbye the face. Alternatively, in a thin seam the face sump may be formed by blasting the floor stone to give an excavation 2 m × 1.5 m × 1.3 m deep so as to have a capacity of 2500 to 3000 litres.

The pump discharges water into the sump of semipermanent pump on the outbye side, called the stage pump, which delivers water to the main pump. The sump of the stage of the pump may be formed by driving a short gallery S in a pillar if the latter is large. If the pillar is small a gallery driven to a length of about 5 m as shown in Fig. 7.6 may serve as a sump.

Small accumulations of water in a dip working at the face are bailed out by bailing majdoors with the help of buckets. The bailed out water is allowed to gravitate to the face pump of main dip or companion dip and from there it is pumped out. As a substitute for bailing majdoors small pumps operated by the coal drill are also available *e.g.* Rana Drill pump.

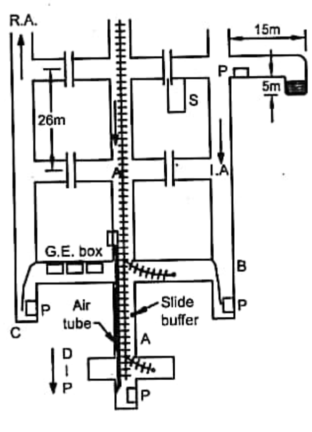


Fig. 7.6 Usual arrangement of pumps, transport (direct rope haulage) and ventilation in dip faces P - pumps, F - forcing fan

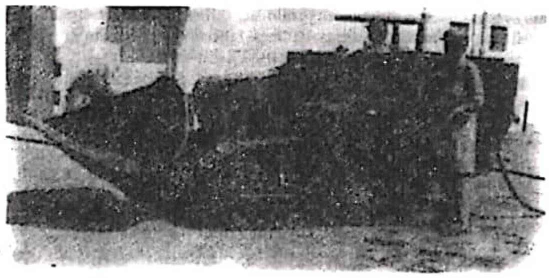


Fig. 7.7 A side discharge loader

Courtesy of EIMCO-ELECON (India) Ltd.

Each mine has a main sump to which all the water of the mine is pumped. Fig. 7.1 shows the main sump in an incline mine and Fig. 6.9 depicts the main sump in a pit mine. The delivery range is taken to the surface through a bore hole, through the incline (in the case of an incline mine) or through the shaft.

Because of the troubles in dealing with water the main dip which has the pump is pushed ahead and the companion dip is driven only a few metres to aoid heavy bailing. When the main dip advances about 3 m in excess of a pillar length level galleries are opened off it. From such level galleries companion rise galleries are driven to make connection with companion dips that have been driven some distance from the higher level.

In a mine having large make of water all the three headings need pumps and water from two headings, say A and C, is discharged into heading B. From there it is pumped out to stage pump.

Transport

The common mode of transport in majority of our mines is rope haulage, though scraper chain conveyors, belt conveyors and locomotives have been introduced in some mines. The transport used for an advancing heading should be capable of providing transport facilities near the advancing inbye faces. Direct haulage suits well in this respect for the advancing and the tail rope haulage for the rise headings. For level headings endless rope haulage is the common practice when the roadway is nearly 100 m or more in length, the shorter distances being covered by hand tramming before installation of haulage.

Two tracks, one for the load and the other for the empties, are always essential at the loading points at the face. The haulage tracks should be as near the face as possible for convenience of loaders and to avoid expenses on lead. When driving two or more headings a skid mounted machine has to pass in the space between the track and the pillars.

Scraper chain conveyors can be readily kept close to the face, within shovellable distance, by addition of pans. The scraper chain conveyor may deliver coal into tuvs or into a belt conveyor. A belt conveyor is not fully utilised if imployed for drivage of one heading or only one pair of headings due to insufficient coal and its extension is not as quick and simple as in the case of scraper chain.

Progress Of Headings

Due to water the advance of the dip headings is slow and the general tendency is to drive level and rise galleries which are free from water. Generally 2 cuts by the coal cutting machine per day in the main dip is a reasonable performance. A weekly progress of 12 m or a monthly progress of 48 m is not difficult to achieve. A progress of 300 m per annum in *dip headings* is satisfactory with skid mounted coalcutting machines and rope haulages.

In a developing mine the aim should be to drive main headings, particularly the dip headings. In a new mine which has no railway siding facilities, concentration should be on drivage of the dip headings instead of on production. To have coal production at a reasonable cost, it is advantageous to keep one development district on the strike with 5 to 6 headings. When the siding is available, or suitable arrangements for disposal of coal are made, it is easy to open districts along the strike and boost up the raisings.

Ventilation

The ventilation arrangements adopted when driving two or three headings have usually to be altered when developing a district. Ventilating air is normally coursed to the working places by stoppings, brattices and doors, but brattices and doors on haulage roads are inconvenient. Often the workers leave them in opened conditoins and their utility is then lost. Partioning a roadway by brattice cloth is inconvenient as it reduces the width of the roadway and restricts the movement and speed of the tub loaders. The brattice cloth partition, to be of any use, has to be extended upto the face. This is not always possible because of movement of coal cutting machines and coal loading. Auxiliary fans (F in Fig. 7.6) of forcing type with flexible ventilation tubing or ducts are good and T or V branches from these tubings can be connected. The tubing is suspended from the roof, or supported on long pegs driven in the sides, to avoid congestion of the roadway. Auxilliary fan should be so installed as to avoid recirculation of air.

In very gassy mines, drivage of rise headings may have to be stopped to avoid gas accumulation. Only level and dip headings may be drive.

(b) Mechanical methods of drivage of roads in coal with such sophisticated machines like the continuous miner, coal auger, tunnelling machines, etc. have been described in the chapter on bord and pillar development in Vol. 3 of this book.

DRIVAGE IN STONE (COAL MINES)

A drift in coal mining terminology is a roadway in stone, and is driven for access to shallow seams, for passage from one seam to another, for transport, sand stowing and other purposes. Considerable drifting is essential for approach to a coal seam in horizon mining, and to cut down development period thereby saving interest charges on the capital invested in the constructional stages, drifting should be properly planned and speedily carried out. With hand held drilling machines and manual loading, speed of advance of a normal size (10 sq. metre) drift is about 5-6 metres a week, while a highly mechanised and carefully organised operation has recorded a progress of 100-120 metres per week in foreign countries.

Maintaining Gradient And Direction Of The Drift

Where a drift is to be driven to prove the throw of a fault and touch the seam on other side, the drift should be driven, not in proposed main haulage road through the fault, but in the companion road. This allows suitable grading of the proposed main haulage road once the fault is proved.

The surveyor has to mark permanent reference points by plugs in the roof at the starting point of the drift. Such reference points should be in the centre line of the drift and as the latter advances, master plugs are fixed in the roof after every 10 m along the centre line.

The gradient line is marked on the side by red lead, 1 m above the floor of the drift. Plugs are fixed on the grade line after every 3 m or 4.5 m. A template on which a spirit level may be mounted, is used to extend the grade line or check the gradient of the floor in day-to-day practice. This is done by the overman or drift-in-charge. The surveyor checks up the centre line and the grade at frequent intervals, usually once a week, by taking levels of the rail track laid in the drift.

Drilling

For soft sandstone and shale the usual 1.25 H.P., hand held, 110 V. rotary coal drill can be used by (i) replacing the gear box to reduce the speed of drill rod from nearly 600 r.p.m. to approximately 240 r.p.m. (ii) use of turbine section drill rod instead of the diamond section drill rod used for coal drilling. The diamond section drill rod wears very fast. The drill bits used for coal can be used for the soft stone also, though the life naturally is much less. The improvised drill which normally has a rating of 20 of 25 minutes in coal, gets hot after use for 10 to 15 minutes and sufficient time should be allowed for its cooling down before re-use. Nearly 15 holes may be drilled in one shift of eight hours. The pull is only 1.2 to 1.5 m and with one hand held drill in conjunction with manual loading of muck into tubs, a weekly progress of nearly 5 m is possible in a drift of 4 m × 2 m cross section, 2-shift working.

For hard standstone or similar rock, the same arrangement with eccentric drill bit works well, but the drilling is slower. More progress and accurate direction of holes can be achieved by the use of power feed or mechanical feed arrangement.

Drilling in hard strata is generally carried out by compressed air operated drills, usually with mechanical feed. The rate of penetration varies from 0.2 to 0.3 m per minute in hard rocks. One jack hammer drill, mounted on air leg and adopted for wet drilling is used in a drift of 4 m × 2 m cross-section. An air leg relieves the operator of the fatigue involved in holding the drill and keeping it pressed forward as it exerts an upward lift and a forward feeding pressure on the drill. An air leg does not increase the rate of penetration or feed as it is used for drifts upto 2 m height (Fig. 7.8). Drilling rigs or jumbos have to be used for high speed and for large-sized drifts. The terms "jumbo" and "rig" are used synonymously. A jumbo is a portable carriage which has arms for mounting of 2 or more drills. The arms can be raised, lowered and slewed at any angle in position by hydraulic or air pressure and all the drill steels are placed in the carriage.

After the alignment and gradient are fixed, the drifting starts with drilling shot holes according to a specified form, called "drilling pattern", which consists of three or four groups of holes. The drilling patterns suitable for different types of rocks under different conditions of excavation in mining and the blasting methods are described in the chapter dealing with explosives and blasting. It may be mentioned as a broad guideline that burn-cut and wedge cut pattern are normally favoured for drifting in stone.

Blasting

All holes are charged and fired simultaneously with the use of milli-second delay detonators as the quantity of explosives is heavy. "Inverse initiation" is the general practice. "Sumpers" are connected to instantaneous detonators, "first eases" to delay detonators with delay *upto* 100 milli-seconds. The second easers and trimmers are similarly connected to delay detonators which give a suitable time gap of 100 milli-second or so, between successive rounds.

In a drift of 3.8 m × 3 m cross-section in sand stone the consumption of explosives per metre advance is 10 to 12 kg. It is less for shales. An average drift of 10 to 11 sq. metre cross-section in sandstones may require 30-40 shots with a total charge of 18 to 22 kg for a pull of 1.5 m.

Mucking

Mucking means removal of blasted rock and cleaning the face. Where economics does not permit of investment on equipment suited for high speed, manual laoding is carried out. The muck is packed in old galleries, not in use. When tubs are supplied at the face and men are restricted in their movement, an output of 2-4 m^3 (broken) per shift per man may be expected. A double track to the face with steel sheet to traverse the tubs from empty to full track is the simplest arrangement.

Mechanical loading is requisite when high speed and lower cost are to be achieved. The slusher loader, also called "scraper", and rocker shovel are more associated with drivage of drifts than other types of loaders.

A slusher (Fig. 7.10) is simply a main and tail rope haulage mounted on a wheeled carraige and having a ram or slide at one end and a hopper for discharge of material at the other end. A return pulley for the tail rope is fixed at the face of the stone drift as shown in Fig. 7.10. The scraper bucket is open at the top and bottom and may be of V shape construction. It has a capacity of 0.5 to 1.5 m^3. Slusher haulages underground are often used in metal mines and in stone drifts in coal mines. The slusher is clamped to rails during operation. It has a capacity of 30 to 40 te per hour and can operate on, gradients upto 1 in 3 and even steeper, but not on soft floor.

A rocker shovel has compressed air powered bucket which is pushed into the muck, and then made to scoop the material with an upward-and-over motion for discharge into a tub or conveyor behind the loader. It may be mounted on crawlers, pneumatic tyres or wheels for travel on rails. A crawler mounted rocker shovel is best for a dipping drift (maxm. gradient 1 in 4) as the shovel after loading, has to travel some distance for unloading into tubs. In a rising drift, at steep gradient a tyred rocker shovel, when loaded, raises problems of braking and control and a crawler mounted shovel has to be preferred. A rocker shovel requires adequate height of the drift which may not be possible in many cases. (Fig. 7.9).

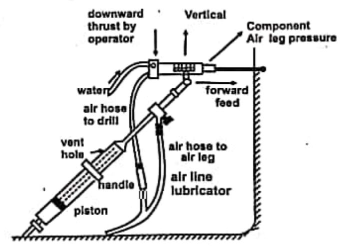


Fig. 7.8 Air leg

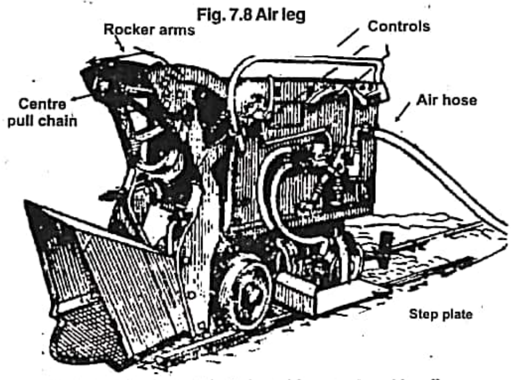


Fig. 7.9 A typical rocker shovel for overhead loading

Some of the machines used for mucking are now-a-days remote controlled. Such machines operate normally 4-6 m from a chute; e.g. Cavo 511 manufactured by Atlas Copco.

Ventilation

Drifting is an unproductive and much costlier work than drivage of roads in coal. In any case, it is not immediately paying. It is therefore a standard practice to drive only one drift, and not parallel drifts at close interval. The ventilation is always provided by an auxiliary forcing or exhaust fan. A small drift, maximum length 300 m, can be ventilated by a single fan, either forcing or exhausting. To avoid recirculation of air by the fan it should be placed in the path of the main ventilating air current (from which a branch is taken to the drift) at least 5 m away from the nearest corner of the drift. In long drifts, however, a combination of exhaust and forcing fan is preferred for efficient ventilation and speedy removal of fumes after blasting. Such fumes are drawn direct into the exhaust tube and the drift-air is unpolluted for persons travelling and working in drift. The mouth of the exhaust tube should be within 2 m for the face and the forcing fan should be installed at least 10 m outbye the end of the exhaust tube to avoid recirculation of air (Fig. 7.12). The increasing use of diesel engines underground makes tremendous demand on ventilation system that have to clear out dangerous or irritating exhaust gases. 200 m^3/min of air is the accepted requirement of normal size drifts (say 4.2 m × 2 m) and where jumbos or drill rigs are used the standard requirement is estimated at 42 m^3/min per drill. Exhaust or compressed air operated machines largely help in ventilation. I.L.O. (Internation Labour Orgnisation) recommends a quantity of 0.175 m^3/s per m^2 of drift face area. In very hot faces upto 0.75 m^3/s have been used. DGMS circular no. 30 of 1973 requires the ventilation of drives exceeding 50 m in length to be such as to dilute the nitrous fumes produced by blating to 5 p.p.m. and CO to 50 p.p.m. within a period of 5 minutes. This is however a stringent requirement in case of long tunnels. It would be more desirable in such cases to increase the permissible time of dilution or use an overlapping system of ventilation.

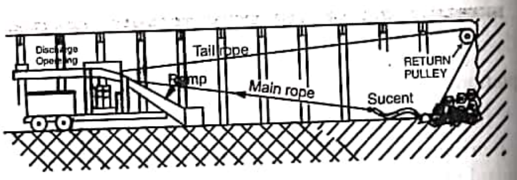


Fig. 7.10 A slusher

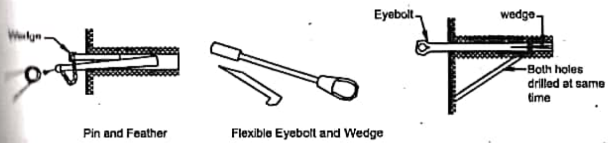


Fig. 7.11 Methods of fixing a slusher sheave

(The inclined hole may be blasted to withdraw the wedge)

Working conditions in a drift are arduous; *e.g.* high temperature if ventilation is inadequate, rock dust produced by drills resulting into silicosis if inhaled over long periods in the absence of wet drilling arrangements, the foul and oil-charged exhausts of pneumatic machines. Noise of the drills is deafening as the sound echoes back and forth in the confined space and diesel engines add to the uproar.

The arrangements for support, drainage, and transport are nearly the same as those followed when driving roads in coal.

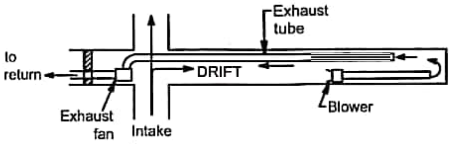


Fig. 7.12 Ventilation arrangement in a long stone drift

Drifting At A Modern Colliery

Sudamdih colliery had to undertake heavy drifiting work because of the system of horizon mining involving nearly 12.5 km of drifts in sand stone, shale and other coal maesures. The drift in south was cut at 400 m horizon was nearly horizontal and the drivage was done in the following manner.

(a)	Excavated area	...	4.8 m × 3.5 m
(b)	Completed area	...	4.3 m × 3.1 m

The workers engaged on drifting were on the principle of "All men of all jobs". Normally a team of 8 workers per shift for a drift 4.2 m wide managed all the jobs, such as drilling, blasting, mucking, setting supports, extension of tracks, extension of ventilation and compressed air pipes, trailing cables, etc.

Drilling

At the face two airleg mounted jack hammers were in actual use; one drill was in reserve and one was under maintenance at the surface. The weight of a drill was 25 kg with air consumption of 2.8 m^3 of free air per min. at 6 kg/cm^2. The air leg weighed 22 kg and had an effective feed length of 1300 mm. Flexible hoses were 25 mm bore for air and 19 mm bore for water (wet drilling). Hexagonal (22 mm) integral drill steels, 1.6 m long were used. The life of drill steel varied from 300 to 600 m. Four workers operated two drills and total 50 shotholes with 1.50 m effective length were drilled on wedge pattern.

Charging, stemming and firing

150-gramme cartridges of Ajax 'G' were used. Delay detonators with delays from No. 5 were employed and all the holes fired in a single round using multishot exploder. The details on blasting are as below :

1. Size of the drift :	4.8 m × 3.5 m
2. No. of holes drilled :	50-55 approx.
3. Charge per hole (Average)	600 g
4. Total charge per round :	25-30 kg.
5. No. of delays used :	0 to 5
6. Pull per round	: 1.4 to 1.5 m.
7. Surface area of the drift	: 16.8 m^3
8. Volume of excavated rock in one round	: 37.6 m^3 broken
9. Tonnage of rock per round	: 48 tonnes
10. Rock yield per kg of explosive	: 1.6 te approx.

The mechanical loader of overhead type moving on rails at the end of a temporary track was used. It unloaded into tubs of 1 m^3 capacity. A small tugger hualge was used for hualing the empties inbye.

The mechanical loader was compressed air operated consuming 5 m^3 air/min. The compressor was installed at the surface and supplied air through flanged pipes of 150 mm bore, 6 m long pieces. Later on it was replaced by a portable compressor in the drift.

Support

Import excavations were lined with concrete blocks but the common arrangement in the drift was to support the roof by steel arches of 60 lb. section rails. The steel arch was in two segments and foot plate 160 mm × 160 mm × 10 mm was welded to each segment. The two segments were joined by fish plates.

Ventilation

Ventilation was provided by an axial flow type auxiliary fan, capacity 270 m^3/min, 7 kw power consumption, 95 mm w.g. Fans were installed in series at every 100 m intervals. The ventilation ducting was 1.5 mm thick M.S. sheets, 580 mm dia. in lengths of 3 m for ease of handling and transport.

Overall cycle time

On an average the time required by the composite face crew of 8 to complete one cycle of advance of 1.4 m was approximately 17 hours; the break-up of this time into major operational groups is shown in the following table.

Operation	Elaspsed time (Minutes)
1. Drilling - preperation, 50 nos. 1.5 m hole dilling, and mop up operation	200
2. Shorfiring - priming, charging, stemming, connecting up, walk out, fire, waiting for fumes to clear, return to face, inspection and dressing :	120
3. Loading - preparation, loading out about 45 cars and cleaning up :	210
4. Setting supports - transporting arches, setting up and lagging :	260
5. Miscellaneous ancillary activities, shift change-over time, enforced idle time and other lost time -	230
Total cycle time : 1020 minutes.	

With the total cycle time of 17 hours, 35 cycles could be accomplished in 25 working days in a month and the monthly advance was about 50 metres.

Laser Rays For Drift Drivage

For maintaining gradient and direction of drift, the theodolite is a reliable instrument in the hands of the surveyor. When driving a drift over a long distance, instruments based on laser rays have been used in some mines in recent years. At Zawar lead-zinc mines in Rajasthan, a tunnel 3.65 m × 3.05 m in cross-section, 2275 m long, with a gradient of 1:250 was driven using laser rays for alignment. The instrument known as Laser adjustment system, LAS, does not eliminate the use of theodolite, but replace it to a great extent after initial use of theodolite in drift drivage.

Two target plates, called the first target plate and second target plate, are used in conjunction with laser insrument. The first target plate has cross marks but no hole in it. The second target plate has cross marks along with a small hole in the centre. The instrument can be fixed to the roof bolts or to the wall of the drift with the help of suitable brackets available from the manufacturers and nuts and bolts.

When driving the drift, the initial distance of about 10 m is driven by giving the alignment and gradient with the help of theodolite. Thereafter for installating the laser system on the brackets, the procedure is as follows :

1. In the drift set a theodolite such that height of its telescope above ground level in nearly the same as the proposed laser system installation. Set the telescope of the theodolite in the desired direction.
2. Fix the target plate (first target plate) away from the theodolite so that the centre of the target plate coincides with the centre of the cross hairs of the theodolite. To facilitate quick positioning the target plate is mounted on a suitable amount which can be fixed on wall roof of the drift.
3. Fix another target plate (second target plate with hole in the centre) on the wall/roof of the drift between the theodolite and the first target plate in such a way that the cross hairs of the theodolite, the hole of the second target plate and the centre of first target plate, all are in one line. Hence, the line through the centre of the two target plates is the required direction of alignment and gradient.

4. Now replace the theodolite with the laser instrument and fix it on the wall/roof of the drift with the help of a bracket. The laser is switched on and so adjusted that the red beam of laser rays passes through the hole of second target plate and falls at the centre of the first target plate. In this position the laser follows the part of direction set by the theodolite.

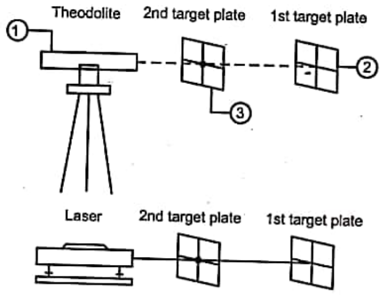


Fig. 7.13 Fixing up laser instrument in a drift

Fig. 7.14 Jyoti Laser Alignment system

Courtesy of Jyoti Ltd.

As the drift advances only the first target plate is shifted from time to time and fixed on brackets in the wall/roof so that its centre is in the line decided by the laser beam and the centre of second target plate.

Drivage Or Roads In Metalliferous Mines In Rock/Ore

Level roads, drifts, drives or cross-cuts are driven in the same manner as described for drivage of stone drifts in a coal mine. Drivage of winzes is not a common practice owing to difficulties mainly of drainage, transport, *etc.* specially in steeply dipping veins. Drivage of raises is however a normal operation in every metalliferous mine. Raising operations have undergone much improvement over the conventional methods of raising by drilling-blasting-mucking. They have been mechanised to give fast progress with less manpower. The recent improvements include Alimak Raise Climber, Raise Bores (*e.g.* Atlas Copco Jarva, Robbins Raise Borer), etc. They are described in detail in 2nd volume of this series.

Tunneling And Road Heading Machines

The modern methods of fast drivage of tunnels and roads in coal as well as soft rock like sandstone employ machines of which the following have proved popular in the Indian Mining Industry in recent years.

1. 2000 DL of Atlas Copco Co.
2. AM-50 Alpine Miner of Voest Alpine Co.
3. Dintheader of Dosco Engineering Co.

Of these, the first two have been described in detail in Volume 3 of this series. Some of these machines are equipped with cutter chains and some others, with cutter booms. The earlier machines in the cutter boom category had only single booms, but machines with twin booms are also available in the market in recent years. The inseam drivages are normally rectangular and ideally suited to all types of heading machines. In deep mines, however, the roadway supports are often steel arches. The arched profile generally needs a heading machine fitted with a boom. Such machines can cut the road for a desired profile of raodway.

The road header machines are capable of cutting rock which is not very hard, as well as coal.

Among the machines marketed by DOSCO Engineering Co., are the light duty, medium duty and heavy duty heading machines. The **Road header MK 2A** manufactured by DOSCO is a medium duty machines weighing only 23 te. The boom has axially rotating cutting heads. A lighter machine for road heading manufactured by DOSCO is known as Dintheader and weights nearly 17 te. The heavy duty machine is MK 3 having nearly 150 kW power. At Selby colliery of National Coal Board in UK, in a tunnel 4.2 m wide × 3.8 m high the average progress of heading with MK 3 machine was 70 m per week (5-days week). The tunnel was completely lined by steel arches.

The **Dintheader** has been used at Pathekhera Coal Mine of WCL. It has a cutting jib 1.72 m wide, *i.e.* in excess of the width of the machine. It weighs 16 te and exerts a ground, pressure of 0.125 MN/m². The tracks are hydraulically operated giving a working speed of 0.03 m/sec and a flitting speed of 0.14 m/sec.

The cutting cycle is started by sumping at floor level and then raising the jib to take the cut. The majority of coal is loaded down the cutter jib, any coal remaining being loaded on the return pass to the floor. Coal passes from the jib into the centrally mounted straplink hydraulically powered scraper conveyor to the secondary conveyor. This can be of two types; either a bridge belt conveyor with upto 16.0 m overlap or a short cantilever mounted swivel conveyor 3.2 m in length. The cutting action of the Dintheader, being based on radially rotating picks, limits its application to coals and rocks of less than 600 kg/cm² unconfined compressive strength; however, this is sufficeint for the majority of coal seams and immediately adjacent associated strata.

The progress with a Dintheader may be upto 10 m/shift or 100 m/weak.

Where the designed shape of roadway is square or rectangular, not more than 2.4 m in height, the Dintheader is a suitable machine.

✧ QUESTIONS ✧

1. How will you drive a pair of dip headings in coal where the transport is by rope haulage and skidmounted coal cutters are used with coal drills ?

2. A stone drift is to be driven for a length of 60 m to prove coal on the other side of a fault plane. Size of the drift is 3 m by 2.4 m. Give the method of

   (a) maintaing gradient and direction of drift, (b) arrangement of shot holes, (c) type of drill and drill bits to be used, (d) method of dealing with debris.

3. What should be the size of coal pillar at a depth of (a) 150 m (b) 350 m if coal is obtained by solid blasting ? In which cases should the size of coal pillar be large ?

4. Why is more emphasis given on drivage of main dip ? What are the various ways of dealing with water in the main dip for proper advance?

5. What is the meaning of the following terms ?

(a) Stage pump	*(b) Crosscut*
(c) Face	*(d) Plat*
(e) Winze	*(f) Slope*
(g) Slusher	*(h) Jumbo*

✧ ✧ ✧

Chapter - 8

Explosives, Accessories And Blasting

Explosives are used in underground mines and quarries to break coal and other rocks. An explosive is a solid or a liquid substance or mixture of substances which change themselves instantaneously into a large volume of gases at high temperature and pressure when a flame, heat or sudden shock (detonation) is applied to it. The high pressure that is built up is capable of doing the work of breaking rock. Detonation is a process of giving sufficiently violent shock to he explosive to bring about an almost instantaneous rearrangement of atoms. The chemical energy in an explosive is released and converted suddenly into heat and mechanical energy when heat, flame or detonation is applied and this conversion is due to a chemical reaction which is essential process of oxidation. An explosive, by virtue of its constituents, contains enough oxygen necessary for complete oxidation. A commercial explosive contains, apart from the explosive substance or explosive mixture. The following materials :

1. Combustible matter such as wood meal, fibre, sulphur, charcoal, etc.
2. Oxidation agents, such as sodium nitrate, ammonium nitrate, potassium nitrate, etc.
3. Stabilizers such as magnesium and calcium carbonates.
4. Antisetting agents to prevent caking of salts.
5. Sensitisers, like metallic powders.

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The following properties of an explosive are of interest to its user :

1. Strength : This is a measure of the amount of energy released by an explosive during blasting and hence its ability to do useful work. The relative strength or power of an explosive is given by the term weight strength in the case of explosives manufactured by ICI India Limited and a few other explosive manufacturers. The weight strength, in the case of ICI explosives, indicates the strength of any weight of explosive compared with the same weight of Blasting Gelatine which is taken as standard because it is the most powerful commercial explosive manufactured by ICI. The weight strength of Blasting Gelatine is 100. At present, Blasting Gelatine is not in the regular manufacturing range of ICI India Limited. The Ballistic mortar is calibrated initially with standard Blasting Geletine and subsequently the weight strengths of other explosives are determined with respect to the above calibration.

Indian Oxygen Ltd., and Indian Detonators Ltd. do not use the term weight strength to indicate the relative strength of their explosives.

2. Velocity of detonation : It is the rate at which the detonation wave passes through a column of explosive and this is of considerable importance as the shock energy of detonation increases rapidly with this velocity. Most of the high explosives, permitted explosives and slurry explosives used in the mines have a velocity of detonation ranging between 2500 and 5000 metres per second. For high explosives which are used as boosters, the V.O.D. is high, e.g. O.C.G. - 6000 m/s; Primer - 7000 m/s.

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3. Density : The density is important when selecting an explosive for a particular use. With a high density explosive the energy of the shot is concentrated - a desirable feature in tunnelling and mining operations in hard ground. On the other hand when the output of lump coal from mine is important, it is advisable to use a low density explosive, which distributes the energy along the shot hole.

4. Water resistance : Explosives differ widely in resistance to water and moisture penetration. Some explosives deteriorate rapidly under wet conditions, but others are designed to stand water long enough to enable the work to be done. When blasting is to be performed under wet conditions a gelatinous or slurry explosive should be used. The higher the nitroglycerine content of an explosive, the better its water resistance properties.

5. Sensitivity : An explosive is required to be insensitive to normal handling, shock and friction, but must remain sufficiently sensitive to be satisfactorily detonated, and capable of propagating satisfactorily, cartridge to cartridge and even over short gaps such as may occur in practice.

6. Fume characteristic : Explosives which are to be used where ventilation is restricted must produced a minimum of harmful gases in the products of detonation. Slurry explosives and AN based explosives are preferable to the NG based ones.

7. Legal permission : Only permitted explosives of proper type should be used in underground coal mines.

Explosives are grouped in two types depending upon the speed with which explosive effects is produced.

1. Low explosives. 2. High explosives.

The terms "low explosives" and "high explosives" should not be confused with the terms "low density explosives" and "high density explosives".

Gunpowder is a common example of a low explosive and ammonium nitrate, nitroglycerine, T.N.T., special gelatine, slurry explosives, etc. are high explosive. When a low explosive is blasted the process of oxidation of the constituent substances is propagated by rapid combustion from particle through the mass of the explosive and the effect of explosion is relatively low. A low explosive is fired by ignition or a flame. High explosive always contains an ingredient which is explosive in itself, at least when sensitised by proper means. A high explosive explodes when a violent shock is applied to it with the help of a detonator; the process of oxidation does not proceed from particle to particle, but is instantaneous and the constituents react with high velocity. High explosives therefore produce a shattering effect.

Gunpowder : This is the earliest known explosive and contains the following constituents (by weight, approx.)

Charcoal 15%, Sulphur 10%, Potassium Nitrate 75%.

None of the constituents is explosive by itself. Gunpowder is cheap, stable, safe to handle and it does not adversely affect the roof in underground mines as its action has a heaving effect. It is easily manufactured and is available from local supply contractors in many mining localities. It is however, not used on a large scale as it losses its explosive power when damp and it is not as strong as other high explosives. Moreover, during blasting, it produces flame of long, duration and the burning practicles are liable to remain in contact with the surrounding atmosphere for some time. For these reasons gunpowder is not used in wet place and in underground coal mines. The explosive is fired by safety fuse.

Important constituents of high explosives which confer explosive properties on them are nitroglycerine, ammonium nitrate, collodion cotton and T.N.T.

Nitroglycerine : It is an oily fluid with sp. gr. of 1.6 and freezing point at 13°C. It is insoluble in water and is very sensitive to explosion by shock of any nature. The sensitivity to shock increases when nitroglycerine freezes. To render it suitable for industrial use, it must either be absorbed by some inert material or be gelatinised. NG based explosives are available in 3 consistencies; gelatinous, semi-gelatinous and powdery. All explosives containing NG have a highly shattering effect and they produce fumes which cause headache after long exposure. Explosives containing NG are liable to freeze when the temperature falls to 8° or less and are then more sensitive to detonation by friction and impact. To avoid this, a low freezing agent, usually di-nitro-glycol which is itself as powerful an explosive as NG, is used. Low freezing explosives are designated by such prefix as "Polar", e.g. Polar viking, Polar special gelatine, etc.

Ammonium nitrate (NH_4NO_3) : It is a white hygroscopic salt, very soluble in water and is comparatively very safe to handle. When it is detonated it is, however, a powerful explosive. Though ammonium nitrate is more powerful than the low explosives it is not as powerful as N.G. and it is difficult to denote it by itself with the help of a detonator, but it can be detonated by a booster of high explosive.

To use ammonium nitrate as an explosive, it should be mixed with diesel oil, N.G. or T.N.T. Ammonium nitrate is an interesting compound, in that it is a high explosive, an oxidising agent, and a cooling agent at one and the same time. Prilled ammonium nitrate of fertiliser grade mixed with diesel oil is used for large dia. holes in quarries.

Ammonium nitrate does not occur in nature and it is prepared by reacting ammonia gas with nitric acid. When detonated by extreme shock NH_4NO_3 decomposes according to the equation.

$$2NH_4NO_3 = 4H_2O + O_2 + 2N_2 + 346 \text{ kcal/gm.}$$

This property comes into play in the utilisation of NH_4NO_3 as an explosive.

Collodion cotton : A reaction between cellulose compounds and nitric acid yields collodion cotton, a high explosive. To render it safe to handle it is gelatinised.

T.N.T. : Reaction between nitric acid and benzene or toluence compounds yields Trinitrotolune (T.N.T.) which is highly explosive.

Most of the high explosives can be blasted with the help of detonators and are said to be cap-sensitive. A detonator has to explosive or kept in intimate contact with it. Such cap-sensitive explosive can be blasted with the help of a detonating fuse like cordtex also if the latter is in close contact. The detonating fuse itself needs to be detonated by a detonator. An explosive which is not cap-sensitive is blasted by keeping it in close contact with a booster which itself needs to be initiated by a detonator or a detonating fuse. A non cap-sensitive explosive is also known as "booster sensitive". An explosive used in underground mines has to be cap-sensitive but the explosive used in quarries may be cap sensitive or booster sensitive.

AN-diesel explosives are not cap-sensitive and have to be blasted by keeping them in intimate contact with an explosive (booster) which itself is detonated with the help of a detonator. Some of the slurry explosives like energel, supergel and powerflow 1, 2, 3, are not cap-sensitive while most others are.

Explosives with NG base which have to be used in watery places should be of gelatinous or semi-gelatinous consistency but not powdery and they should sink in water i.e. they should have sp. gr. of more than 1.0.

Booster : For effective detonation of some slurry explosives and AN-FO mixture such as GN-1, use of a high detonation-velocity booster is necessary. ICI India Ltd. manufactures a booster with the trade name "Primex" which is a mixture of PETN and TNT. It is water resistant, has a velocity of detonation of 7,000 m/sec, weight strength as 82 and it can be detonated by a detonating fuse or, a detonator. The booster manufactured by IDL Chemicals Ltd. is marketed by the trade name "Pentolite" which has a sp. gr. of 1.55 to 1.61. Compared with normal explosives boosters are quite costly. Primex is cast in cylindrical pellets provided with two logitudinal holes for threading on to a down line of detonating fuse. For priming, a detonating fuse is threaded through the two holes in the Primex pellet and a knot tied at the top. This assembly is then inserted into a cartridge of slurry and its mouth re-tied by a wire. After lowering the primer cartridge, are freely dropped down the hole.

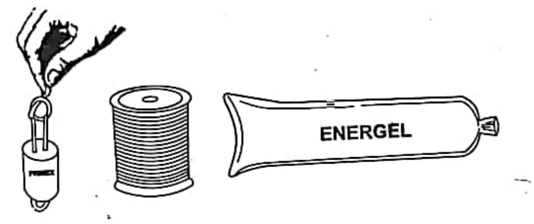


Fig. 8.1 Primex and Pentolite boosters; a cartridge of slurry explosives

A cast booster is not a substitute for the explosive charge; it may be compared with a very powerful detonator of large size and is preferred for deep large dia. blast-holes in opencast mines. During use a cast booster is knotted to the detonating fuse for placement at the bottom of the blast-hole and additional boosters are threaded in the same detonating fuse so that their positions coincide with the level of hard rock when the fuse is in position in the blast-hole. After lowering the booster by the detonating fuse, AN-based site-mixed or plant-mixed slurry is poured or pumped in the hole. As the slurry is pumped in, special ingredients are added to it at intervals to cover the hard-rock portion.

Gelatinous NG-based explosives having high NG content like OCG or special gelatine can be used as a booster or primer explosive. Pentolite boosters manufactured by IDL Chemicals Ltd. do not contain NG or other headache - causing ingredients. They are equipped with 2 holes for easy initiation with detonating cord or detonators. A PETN based booster in a shot-hole occupies only 1 to 2% of the total explosive charge but a high explosive like OCG, if used as a booster, occupies 15-20% of the total explosive charge.

AN-Fuel Oil Explosives (ANFO)

Ammonium nitrate, mixed with diesel oil, is used on a large scale for blasting in the quarries of coal and metal mines. The most effective and oxygen-balanced explosive mixture is one with 5 to 6% of diesel oil (by wt.) It has a sp. gr. of 0.8 to 1.0, wt. strength of 75-80 and velocity of detonation at 3500 m/sec. In the dry season, 7 litres diesel for 100 kg of AN suffice but in wet season, the quantity should be increased to 9 litres. Diesel oil in excess of 8% tends to lower the sensitivity of ANFO to initiation. The mixture causes irritation of the skin and the workers should, therefore, wear gum boots and rubber hand gloves. The mixing should be done with wooden shovels avoiding contact with iron. The mixture is safe to handle and without formation into cartridges can be mechanically loaded into blast-holdes. Where ANFO consumption is heavy, stationary ANFO mixer similar to the concrete mixer, may be placed at a centrally selected site. In case of a pneumatic ANFO loader, an electric detonator should not be used unless steps are taken to prevent premature initiation due to static electricity.

As the mixture cannot be initiated in the normal way by a detonator it is necessary to prime it with a small quantity of O.C.G. or a booster. It is a good practice to use high explosive primer cartridges at the top as well as at the bottom of the hole explosives are difficult to sink in water due to low density of AN, and should preferably not be used in watery holes. If however, AN-fuel oil mixture has to be used n watery holes it should be packed in polythene bags and forced down the hole with the weight of a high explosive and the stemming above it. Holes of 62 mm dia. and above are considered economic for use of AN-FO explosives.

An increase in blast-hole dia. beyond 300 mm decreases the sensitivity to initiation of ANFO explosives. With above 4 percent water in ANFO, the velocity of detonation decreases sharply and the mixture with 9 percent or more water cannot be detonated. When using ANFO, it is essential to have uniform mixing of ammonium nitrate and fuel oil. If the ammonium nitrate is not of adequate porous quality, it may separate from fuel oil resulting in inferior performance.

AN-FO explosive cannot be initiated direct by No. 6 detonator. It can, however, be blasted by a detonating fuse which needs no. 6 detonator for initiation. It may be intitiated by no. 8 detonator which is not much used in mining practice.

Slury Explosives

The slurry explosives are with jelly like consistency and are water gels. (The water-gel is a mixture of an oxidiser and fuel sensitiser in an aqueous medium, thickened with a gum and gelled with cross linking agent). In the case of a permitted slurry, a coolant is added to reduce incendivity.

The first commercial slurry explosive developed by Dr. Melvile Cook in U.S.A. in 1957 consisted of TNT, AN and water in the ratio of 20 : 65 : 15. To this traces of chemicals for gelling and crosslinking were added to stabilise the homogeneity of the mixture. In subsequent years the manufactures developed slurries with AN as the main ingredient and using variety of sensitisers and fuels. The addition of metallic powder to the slurry enables the explosives to reach very high strength. Some of the common ingredients are :

Oxidisers	: ammonium, sodium or calcium nitrates.
Cross-linking agents	: potassium or sodium dichromates, antimony or boron compounds
Gelling agents	: starch
Fuel sensitisers	: TNT, PETN, pentolite - (all explosives). aluminum, sugar, urea, paraffin, hexamine, ethylene glycol, woodpulp - (all non-explosives)

The slurry explosive has a sp. gr. more than 1 and like ANFO can be poured directly into watery holes. They are also available in the form of cartridges with plastic or polythene wrapper and some (permitted type) can be used in underground coal mines. Such slurry explosives for use in underground coal mines have to be cap-sensitive and approved by the DGMS. The slurry explosive is highly water resistant. In the quarries holes of diameter 62 mm and above are economical for use of slurry, just as for ANFO, if it has to be poured into blasthole.

The components required for ANFO and slurry explosives may be mixed at a plant away from the blasting site or at the blasting site itself. In the case of PMS (plant mixed slurry) system, the explosive is loaded into special tankers and from these tankers, the slurry is pumped directly into the blast-hole.

Where the volume of blasting is high enough to justify cost of transporting trucks, ICI has designed a slurry pump truck which is capable of pumping Powerflow slurry directly into the blastholes. The system has been extensively tested in overburden strata in opencast coal mines as well as in hard iron ore formations with very satisfactory performance. The slurry remains in intimate contact with the walls of blast holes and this results in effective utilisation of the explosive. Packaged explosive products are at least 15% less effective than the corresponding bulk explosive products due to less perfect blast hole contact.

Unlike cartridges slurries, pumpable slurries can be tailored to have the appropriate density depending on strata conditions. In the case of site mixed slurry system (SMS) only nonexplosive ingredients are stored at a warehouse and transported to the blasting site in a specially designed pump-truck.

In the semi-gelled condition, pumpable slurries can be stored for about 2 weeks in the truck without any adverse effect on performance. The slurry is suitably gelled (cross linked) at the time of manufacture and incorporation of an additional gelling (cross linking agent at the time of loading into blast-holes can prevent the possibility of explosives flowing into the cracks and fissures in the blast-hole.

In case of SMS system for blasting, one pump truck can charge nearly 25000 kgf in one shift. A small team of 5-6 professionally trained persons can load 50,000-60,000 kgf of explosives into a large number of blast holes in a single working shift.

The principal ingredients in I.D.L.'s permitted slurry explsoives are ammonium nitrate, water, sugar, aluminum (as fuel sensitiser), sodium chloride, sodium nitrate and gum.

Slurry explosives will not explode accidentally if dropped, or from shovel impact, or even when involved in a fire. They have low non-toxic fumes and do not cause headache. Exudation from an unstable water-gel explosive is harmless since in most cases it is nothing but water containing some dissolved inorganic salts. On the other hand

NG exudation in NG-based explosives is hazardous; a spark or violent shock can set it off. Non-cap-sensitive slurry explosives magazine, but permitted slurry explosives have to be stored in a licensed magazine. Shelf life of slurry explosives manufactured by most of the companies is usually one year. The best performance of slurry explosives is obtained within four months from the date of manufacture.

The standard safety tests for slurry explosives are (1) Burning test, (2) Friction and impact test (Torpedo test), (3) Impact sensitivity by hammer fall test, (4) Rifle bullet test, (5) Sensitivity of flame head (fuse) test, etc.

Emulsion Explosives

An emulsion is an intimate mixture of two liquids that do not dissolve in each other. In more technical terms, an emulsion is described as a two-phase system in which an inner or dispersed phase is distributed in an outer or continuous phase. Emulsions have for many years, contributed to our daily lives in such products as insecticides, photographic films and papers and cosmetics.

The *unique feature* of an emulsion explosive is that both the oxidizer and the fuel are liquids. The unique properties of emulsion explosives are due to the minute size of the nitrate solution droplets and their tight compaction within the continuous fuel phase.

The emulsion slurry contains AN solution at high temperature, combined with diesel oil, and an emulsifier which is passed through a fast moving blender. The blender mixes the ingredients into pumpable emulsion slurries with grease like consistency. The resultant emulsion slurry is now made up of microscopic droplets of AN, surrounded by fuel oil film and artificially created air bubble know as *microspheres* which makes the emulsions detonable. There is not sensitisation by aluminum or explosive sensitiser but by mere air bubbles. Thus, there is maximum energy generation because of intimate contact of oxidiser and fuel. In small diameter products, glass microbaloons and perlites are used to maintain sensitisation by aeration.

Emulsion explosives depend entirely on the presence of voids for initation and propagation. A change in the amount of voids effects a change in density. It is convenient and useful to relate properties to density and to consider voids and density adjustors.

Slurry explosives require thickners and gelling agents to prevent segregation, to provide water resistance and to control losses through cracks and fissures. Emulsion explosives cannot be gelled or cross linked. They do not have the gel structure that characterises all slurry explosives. Velocity of detonation is a good indicator of reaction efficiency and is very dependent on particle size.

Explosive	*size*	*Form*	*VOD (km/sec)*
ANFO	2.000 mm	All solid	3.2
Slurry	0.200 mm	solid/liquid	3.3
Emulsion	0.001 mm	Liquid	5.0-6.0

Emulsion explosives are highly water resistant. They are more fluid than slurry explosives and therefore create problems when loading a blast hole with fissures or cracks. They lack the strong cross-linked gel that characterizes the TNT-sanitized slurry explosives.

Emulsion slurries are claimed to have lower ingredient cost, higher density, higher VOD and higher energy conversion and resultant bore-hole pressure as compared to watergels/slurries. Therefore in low diameter sector they are fast replacing slurries and watergels in USA. The emulsions are also claimed to be sensitive even at smaller diameter holes, less than 25 mm. In addition, they are easily pumpable without affecting their quality. Thus, emulsion slurries are hoped to be the low-cost, less hazardous replacement of NG based small diameter products.

Heavy ANFO

The latest development of 1980's had been the use of emulsion slurries mixed with different proportion of ANFO to give water resistant and higher density mixture which are named as Heavy ANFO or HANFO. Thus, the emulsion to ANFO ratios can be from 20:80 to 50:50, depending on the severity of watery conditions and need of stronger blast energy. One of the US manufacturers has successfully developed Heavy ANFO, using NCN slurry with ANFO. Since the air spaces between AN prills are filled by emulsion, Heavy ANFO gives the advantage of lower cost like ANFOR with higher density, hoger energy and better water resistance than ANFO or AN. Thus the mixture can have bulk density from 1.10 to 1.25 gm/cc (compared to ANFO 0.8 gm/cc) and bulk strength almost 45% more than ANFO. The high energy ANFO through HANFO system or slurry concentrates system allows expansion of drilling pattern, thereby reducing drilling costs.

Permitted Explosives

All explosives create heat and some flame when fired. To avoid explosion of gas or coal dust in an underground coal mine, it is essential that the heat and flame produced by an explosive should be incapable of igniting the gas or coal dust. For this reason explsoives for use in underground coal mines are approved by the CMRS and DGMS after certain tests.

A permitted explosive is one which has been subjected to stringent test by the CMRS and found to be incapable of igniting firedamp or coal dust when used in charges up to a specified weight. Each cartridge of permitted explosive is contained in a wrapper clearly marked with the letter P.

The safety of "Permitted Explosive" depends primarily upon (i) low temperature, and (ii) duration of flame produced, the flame lasting only 1/1,000th second. A cooling agent like sodium chloride, potassium chloride, sodium fluoride, etc. is an essential constituent of a permitted explosive and the main constituents are NG (only in some explosives), ammonium nitrate, sodium chloride and in non-slurry type permitted explosives, absorbing materials.

Permitted explosives are categorised in the following groups and the group to which a cartridge of explosive belongs is printed on the wrapper of the cartridge.

Group :

P1 – Unsheathed explosives, such as Ajax G, Viking G and slurry explosives like Godyne.

P2 – Sheathed explosives.

P3 – Equivalent sheathed explosives (Eq. S) e.g. Unisax G permadyne.

P4 – Explosives approved for special purposes, such as for delay firing, firing in ripping, etc; hitherto known as ultra- safe explosives. These are not produced in India. British examples are; carribel, Nobel's explosive 1235.

P5 – Off-the-solid explosives (for solid blasting) *e.g.* Soligex, slurry explosive Pentadyne.

Sheated explosives (P2 group) and P4 explosives are not manufactured in India and are not used in our mines. The cartridge of a sheathed explosive is coated with a sheath of $NaHCO_3$ along its length (but not at ends). As a result of blasting, $NaHCO_3$ decomposes forming an extinctive blanket of CO_2 and H_2O round the explosive which is thus rendered more safe.

In equivalent sheathed explosives an inert material like mixture of Ammonium chloride and Soda nitrate is incorporated directly and uniformly throughout the composition of the explosive so that the explosion products render the explosives as safe as the sheathed explosives.

Explosives used in opencast mines are sometimes referred to as suitable for column charge, bottom charge, base charge, primer for booster. A column charge explosive is not cap-sensitive and it occupies nearly 80% of the column of explosives in a blasthole. Below it is the base charge (also called bottom charge, primer or booster) which is more powerful than the column charge. A primer and a booster charge is always cap-sensitive, depending upon the manufacturer.

Regarding the use of explosives the following guidelines should be noted.

1. Only permitted explosives approved by D.G.M.S. should be used in underground coal mines. Such explosives are always cap-sensitive.
2. Non-permitted explosives can be used in underground metalliferous mines and in the quarry mines of coal as well as metals and also for other surface applications.
3. In watery places, gelatinous and semi-gelatinous explosives should be used. But NG based powdery explosives are not recommended for blasting at watery places.
4. Slurry explosives are water-resistant and they can be used in watery holes but AN based slurry explosives with a specific gravity of one or less should not be used in watery holes.
5. Only explosives with a high velocity of detonation over 2500 m/s, should be used for blasting in hard rocks.

Non-cap-sensitive explosives should be blasted with the help of detonating fuse.

The liquid oxygen explosives are used in opencast mines and are described later.

TABLE 1

Explosives and fuses of various manufactures

Explosives, detonators and fuses manufactured by some leading companies for mining are listed below. Explosives and detonators, fuses, etc, used for seismic blasting are not included. Figure in bracket shows diameter, in mm. CS-Cap-sensitive, WS-weight strength relative to Blasting Gelatine as 100. Permitted explosives are for underground coal mines; they are 32 mm diameter and cap-sensitive. A permitted explosive, if not permitted for solid blasting, can be used for cut faces and in depillaring.

ICI India Ltd.

Explosives

Unless mentioned otherwise, the explosives are NG based.

Special gelatine of 90%, 80% and 60% designation (25,50), CS.

Gomia gelatine 50% (75, 83, 125, 175), CS.

Opencast gelignite (75, 83, 125, 175, 200), CS.

Hectorite (25, 32), CS.

Primex (23, 37, 50, 63), cast booster, CS.

NG/1 (83, 125, 175, 200), AN-based, non-CS.

Powerflow-1 (125, 175, 200), AN-based, slurry, non-CS.

Bulk slurry (100 mm and above), AN-based, non-CS.

Emulsion explosives

Powergel-C ,(83, 125, 175, 200), booster, CS.

Powergel-3 (83,125,175, 200), base charge; non-CS.

Powergel-2 (83,125,175, 200), base/column charge; non-CS.

Powergel-1 , column charge, non-CS.

Powergel-F (83, 125, 175, 200), column charge in fire areas, non-CS

Bulk emulsion (100 mm and above) AN-based, non-CS.

Permitted explosives

Ajax-G (P1), deg. 1 gassy

Unisax-G (P3); all gassy

Powergel P 101 (for coal, undercut faces) permitted for Deg. 1 gassy mine; emulsion explosives, CS.

Alphagex (P1); deg. 1 gassy

Permigex (P3); all gassy

Soligex (P5); all gassy; solid blasting

Solimax (P5); deg. 1 gassy; solid blasting.

Uniring for longhole blasting gallery technique.

Viking-G (P1); deg. 1 gassy

Detonators

These are in No. 6 and 8 strengths. Plain detonators are for use with safety fuse. Following are electric detonators :

Instantaneous (aluminum); for U/G coal mines these are of copper.

Long delay (aluminum or copper); delay numbers 0-10; delay interval 25 ms.

Carrick-nonincendive short delay (copper); delay numbers 0.6; delay interval 25 ms. These are used along with P-5 explosives.

Safety fuses

Blue sump, OCPS blue plastic (all with burning speed 100-120 sec/m); Ailsa with burning speed 120-145 sec/m;

Detonating fuse (initiation by No. 6 detonator)

Cordtex and Cordflex for commercial explosives.

Rigncord (for blasting gallery method in U/G coal mines); used in conjunction with Uniring.

Delay detonator-relays - delay interval 17 ms.

IDL Chemical Ltd.

Explosives

Unless otherwise mentioned, the explosives are AN-based, slurry type.

Aquadyne (83, 125, 175, 200), booster, CS.

Boeblat, (83, 125, 175, 200) booster, CS.

Energel (83, 125, 175, 200), column charge, non-CS.

Supergel (83, 125, 175, 200), base charg, non-CS.

Aquagel-200 (83, 125, 175, 200), column charge, non-CS.

Aquablast (83, 125, 175, 200), as booster, CS.

Superdyne (25, 40, 50), CS.

Pentolite (45, 50, 75), cast booster, with PETN/TNT, CS.

Permitted explosives

Godyne (P1), deg. 1 gassy;

Permadyne (P3), deg. 2 gassy

Pentadyne (P5), deg. 1 and 2 gassy; solid blasting

Emulsion explosives

Emulgel (83, 125, 175, 200), non-CS.

Emulprime (83, 125, 175, 200) as booster, CS.

Emulprime (83, 125, 175, 200) as booster, CS.

Supermix-100; pumpable; non-CS.

Emulking-100; pumpable; non-CS.

Emulking-200; pumpable; non-CS.

Detonators : NO. 6 type; electric

Istantaneous (aluminum); for U/G coal mines, these are of copper.

Short delay (aluminum); delay Nos. 0-10, delay interval 25-75 ms.

Long delay (aluminum); delay nos. 0-10; delay interval half sec.

Coal delay (copper); for solid blasting in deg. 1 and 2 gassy U/G Coal Mines; delay nos. 0-6, delay interval 25 ms.

Special ordinary detonators (aluminum); used with safety fuse and are of no. 8 strength.

Raydet - Non-electric initiating system; 3 mm plastic tubes with inside explosive coating; no. 8 strength detonator needed; delay no. 1-15.

Safety fuse - yellow colour.

Detonating fuses. For commercial explosives;

D-Cord I, coreload of PETN 6 gms/m; yellow;

D-Cord II, coreload of PETN 10 gms/m; orange.

Detonating relays or delay detonator relays - delay interval 17 and 25 ms.

Karnataka Explosives Ltd.

The explosives are marked by the prefix "Kel". They are watergel slurries in cartridge form and based on momethylamine nitrate, an organic chemical sensitizer.

Explosives

Kelvex-100 and Kelvex-200 (25-50), CS.

Kelvex-P (50), as booster; its charge should be 4-5% of the total ANFO charge; CS.

Kelvex-70 (83, 125, 200), as booster for opencast blasting where blast-holes were filled up water recorded 95°C temperature; CS.

Kelvan-extra, Kelvex-extra, Kelvex-500 (83,125,200), column charge; non-CS.

Kelvex-800, kelvex-600, (83, 125, 200), as primer/booster, CS.

Permitted Explosives

Kelvex-310, (P1) deg. 1 gassy

Kelvex-330, (P3) deg. 1 gassy

Kelvex-5, (P5); solid blasting in all degree gassy mines.

Emulsion Explosives

Kelem-800; booster, CS.

Kelbase; column charge, non-CS.

Kelcolem; column charge, non-CS.

IBP Co. Ltd.

Explosives are AN based slurry type, marked by the prefix INDO.

Explosives

Indomix (83), CS but a booster for initiation is preferred.

Indomite - 60 (25-50), CS.

Indomite - 80 (25), CS.

Indogel - 210 (83-200), non CS.

Indogel - 230 (83-200), non CS.

Indogel - 250 (83-200), non CS.

Indodoost (83-125), as booster, CS.

Detonating fuse :

Dynacord-6 with PETN coreload 6 gm/m.

Dynacord-10 with PETN coreload 10 g/m.

Apex Explosives (P) Ltd.,

Explosives are with prefix *Apex.*

Explosives

Apex gel 90, (25, 50, 83, 125, 200), AN-based slurry as booster, CS.

Apex gel 80, (83, 125, 200), AN slurry, base charge, non-CS.

Apex gel 70, (83, 125, 200), An slurry, column charge, non-CS.

Apex gel 70, (83, 125, 200), dry blasting agent, column charge, non-CS.

Pentex (50), cast booster, PETN & TNT-50 : 50 ; CS.

Apex Emulite (45, 62), emulsified cast explosive with PETN; CS.

Detonating fuse, core load of PETN, 10 gm/m, Post Box red colour.

Maharashtra Explosives Ltd.

Explosives carry a prefix a prefix Mex; All are AN-based slurry explosives.

Explosives

Mexboost (83, 125, 175, 200), as primer/booster, CS.

Mexprime (as above), as booster/bottom charge, for plaster shooting : CS

Mexprime XX (as above), as booster/bottom charge; CS.

Mexblast (125, 175, 200); base charge, non CS.

Mexan I (83, 125, 175, 200), column charge, non CS.

Mexan II (as above); column charge; non CS.

Mexcol (as above); column charge, non CS.

Mexpro (83 and above), column charge, non CS.

Mexgel (25, 32), CS.

Mexgel HI (25, 50); CS.

Permittends

Mexperm-1, deg. I gassy

Mexperm-3 deg. I gassy

Mexperm-5 deg. I and II gassy; solid blasting, CS.

Detonating fuse - Mexcord II, coreload 10 g/m; pink.

Tamilnadu Industrial Explosives Ltd.

All explosive carry the prefix Te. Excepting the slurries, which are AN based, all are NG based.

Explosives

Telgex LD (75, 83, 125), WS 78, primer, CS.

Telpow LD (75, 83, 125), WS-60 primer, CS.

Telgex-90 (25,32,50), WS-90, CS.

Telgex-80 (25, 32, 50), WS-78, CS.

Telgel (83, 125, 200), AN based slurry, WS 55% and 60% as column charge, non CS.

Permittends

Telperma-1 WS-60, deg. 1 gassy

Telperma-5 WS-35, solid blasting

Detonators :

Ordinary reinforced - used with safety fuse.

Instantaneous (aluminium); also of copper for deg. 1 and 2 gassy U/G coal.

Millisecond delay, delay interval 500 ms.

Copper delay; delay interval 25 ms; solid blasting in U/G cola mines.

Detonating fuse :

Telcord II (10 g/m coreload) sky blue. Telcord I (6 gm/m).

Telcord III (20 g/m coreload) sky blue.

Rajasthan Explosives & Chemicals Ltd.,

All explosives except booster are AN based slurries, and have the prefix Raj.

Explosives

Rajbase (83, 125, column charge, non CS.

Rajcol (83, 125); column charge, non CS.

Rajblast (75, 83, 125, 175, 200), primer and plaster shooting; CS.

Rajdyne (25, 40, 50); CS

Rajmix, pourable slurry; non-CS.

Rajolite, cast booster, CS.

Detonators - Instantaneous elec. No. 8 strength, (aluminium).

Detonating fuse - Red, coreload 10 gms/m.

Sri Krishna Explosives & Accessories Co.

All explosives are AN based slurries.

Explosives :

Maruti blast (25,50), base charge, CS.

Maruti-boost/Maruti prime (83, 125, 200), primer, CS.

Maruti column/Marutibase, (100, 150, 225), column charge, non-CS.

Marutiplast (pouch of 250 gms), plaster shooting, CS.

Orient Explosives (P) Ltd.

All explosives are AN-based slurries.

Explosives :

Ore boost, (83, 125, 175, 200), base charge, CS.

Orion-80, (83, 125, 175, 200), column charge, non CS.

Ore gel I & Ore gel II (83, 125, 175, 200), column charge, non CS.

Ore blast (25), CS.

Navbharat Explosive Co. (P) Ltd.

All the explosives are AN-based slurries. They have the prefix Bharat.

Explosives :

Bharat column & Bharat base (83, 125, 175 & 200), primer charge, Non CS.

Navbharat Fuse Co. (P) Ltd.

All products carry the prefix Nova. The explosives are AN-based slurries.

Explosives :

Novaprime (85, 125, 175 and 200), base charge, CS.

Novaboost (as above), base charge, CS.

Novacolumn (as above), column charge, non-CS.

Novabase (as above), column charge, non-CS.

Novapower as (above), base charge, non-CS.

Novamite (75, 83, 125, 175), column charge, CS (no. 8 det).

Novadyne (25), CS.

Permitteds

Novcoal I Deg. 1 gassy.

Novacoal II deg. 2 gassy.

Novacoal V solid blasting.

Pentolite booster

Novalite 20, CS

Novalite 250, CS

Detonating fuse

Novacord 1 (coreload 6 gm/m), Novacord II (10 gm/m) and Novatwin

SUA Explosives and Accessories Ltd.

(Previously known as MACWIN Explosives)

All explosives carry the prefix MAC. They are AN-based slurries.

Explosives

Macwin LD (83, 125), as booster, CS

Macwinex NCS-I (83, 125), column charge, non-CS.

Macwinex NCS-II (83, 125), column charge, non-CS.

Macwinex base (83, 125), CS. Macwinex SD (25 mm)

Macpentolite cast booster, CS.

Detonating fuse

Macord I, 10 gm/m coreload, blue   Macord ... 5 gm/m, white

Ordance Factory, Bhandara

Permitted Explosives

Prakash, P1, deg. 1 gassy.   Prachand, P1, deg. 1 gassy

Pragalbh, P3, deg. 2 gassy.

prachar, P5, solid blasting

Narendra & Co.

Explosives

AN-based slurry explosives, 25 mm dia. cartridges

Gunpowder F.O.V.F.

Safety fuse for wet location.

Namonitegel slurry explosive, 83, 125, 200 mm dia cartridges

Namodyne, booster charge, 25, 83, 125, 200 mm dia.

A few other explosive manufacturers not listed above are :

Orissa Explosives (P) Ltd; Bharat Explosives Ltd; Chamundi Explosives (P) Orissa Explosive Explosives (India) Ltd.; Deejay Dynamix (P) Ltd; Ideal Industrial Explosives (P) Ltd; Eastern Explosives and Chemicals Ltd.; Anil Chemicals Ltd.

End of Table

DETONATORS AND ACCESSORIES

High explosives are initiated by detonators or detonating fuses. A detonator is a small copper or aluminium tube containing essentially a small auxiliary charge of special explosive. A chemical reaction initiated by a flame or electric current in the special explosive can build up very rapidly into an explosion of sufficient intensity to project a detonation wave throughout a high explosion enclosing the detonator.

Detonators are of the following types :

1. ***Plain detonators :*** These are fired by safety fuses, the spark or "spit" from the fuse causing the detonator to explode; these are sometimes called "ordinary" detonators.
2. ***Ordinary electric detonators :*** These are fired by passage of electric current through the detonator. They are further subdivided as :
   (a) Low tension detonators, and
   (b) High tension detonators (not generally used in mining)

Ordinary electric detonators are of instantaneous type, *i.e.* without any delay element. They are of copper or aluminium tubes.

3. **Delay detonators :** These are essentially low tension electric detonators with a delay element, the object in their use being to phase the firing of shots, so that time and effort are saved in charging and firing several successive rounds of shots.

   These are subdivided as :

   (a) Half second or long delay detonators, and

   (b) Milli-second delay detonators (also known as short delay detonators).

A plain detonator. i.e. non-electric consists simply of an aluminium tube 6 mm dia., 37 to 50 mm long, filled 1/3 with A.S.A. composition and penta-erythritol tetra-nitrate (P.E.T.N.). The A.S.A. composition consists of a mixture of lead sized. (A), lead styphnate, (S) and a little aluminium powder (A). The A.S.A. priming composition initiates the base charge of P.E.T.N. which is a much more powerful explosive.

No. 6 detonator is suitable for normal requirements of mining work. No. 8 detonator is more powerful than No. 6 but is not generally used.

In an ordinary electric detonator, *i.e.* non-delay type or instantaneous type, the priming charge and base charge are the same as for plain detonator, but they are fired, not by ignition of a safety fuse, but by passing electric current through a fusehead. The current ignites a flashing composition in the fusehead, which in turn, initiates the priming charge (Fig. 8.2).

The resistance of low tension detonators with a 45 m long shotfiring cable is about 7 ohms. Current required for ignition of the fusehead is 0.5 amps so that a single detonator can be blasted with minimum voltage of 3.5 V. The circuit continuity of a L.T. detonator can be tested by a galvanometer and simultaneous shotfiring of a number of detonators with series connection is possible.

Delay Detonators

In appearance and composition, this is like the L.T. detonator. The L.T. detonator described above is of instantaneous type; the moment voltage is applied to it, the detonator explodes and along with it, the enclosing explosive. A delay detonator has a delay element introduced between the fusehead and the priming charge. (Fig. 8.2). The delay element consists of a copper or brass sleeve filled with a special composition which burns at a specified rate and the delay is obtained by varying the length of the sleeve containing the special composition.

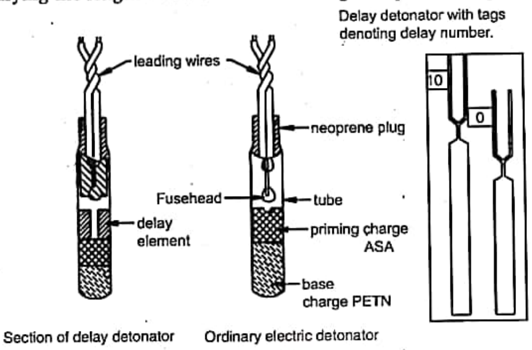


Fig 8.2

Delay detonators of some manufactures have antistatic sleeve over the fusehead as a protection against static electricity hazard.

The delay detonators and non-delay detonators are distinguished by the colours of lead wires. The delay period is marked on a tage attached to the wires. Moreover the delay number is stamped on the bottom of tube.

In underground coal mines aluminium detonators are not permitted but only copper detonators should be used.

Permission from the D.G.M.S. is required before using delay detonators in underground coal mines. Delay detonators and non delay detonators should not be kept in the same box.

Advantages of Delay Detonators

1. Reduced consumption of explosive as blasting is more efficient due to availability of a free face for each row or round of shots e.g. blasting due to No. 1 delay detonator gives a free face for the blasting effect of shots fired by No. 2 detonator.
2. Increased fragmentation and ease of loading the rock or coal. Broken rocks from successive shots collide in air, thereby increasing the fragmentation.

3. Considerable time is saved in that the whole round of shots is fired in a fraction of a second. This is the chief advantage. If individual shots, or even groups of 5 or 6 shots simultaneously by L.T. detonators are fired, the time required for inspection and clearance of fumes and gases between successive firings is considerable. In steep seams, the exertion involved in frequent trips for such examinations and connections is saved.
4. The millisecond delay (short delay) detonators have been observed to produce less ground vibrations than the half second delay detonators and are therefore used in mechanised quarries where blasting of large diameter holes containing heavy charges is likely to produce excessive ground vibrations and damage to nearby surface buildings and other important engine foundations, etc.

Accessories

Safety Fuse :

A safety fuse which looks like a cord consists of a core of fine grained gunpowder wrapped with layers of a tape or textile yarn and waterproof coatings. The burning speed is usually 100 to 120 sec/metre. ICL manufacturers a range of safety fuses to suit various conditions, *e.g.,* Double Bull brand for dry conditions, Blue Sump for damp conditions, OCPS (orange coloured plastic sheathed) and Blue Plastic for wet and very rugged conditions. IDL also manufactures safety fuse (yellow). When one end of the fuse is ignited, it carries the flame at a uniform rate to ignite gun powder or to detonate an ordinary detonator which is turn can detonate a high explosive.

Detonating Fuse :

For shallow depths, say less than 3 m, and for small number of holes, a detonator is inserted in the cartridge itself and detonated by ignition of safety fuse or in the case of elec. detonator, by an exploder. For a large no. of holes blasted at a time in mechanised quarries and in U/G coal mines electric detonators are used. A deep hole in a quarry needs a long length of detonator leads and to avoid this it is common to use a detonating fuse like cordtex (trade name of ICI. The fuse consists of a core of PETN enclosed in a tape which is wrapped with cloth. The fuse is then completely enclosed in a tubular cover of plastic material which is white for Cordtex and orange for Geocord detonating fuse (ICI). The detonating fuse looks like a plastic cord; its external dia. is about 5 mm and weight about 20 g per metre length. It has a velocity of detonation of 6500 m/sec. and it is practically instantaneous in its action.

A large number of shots connected with detonating fuse can be blasted by a single detonator. A detonating the fuse through the plastic cover. Water may however penetrate into the core through the cut ends which can be guarded against by sealing them with tape or water-proofing compound.

A detonating fuse is often used for demolition operations.

Nonel

The Nonel system of detonation is developed by Nobel AB of Sweeden. Primers of explosives with Nonel detonators inserted in them are charged in the blast-holes and the Nonel tubes are bunched for convenience of connection to the mains blasting system. Upon initiation, the shock wave passes down the plastic tubes, the insides of which are coated with reactive substance that maintains the shock wave at a rate of approx. 2000 m. per second which has sufficient energy to initiate the primary explosive or delay element in a detonator. Since the reaction is contained in the tube, this has no blasting effect and acts as a signal conductor.

Nonel means non-electric detonator. The flexible plastic tube has 3 mm external and 1.5 mm internal diam. The tubes are available in pre-cut lengths. One end of the tube is fitted with a non-electric delay detonator which is crimped to it in the factory while the other end is sealed. The end having detonator is lowered down into the blast hole while the sealed end projects outside the hole. The sealed end is initited by a detonator or detonating cord.

The advantage of the Nonel system lies in its extreme resistance to accidental initiation by static electricity, stray current, radio transmission, flame, friction and impact. It is also immune to misfires caused by current leakage in conductive orebodies and eliminates the need for complicated electrical circuit testing and shot-firing equipment.

Raydet manufactured by IDL Chemicals, is just like Nonel.

Raydet

Raydet is a non electric initiating device combing the versatility and advantages of electric detonator and detonating cord. It consists of a plastic tube carrying a very small quantity of explosive material on its inner surface. A high strength no. 8 instantaneous or delay detonator is crimped to one end of the raytube. When initiated, a low order shock wave travels through the tube and initiates the detonators. Raydet can be initiated by a detonator or a detonating cord. A tag indicates the delay number of reydet and a tape fastening the tube in a coil indicates the tube length. Length of tube varies from 3 m to 45 m. The delays are from no. 0 delay to no. 15 delay; No. 0 delay is instantaneous No. 1 delay is 50 ms and No. 15 delay is 625 ms.

When using the raydet, do not cut factory sealed end of raytube and do not connect two raytubes. One raytube will not initiate another.

Fig.

Detonating Relays

In opencast workings, detonating relays using detonating fuse for initiation provide a non electric delay firing system. This method avoids the electrical connections which are required when using delay detonators. A detonating relay is essentially an assembly of two open-ended delay detonators coupled together with flexible neoprene tubing in an aluminium sleeve suitable for crimping into a detonating fuse.

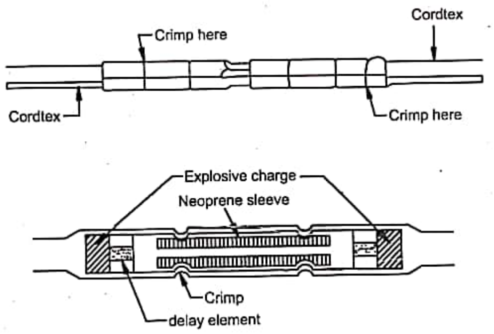


Fig. 8.3 Detonating relay (top) and (above) in section

Inside the detonating relay, the construction is symmetrical with the delay element at either end so that the detonation wave may pass in either direction. The delay interval for each detonating relay varies from 15-45 millisecond. In sue, the main or branch line of detonating fuse is cut at the point where a delay is required, and the detonating relay is then crimped between the two cut ends of the line. By judicious selection of the points at which the detonating relays are inserted, any delay firing sequence can be arranged (Fig. 8.3). Being nonelectric in nature, detonating relays are insensitive to stray current and static electric.

Cord relays

Delays detonating relays manufactured by IDL Chemicals are known by the trade name *Cord Relays.*

A cord relay has a dia. of 11.5 and length of 152 mm. They are available with two delay periods, 15 ms and 25 ms.

Under the Explosives Rules, the various explosives and accessories are classified under the following headings :

Class 1	–	Gun powder.
Class 2	–	Nitrate mixture (e.g. GN/1, Powerflo, Godyne, Pentadyne, etc.)
Class 3	–	Nitro compounds, e.g. Blasting Gelatine, Special Gelatine, O.C.G. permitted explosives, Permex, Gelonite, Powex 80, TNT, gun cotton, PETN, etc.
Class 4	–	Chlorate mixtures.
Class 5	–	Fulminate.
Class 6	–	Ammunition safety fuse, detonating fuse, detonators, delay detonator relays, etc.
Class 7	–	Fireworks.
Class 8	–	Liquid Oxygen Explosives

Circuit Tester :

In electric shot firing before any attempt is made to fire the shots, the circuit is sometimes tested to make sure that there is not open or short circuit. Such testing should be done by approved apparatus and it is important that the current passed during testing should be limited, so that there is no possibility of accidental explosion of the detonators. In addition, all testing must be done from a safe a place and safe distance from the blast site.

An instrument to test continuity of an electric circuit for blasting is the blastometer manufactured by IDL chemicals Ltd. It is an electronic solid state circuit tester and is available in two ranges :

(a) 0 to 100 ohms for underground coal mines.

(b) 0 to 100 ohms for other applications.

Crimper

A crimper is a paid of pliers to crimp or press the end of a plain detonator tube on a safety fuse inserted into it so that the fuse cannot come out of the detonator. It is dangerous to crimp the tube end with teeth.

Shot firing cables.

During electric shot firing the leads for the detonator are connected to long shot firing cables (not less than 30 m in length) to fire the shots from a safe distance. The cables are twin-core and insulated to withstand at least 250 V.

Other accessories for shot firing include.

i. Wooden stemming rod to stem the holes.

ii. A wooden dolly weighted with lead or brass for deep holes in mechanised quarries.

iii. A scraper made of brass to clean the holes and detect cracks.

iv. A pricker made of brass, aluminium or wood to prick the cartridge prior to inserting the detonator or detonating fuse.

Exploder

Electric shotfiring is safer, quicker, and more convenient than ordinary fuse firing by flame or ignition as it provides safety to the shotfirer and his helpers. Tapping of electric power from lighting or signalling lines is not permitted by the Regulations.

The portable apparatus which provides the current necessary for firing electric detonators is called an exploder.

The types of exploders than are currently used in India.

Magneto (or Dynamo) Exploder

The magneto exploder consists essentially of a permanent steel magnet between the poles of which revolves an armature rotated through gearing by rotary handle or by a rack and pinion. The value of the voltage depends upon the speed at which the armature revolves and the flux created by the magnets. A low tension exploder gives a voltage of about 15 volts. A.H.T. exploder gives about 125 volts. The mangneto exploder fires only 1 or 2 shots at a time with single shot exploder and upto 6 shots in series with a 6-shot exploder.

The exploder for U/G coal mines should be intrinsically safe. The armature is actuated by a special twist action detachable key which should always be with the shotfirer.

Battery Condenser Exploder

The current is provided from a battery of 4 or more dry primary cells connected in series, each giving an e.m.f. of 1.5 V per cell. It is operated by a detachable key. Rotation of the handle through half a revolution first winds up a time up a time switch against the tension of a spring and then trips the mechanism which is controlled by a centrifugal governor so that the battery is connected to the condenser during a predetermined interval of time (a small fraction of a second).

The Rhino exploder is the market is an example of battery condenser exploder. It in battery operated condenser discharge type exploder capable of firing upto 3 shots at a time in Rhino 3 model (and upto 25 with Rhino-25). When using the exploder, the two wires of shot firing cable are connected to the terminals on the exploder. Insert key and rotate one step clockwise. The action charges a 150 microfarad eletrolytic condenser and a small neon lamp glows very brightly. To fire, turn key anticlockwise to "Off" position. This action explodes the detonator and also discharges the residual charge instantly. In Rhino-25 the firing current is 1.5 amps, input D.C. volts 6, output D.C. volts 600 and firing current duration 3-4 millisecond. The exploder uses only one P-276 Eveready battery. (Fig. 8.4).

Rhino exploders are permitted only in degree 1 and degree 2 gassy mines of coal.

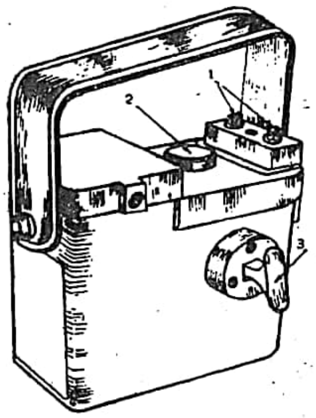


Fig. 8.4 Rhino exploder 1. Terminals. 2. Neon Lamp. 3. Key

Condenser Dynamo Exploder

These are manufactured by Narendra & Co. The machine is suitable for operations in places where inflammable gas does not constitute a hazard. Three types are in the market.

CNT 50 to fire 50 detonators connected in series.
CNT 100 to fire 100 detonators in series.
CNT 200 to fire 200 detonators in series.

All the three are suitable where the circuit resistance is 160 ohms. The CNT-100 exploder has an output voltage of 750 V.D.C. The duration of current impulse to the line varies from 5 to 6 millisecond. It has got one visual neon indicating lamp to indicate that the exploder is ready to fire. To operate - connect the shot firing cable to the exploder terminals. Place handle on to the unlocking shaft; rewind until the moon light comes on and then fire; switch off and disconnect.

Transport And Storage Of Explosives

A building where explosives and detonators are stored is called a Magazine. A magazine construction has to be approved by the Inspector of Explosives and it has to be on a site which also needs his approval. Certain safe distances are specified for the magazine site and they specify the minimum distance of residential quarters, public roads, etc. from the magazine. These distances depend upon the capacity of the magazine; larger capacity requires longer safe distance. Small magazines in the mine premises are according to lincence in Form J. Such a license permits the magazine owner to store upto 45 kg of gun powder, 5 kg. of other NG explosives, 200 detonators, 50 kg slurry explosives (nitrate mixture) and any quantity of safety fuse. The large magazines in the mine premises are according to lincence in Form L for storing high explosives upto 25,000 kg and 2,00,000 detonators.

Under the Explosives Rules the following can be stored together in the same magazines :

Gunpowder (class 1), nitrate mixture (class 2), nitro compounds (class 3), chlorate mixture (class 4), safety fuse, plastic igniter chord and detonating fuse (all of class 6).

Procedure For Establishing a Magazine

An application in form 'C' and 'D' should be made to the Regional Controller of Explosives together with six copies of site plan and magazine constructional details. He will forward all the documents to chief Controller of Explosives (CCE). CCE will then issue a form 'E' plus a draft copy of the lincence 'L' and will also pass one copy of site plan to District Magistrate who will issue a 'No Objection Certificate'.

Based on this, CCE will allow the applicant to proceed ahead with the construction of the magazine and also issue a lincence in form 'L'. On completion of the building the Regional Controller will inspect the magazine and endorse the lincence which must be renewed annually.

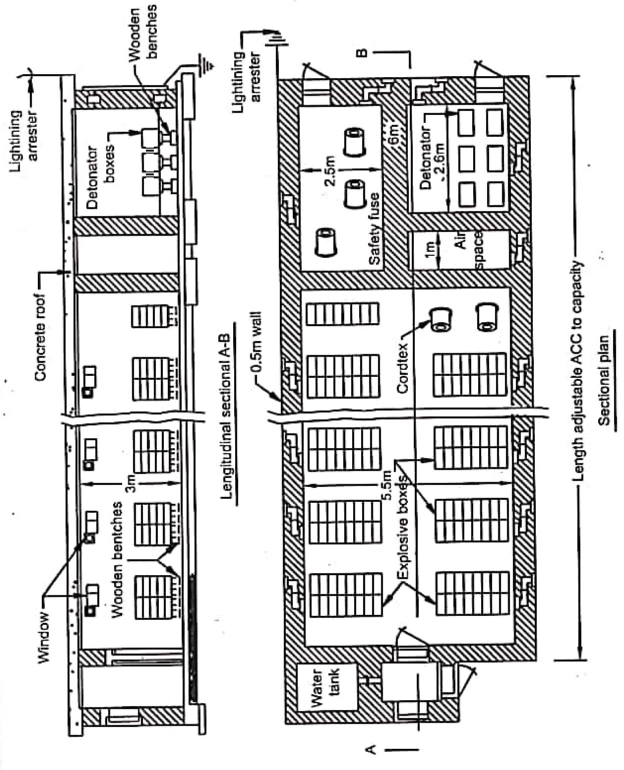


Fig. 8.4 (A) A magazine of L Type for storage of large quantities.

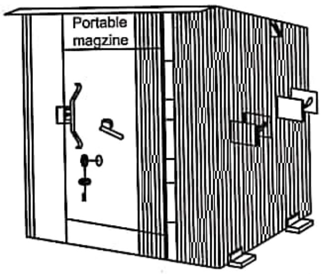


Fig. 8.4 (B) Portable explosive magazine; capacities vary from 100 kg to 500 kg. Outer steel shell 5 mm thick.

A portable magazine requires a lincence from the Chief Controller of Explosives. It should be located on a ground about 5 m × 2 m keeping the following safe distances :

i. From all buildings, huts, places of worship, officers, houses, schools, factories, etc..... 95 m.

ii. From all roads, river walls, market, playground, etc....48 m.

iii. From overhead high tension electric line ... 91 m.

Deterioration in quality during storage

The chemical composition of explosives may change slightly after storage under humid and hot conditions. The explosive may become insensitive to detonation or unsafe to use if the change in chemical composition is excessive. At the mines no laboratory tests are carried out to decide the extent of deterioration, but visual examination is the only guide. When the wrapper of a gelatinous nitroglycerine explosive is removed carefully an explosive in a good state would permit the wrapper to come off cleanly from the cartridge and there would be no evidence of discoloration, incrustation, softening or exudation of liquid in the explosive itself. A gelatinous explosive slightly darkens during storage, particularly at the ends due to absorption of moisture from the air. Such discoloration is usually accompanied by softening and dampening of the explosives. The moisture absorbed by the explosive may cause a white incrustation of ammonium nitrate or sodium nitrate on a gelatinous explosive.

A higher degree of deterioration is indicated by exudation of nitroglycerine or aqueous solutions of sodium or ammonium nitrate and the cartridges becoming too plastic. Such explosive should not be used but destroyed.

Powdery nitroglycerine explosives detenorate much faster than the gelatinous quality as the former absorb moisture more readily. The deterioration is indicated by discoloration at the ends, softening and dampness. If there is any exudation of liquid the cartridges should be destroyed.

Ammonium nitrate explosives mixed with T.N.T. cake and become solid due to absorption of moisture during storage. If the cartridges become so solid that they cannot break easily between the finger and the thumb, they should not be used but destroyed.

Slurry watergel emulsion explosives : Deterioration of slurry explosives is usually indicated by a breakdown of the gel into separate liquid and solid components. It may also appeal as hardening of the explosive cartridge. Cartridges may also be swollen or distored and salts in the composition may crystallise on the outside.

Detonators : Detonators showing signs of corrosion of the tubes are dangerous and must not be used; this applies particulary to plain detonators.

Detonating cords : Almost the only cause of deterioration or desensitization of detonating cord is moisture penetration into the explosive core. Prolonged exposure to sunlight may cause the plastic outer coating to become brittle.

Destruction Of Unserviceable Explosives

Explosives which have become unserviceable for blasting should be destroyed and never dumped into a garbage heap. It is a sound policy to seek expert advice from the manufacturers for methods of destroying explosives. Always assume that burning an explosive for destruction may change to detonation. The method adopted for destroyed are as follows.

Gunpowder : It should be thrown into water, preferably hot water, which dissolves out the saltpeter and renders the explosive harmless.

Ammonium nitrate explosives : These are hygroscopic and may be buried in a damp ditch where they become harmless with the absorption of moisture. Or they may be destroyed by drawing in a large volume of water.

Explosives with nitroglycerine : Not more than 25 kg of explosives should be destroyed at a time. A clear space of ground free from dry grass, about 100 m all around, should be selected and a lie of shavings of dry straw or grass laid down. On this the cartridges should be placed in a continuous line not more than two abreast with the cartridge wrappers below them and with an interval of about 25 mm between two cartridges. The line of shavings of straw or grass should extend some distance, say 6 m, beyond the explosives. Kerosene should then be sprinkled over the shavings of straw or grass. The straw should be lit with a short length of safety fuse and the operator should withdraw immediately to a safe distance. The direction of wind should be given due consideration when burning the straw.

The degree of continement caused by the cartridge wrappers in high explosives is sometimes sufficient to cause explosion. It is, therefore, essential to open the wrappers and unroll them.

Safety fuse : It should be destroyed by burning in length in the open under suitable precautions.

Detonators : They should be thrown into a deep river or alternatively, they may be soaked in mineral oil for 48 hours and then destroyed, one at a time under suitable precautions, by burning.

When destroying explosives do not select a stony site where burning or detonation could create situation of flying stones during destruction.

BLASTING PRACTICE IN MINES (UNDERGROUND)

If the blasting is not off-the-solid, the position, direction, depth and number of holes at a face are governed by the following factors in an **underground coal mine.**

1. ***Position and depth of cut :*** A hole should be at least 150 mm shorter than the depth of the cut, made by coal cutting machine.

2. ***Type of roof :*** If immediate roof is shale the hole should terminate 0.3 m below the roof; if the roof is sand stone, the hole should terminate 150 mm below the roof.

3. ***Type of bands and their position :*** the hole should be drilled near the band and if it is of hard rock, it should be drilled towards it, slightly below or above so that the inclined hole terminates near the band where the explosive force is maximum. The holes should be drilled below the band if it is near the roof, specially when the roof is weak or friable. Fig. 8.5 shows the position of holes at an undercut face with a stone band.

4. ***Cleavage in the coal :*** Shot holes should cross the cleavage planes as near to right angles as possible.

5. ***Strength of explosives and hardness of coal :*** Harder coal requires more density of holes or explosives. If output of lump coal from the mine is important it is advisable to use a low density explosive and this influences the number and position of holes.

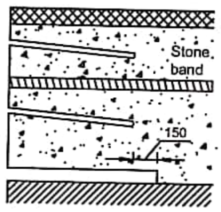


Fig. 8.5

6. *Dimensions of the gallery :* A gallery of large cross section gives better yield of coal per kg of explosive. Gallery with more height, say between 1.5 m and 2.4 m, requires two rows of holes but only one row suffices for low height.

Drilling Patterns In Stone

Drilling patterns, also called shot-hole patterns, are named after the type of cut holes used and the principal patterns are :

(1)	*Pyramid cut or cone cut*	(2)	*Wedge cut*
(3)	*Drage cut*	(4)	*Fan cut*

(5) Burn cut (6) Coromant cut,

(7) Ring drilling

In general each hole in a round covers an area of 0.3 to 0.5; cut holes are located about 0.45 m vertically apart, and easers 0.5 to 0.6 m apart; trimmers are drilled at intervals of about 0.6 m round the perimeter of the drift.

Pyramid cut or cone cut : Pyramid cut consists in drilling holes (in the centre of the drift axis) at corners of a square, 0.7 to 1 m sides, almost to meet at a point at the back of the round. In a modified design known as cone cut, illustrated in Fig. 8.6, holes are drilled forming corners of a polygon with a centre holes, all nearly meeting at a point at the back of the cut. The depth of pyramid cut is generally restricted to 50% to 60% of the width of the drift:

Wedge cut : Wedge cut takes the form illustrated in fig. 8.6 in which 2 to 4 pairs of holes are drilled to form a wedge, each pair starting from two sides of the drift centre and inclined at an angle less than 45° towards the centre almost meeting at the back of the cut along a line.

Pyramid and wedge cuts, the most commonly used forms, are suitable for uniform, thickly bedded and hard rocks. They consume the least total quantity of explosives, but the depth of pull is restricted by the width of the drift.

Drag cut : Drag cut, used for small drifts, 1.8 to 2.4 m wide, consist in drilling holes at an angle to the cleavage so that strata break along the cleavage planes (Fig. 8.6). This pattern, being dependent on the direction of cleavage planes, calls for frequent changes, which are detrimental to systematic work and the pattern is, therefore, not favoured for large excavations.

Fan cut : Fan cut, favoured for laminated strata, mostly soft, covers the face with a fan-like pattern as shown in Fig. 8.6 As each shot has to act for itself, charge in each hole is heavy. This cut is not recommended for hard ground.

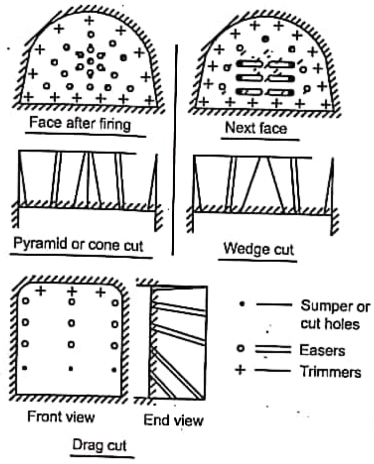


Fig. 8.6

With the above mentioned cuts it is normally difficult to drill deeper than half the width of the drift because of the angle of the drill, but with the burn cut, advance equal to the width of the drift can be obtained.

Burn cut : In Burn cut, parallel holes at right angles to the face are drilled in a cluster which may take several forms one of which is illustrated in Fig. 8.7. Some of the holes (which are sometimes of larger dia. than the rest) are left uncharged to give relief to the heavy concentration of explosives in charged holes. This cut is effective in hard, brittle, homogeneous ground which breaks evenly; but cannot be used in springy plastic ground. The advantages claimed for this cut are :

1. Drilling time is considerably reduced and supervision in drilling is easier as holes are straight.
2. Depth of pull is dependent of the size of the drift, and
3. The quantity of blasted material is not projected far with a suitable form of the cluster.

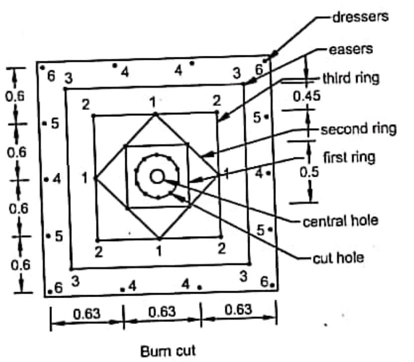


Fig. 8.7

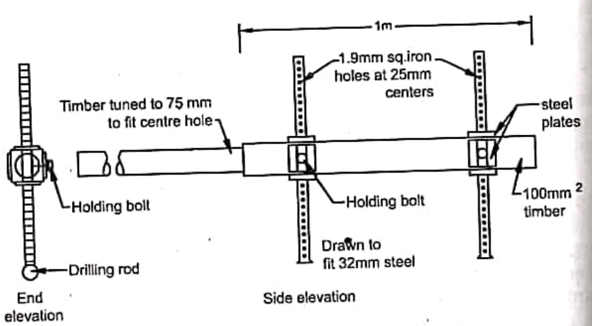


Fig. 8.8 Drill guide for holes on burn cut pattern

All the holes, charged as well as uncharged ones, may be drilled by the same drill and those to be left uncharged may be reamed by a reaming bit. At Mosaboni mine shot-holes for burncut in the drives are 32 mm dia. and the central holes which is not charged is reamed to 58 mm. dia.

The coromant cut : The coromant cut is a new type of parallel hole cut which has been worked out with the object of achieving greater advance per round in tunnels or drifts of small area. In principle, the Coromant cut consists of a slot, which is left unloaded, together with 6 outer cut holes, the locations of which are carefully calculated. All the drilling is done with the same pusher-leg drill. The slot is produced by the contiguous drilling of two holes. Drilling is carried out using a 20 mm drill rod with taper and a special drill bit of 57 mm diameter. The hole is drilled first, being directed slightly upwards to assist in the disposal of cuttings. Continuous drilling calls for guidance in drilling the second hole. For this purpose a guide tube is used, being inserted into the first hole and secured by means of an expander.

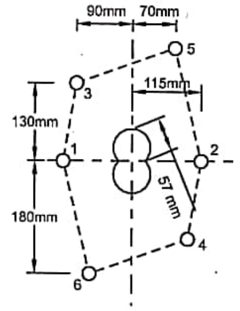


Fig. 8.9 Coromant cut pattern

Burn cut and coromant cut are known as parallel-hole cuts. In drilling parallel-hole cuts, precision is always an important factor. A special *drill jig or template* which gives the desired precision and facilities drilling, has, therefore been designed for the outer cut holes. This template also drilling considerably faster. The template normally used is the 6-hole template which fits in with the standard drilling pattern of the Coronmant Cut. The two centre tubes of the template are slid into the finished slot, and the template is then secured in place by means of an expander in one of the tubes. The drilling of peripheral holes in the cut is then carried out using ordinary integral steels.

It may be possible to have only one centre hole depending on the nature of the rock. The charge varies with the dia. of the empty hole and in the case of the pattern shown, the recommended charge is 0.3-0.4 kg/m length of the hole. The holes are fired in a sequence shown by delay number in the pattern.

The cut holes and the rest of the holes in the drift are normally blasted in the same round with the help of milesecond delay detonators. The standard coromant cut pattern covering six cut holes has been found to be good for the great majority of rocks. The arrangement of the holes and the firing order have been so chosen that the dislodged rock is given sufficient room to expand.

Ring drilling : For some types of stopping operations in metalliferous mines drilling with long holes is practiced. There are 2 types of ring drilling. (a) Vertically ring drilling where rings are drilled in vertical planes, radially like a fan to break to a vertical face; applicable in sub-level stoping, illustrated in Fig. 8.10 (b) Horizontal ring driling where the holes are fanned out radially in horizontal planes to break to a horizontal face; applicable in shrinkage stoping. The principle is the same as for vertical driling and blasting. Short delay interval of 25 millisecond is usually employed between holes in each row or ring, starting from the easiest breaking section in the middle and progressing towards the walls.

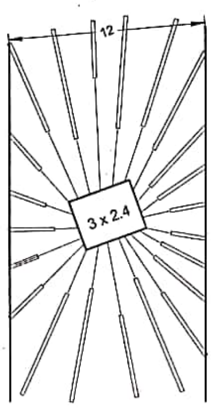


Fig. 8.10 Ring Drilling (figures in metres)

Long hole blasting is done in wide or bodies with strong walls so that dilution is minimal. In narrow ore bodies, on the other hand, long hole blasting is not generally economical due to the high cost of drilling. The method is also not quite applicable in irregular ore bodies because long holes would either tend to leave ore unblasted in places or to break into waste rocks.

Solid Blasting

In a development gallery, coal can be blasted without giving an undercut, by the use of explosives of P5 type. The blasting is known as blasting-off-the-solid or, in short solid blasting. P5 explosive is used alongwith carrick short delay detonators (of I.E.L) or coal delay detonators (of I.D.L.) Introduction of such blasting in mines needs approval from the D.G.M.S. for exemption under the following regulations :

(a) Reg. 173 : To blast off the solid.

(b) Reg. 175 : To use delay detonators.

(c) Reg. 168 : (5) : To fire rounds of more than 6 shots in coal.

The pattern of shot-holes drilled for solid blasting in coal in galleries of bord and pillar working is :

1. Wedge cut, or *2. Fan cut*

Drivage of a narrow gallery in coal without an undercut can be compared with drivage of a drift in stone. Blasting out coal in the centre of the face in the wedge cut pattern gives free face for the remaining coal. The maximum permissible charge to be used per hole therefore varies when blasting out the central wedge with P5 explosives and when blasting out the remaining cola (after the central wedge is blasted out thereby providing free face) with P1/P3 explosives if these are used. On a longwall face, the shotholes are drilled at an angle of 45° to 60° to the face; a coal face 2.4 m high requires 3 rows of shotholes and distance of holes in the same in the same row should be nearly 1.2 m.

Powder factor with solid blasting is 1.5 to 2.5 te of coal per kg of explosive and 0.8 to 1.25 te of coal per detonator.

DGMS stiplations on maximum permissible charge in a shot-pole, delay intervals, etc. in u/g coal mines.

1. Explosives

Types of Explosives	Degree of gassiness/Type of Application	Max. permissible charge per shot-hole (gms)
P1	Degree 1 mines, cut face	800
P3	Degree 1, II & III, cut face	1000
P5	Degree I-"BOS"	1000
P5	Degree II & III - 'BOS'	565

2. Delay detonators

(a) While using non-incedive delay detonators in 'BOS' applications the maximum delay period between the Ist & last shot in a degree I and II gassy coal seam will not exceed 150 millisecond.

(b) While using non-incendive delay detonators in 'BOS' applications, the maximum delay period between the Ist & last shot in degree III gassy coal seems will not exceed 100 millisecond.

(c) The delay period between 2 consecutive shots with different delay numbers will not exceed 60 milli-second.

3. Distance between 2 adjacent shots with different delay numbers will not come closer than 0.6 m at the explosive-charged ends.

The advantages of solid blasting are :

1. It eliminates the use of coal cutting machines. The particularly important in steep seams where flitting and control of coal cutting machine is difficult and risky.
2. Skid mounted coal cutters are fitted with the help of anchor pipes which can be used in seams of upto 2.2 m height. Moreover, the roof and floor of the seam to use such pipes should be hard. Where these working conditions do not exist the skid mounted coalcutter cannot be used and solid blasting is an advantages.
3. Saving in capital expenditure on equipment at the face, like coal cutting machine, gate-and boxes, cables, etc.

Compared to the undercut faces, the amount of drilling on a solid blasting coal face is high but the provision of one or two extra drills at the face is not very costly, considering the cost of coal cutting machines, gate-and-boxes and medium tension cables of special construction for remote control operation.

4. If tubs are used for transport of coal, track can be laid right up to the face in the absence of coal cutting machines.
5. When working a seam over a caved area, the floor is irregular and coal cutting machines cannot be used. Solid blasting offers a definite advantages in such cases and in other cases where floor may be irregular for some other reasons.

For Deg. 1 gassy mines at least 284 m^3 of air per min. shall be conducted in the ventilation connection outbye of every face where solid blasting is to be practiced.

For Deg. 2 gassy mines at least 284 m^3 of air/min. shall be conducted upto every face where solid blasting is to be carried out.

The results obtained by solid blasting in one mine in Hazaribagh area (C.C.L.) are given below for guidance :

Gradient of seam	– 1 in 3.7
Category of gassiness	– degree 1.
Working method	– Bord & pillar development
Gallery size	– 4.2 m wide × 2.4 m high.
Shot holes at a face	– 1.4 m deep, wedge cut pattern total No. 12
Explosive charge.	– 500 g/hole; total charge 6 kg.
Pull per round.	– 1.06 m.
Coal yield/kg of explosive	– 2.0 te (soligex)
Coal yield/detonator	– 0.9 te

One driller and two helpers drill 60-70 holes in a shift of 8 hours; one shotfirer, 2 explosive carriers and 1 stemming material carrier constitute the blasting crew and fire nearly 60 shots in a shift of 8 hours.

Blasting patterns for shot holes in coal for solid blasting are shown in Fig. 8.11 and 8.12 in a narrow heading and in Fig. 8.13 for a longwall face.

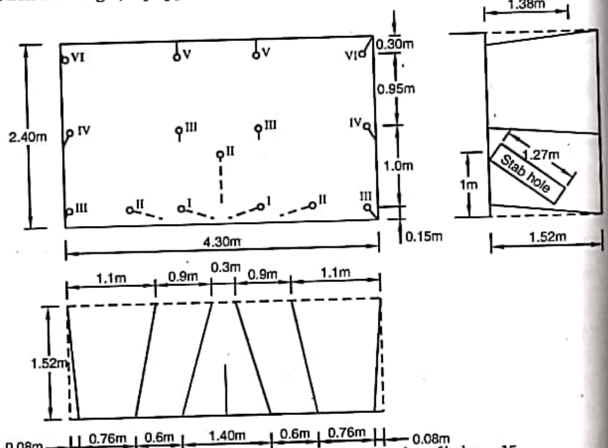
Limitations On Carriage Of Explosives

Explosives and detonators should not be carried in the same box. Not only this, one person should not carry explosives and detonators together though they may be in different boxes. An explosive carrier can carry only 1 case containing 5 kg of explosive but Director of Mines Safety may give relaxation. Explosive case should be numbered. Jeep or trailer is allowed to carry more explosives for opencast mines for deep holes.

Preparation Of Charge

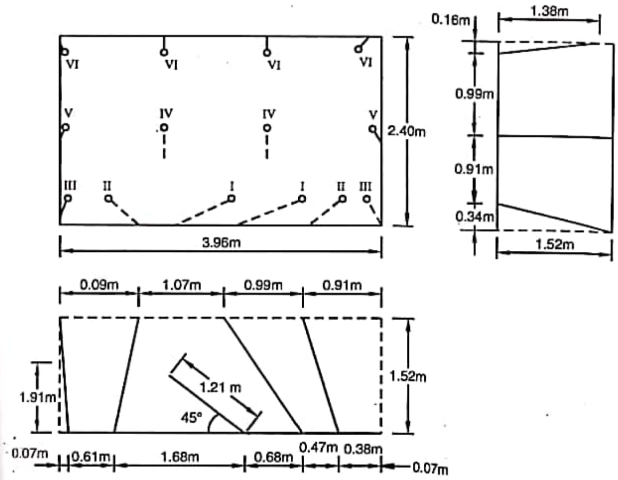
The charge for blasting in a hole may consists of one or more cartridges. It is desirable to have the least number of separate cartridges as far as possible, commensurate with the work to be done.

One of the cartridges should have a detonator inserted into it. Such cartridge, equipped with a detonator, is called a "primer cartridge".



Number of holes	:	15
Charge	:	600 to 550 g.
Rest of the holes	:	350 to 400 g.
Total Charge	:	6 to 7 kg.
Blasting Ratio	:	2.6 to 2.8t/kg.
Yield/detonator	:	1.1 to 1.2 t
Pull	:	1.30 to 1.52 m

Fig. 8.11 Wedge cut for solid blasting in coal



Number of holes	:	14
Charge	:	
Holes in bottom row	:	500 to 550 g.
Rest of the boles	:	350 to 400 g.
Total Charge	:	6.0 to 6.5 kg.
Blasting Ratio	:	2.5 to 3t/kg.
Yield/detonator	:	1.2 to 1.4 t
Pull	:	1.4 to 1.52 m

Fig. 8.12 Fan cut for solid blasting coal

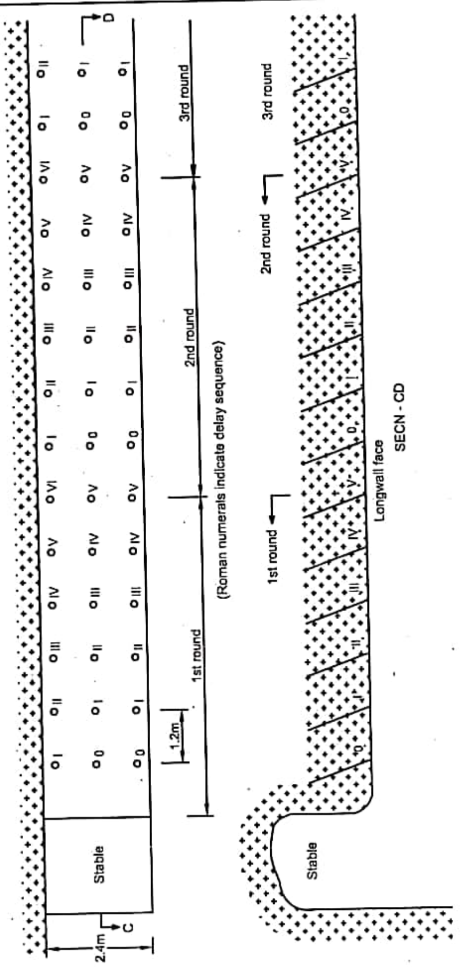


Fig. 8.13 Typical layout of shot-holes on a longwall coal face for blasting-off-the solids

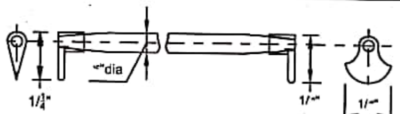


Fig. 8.14 Scraper and break detector

If an electrical detonator is used to prepare in primer cartridge, open the cartridge at one end, make hole with a pricker of brass or wood, insert the detonator until it is completely buried in the explosive, put back the flap of the cartridge and hitch the leading wires around the cartridge to prevent the detonator being withdrawn accidentally during charging. (See Fig. 8.15). Permitted slurry explosive can be primed from the sides by pricking a hole into it.

Charging A Shothole

A cartridge of non-combustible stemming material is first pushed in the hole. The charge is then placed in the hole and the primer cartridge pushed last of all, so that the "business end" of the detonator points towards the main body of the charge. This position of primer cartridge is termed "direct initiation" (Fig. 8.17). With this position of the detonator, the strongest wave is directed towards the back of the hole and the chances of all the cartridges being properly exploded are a maximum. Direct initiation is best to prevent ignition of fire damp, reduces risk of blow-out shots, and gives maximum yield of coal having a free face.

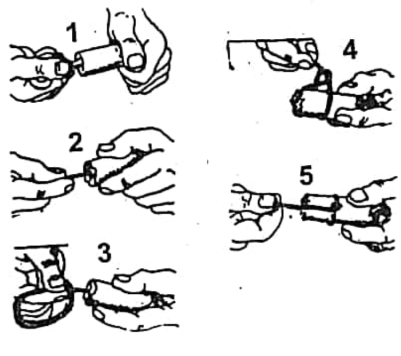


Fig. 8.15 Preparation of primer catridge

When the detonator is at the back of the charges and the "business end" points towards the front of the hole, it is called "inverse initiation". This is not practiced on coal faces, but is adopted.

(a) In "sumping" or "cut" shots in shafts and tunnels, and

(b) When using delay action detonators to fire a round of shots.

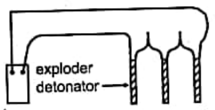


Fig. 8.16 Detonators in series

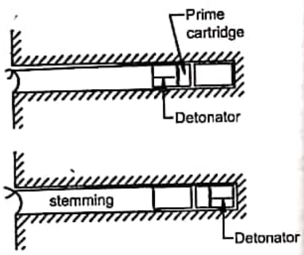


Fig. 8.17 Direct (top) and inverse (bottom) initation

After the charge is placed in the hole, the shot-hole is stemmed with stemming material, keeping ends of the leading wires of detonators out of the hole. The stemming material should be compact but plastic, consisting of sand and clay in 3 : 1 proportion, and dried at the surface in the form of cartridges 150 mm to 200 mm long and 25-30 mm dia. The first 2 or 3 stemming cartridges near the charge should be tamped lightly by the stemming rod and the rest should be tamped hard.

Complete Procedure For Shot Firer

The whole procedure which the shot firer has to follow to fire shots using electric detonators, is outlined below.

1. Test shot holes for breaks. If a crack is found, the hole shall not be charged.
2. Test for gas from the hole and within 8 m of the hole.
3. See that the shot hole is 150 mm less than the depth of the cut, if a coal cutting machine is used.
4. Mark direction of the shot hole on the roof or side where practicable.

5. Charge the hole with explosive; insert the primer last of all Don't force a primed cartridged into a shot hole of small size.
6. Stem the hole, first lightly and then hard, upto its mouth.
7. Spray stone dust or water within 18 m of the area.
8. Warn the workers to clear up and post helpers at suitable places 27 m away in approach roads to prevent workers inadvertently entering the area.
9. Lay out the shot firing cable.
10. Test again for gas within 18 m.
11. Couple the firing cable to detonator wires. If more than one shot is to be fired, all connections of detonator leads should be in series.
12. Take shelter.
13. Couple shot firing cable ends to the exploder.
14. Shout a warning again; ensure that workers have taken shelter, and fire the shots by a sharp twist of the exploder key. If the charge does not explode, try again with a sharp twist of the exploder key.
15. Allow the fumes and gases to clear.
16. Return to the shot hole, examine the roof, sides and timber supports and shout "all clear" for the workers to return to their work if the place is safe. Otherwise, have it dressed by dressers, and supported by timbermen before workers enter it.

At the end of the shift the shotfirer should write a report about the quantity of explosives blasted and the place of blasting.

Ignition Of Gases

The main danger from explosives in U/G coal mine is the ignition of fire damp. It may take place in the following ways.

1. *By the flame and hot gases :* Though this is a common cause, because of "lag on ignition", a flame, coming in contact with gas, does not ignite it instantaneously. If the flame is sufficiently cooled during the interval of lag on ingition, which is a fraction of a second, the gas will not ignite. If the heat energy generated during blasting of the explosive is effectively coverted into mechanical work, the explosion products are not hot enough to ignite the gas.

2. *By incompletely detonated explosive* : Such explosive may continue, to burn like an ordinary combustible material.
3. *By incandescent particles* coming out of the shot hole after blasting, if such particles come in contact with coal dust or gas.
4. *By the compression wave of the blast* which may compress the gases in the cracks connected with the shot hole and raise the temperature of the compressed gas to such an extent as to ignite it. Such 20-fold compression is known to be sufficient to ignite all inflammable mixtures of fire-damp and air. The gas in the breaks and fissures connected with shot hole and leading to a fire damp pocket cannot be easily removed and the presence of breaks and fissures is one of the commonest causes in explosions of fire damp due to shot firing. If any break is found, the shot must not be fired.

Blown out shot

A blown out shot is one which has not done any useful work of blasting coal, but has ejected itself out of the shot hole. An undercharged hole will result in blown out shot. Similarly in an overcharged hole where only part of the charge is utilisied in doing the useful work of blasting cola the excess charge may still be burning when the coal is broken, and it has the same effect of igniting gas or coal dust as an undercharged blow out shot.

Common Causes Of Accidents

1. Not taking proper cover. This is the most common cause of personal injury due to explosives. It is essential that the shot-firer shall himself take adequate cover and see that all workmen in the vicinity of a shot are removed to a safe place. No place in direct line with a shot can be regarded as safe and every person should be protected by at least one right angled corner. All approaches to the danger zone should be guarded by sentries or otherwise so as to prevent anyone entering inadvertently.
2. Failing to warn persons in an adjoining place into which the blasted rock may be thrown, as is possible when two galleries are about to join and partition is thin.
3. Carelessness in handling detonators causing them to explode or to be lost in a mine.

4. Carelessness whilst charging a hole, e.g. tamping too forcibly in the neighbourhood of the detonator, or ramming the primer cartridge into a hole of insufficient diameter.
5. Firing a shot when persons are at the shot hole due to instructions being misunderstood, or lack of proper sentries.
6. Returning to the face too early after firing a round of shots, one of which is a "hang-fire" (i.e. a delayed ignition), or before authorised to do so by the shot-firer.
7. Dealing with misfired shots otherwise than in the prescribed manner.

Ignition of gas or coal dust is a major accident which may arise from the use of explosives if proper precautions, as laid down in the Regulations, are not taken.

Misfired Shot

When a detonators fails to explode, or after exploding fails to blast the charge of the main explosive cartridge, it is known as "misfire".

The reasons for misfire are as follows :

1. Defective firing exploder.
2. Defective detonator or bad quality explosive either due to bad manufacture or due to deterioration during storage.
3. Broken detonator leads, defective or broken shotfiring cable, bad connections between exploder and cable, between cable and detonator leads or at other places.
4. Short circuit of the cable or detonator leads due to poor or broken insulation.
5. Where fuse blasting is done, it may be due to (i) wet fuse, (ii) improper timing of fuse, so that blasting by one fuse may cut the fuse of another hole, and (iii) the fuse being drawn out during stemming.

To guard against misfire, the cable should be checked before blasting, the exploder should be examined once every three months by competent persons, and only good quality explosives and detonators, not spoilt during storage, should be used. It is better to have two single-core cables for shot-firing separated by a good distance, instead of a twin core cable, which may be liable to short circuit if the insulation is bad. During use all the connections should be carefully made.

For important blasting work, the circuit should be tested for continuity blasting work, the circuit should be tested for continuity by a galvanometer, described earlier.

Dealing with a misfired shot underground

If shots are fired with safety fuse, no person should enter the site of blasting for 30 minutes after firing. If elec. detonators are used, this time may be reduced to 5 minutes after cable is disconnected from the exploder.

All the entrances to the place should be fenced.

Another attempt be made to blast it by making proper connections of the cable. If the misfired shot does not explode, the shot should be dislodged by drilling another relieving hole at least 0.3 m away from the misfired hole and by blasting it. The new hole should be drilled in the presence of the shot-firer who knows the directions of the misfired shot hole, so that during drilling the drill bit does not touch the misfired charge.

After the relieving shot has blasted the rock, a careful search for misfired cartridges and detonators should be made in the presence of the shot-firer in the material brought down by the shot. If the misfired explosive is not traced, the material should be loaded in a separate tub, distinctly marked, for further search on the surface.

The misfired explosive and detonator, when traced should be destroyed on the surface.

Except in the case of coal mines where the Regulations currently in force prohibit the placing of a second charge in a misfired hole, the stemming may be sludged out with compressed air or water under pressure, and the hole re-primed and fired. The stemming should not, however, be scraped out.

In underground coal mine the **yield** of coal on an undercut face is nearly 5-8 te per kg of explosive and 1-2.5 te per detonator but with solid blasting the figures are 1.8 to 2.7 te per kg of explosive and 0.8 to 1.35 te per detonator.

Some of the slurry explosives of I.D.L. have produced the following results during experiments at a few coal mines on precut face (Pentadyne figures are for solid blasting).

	Coal yield (te)			
	per kg. explosive		per detonator	
	min.	max.	min	max.
Godye (P1)	4.4	10	1.5	2.6
Permadyne (P3)	3.1	6	0.9	1.80
Pentadyne (P5)	1.87	2.72	0.8	1.37

Maximum Shots Fired By A Shotfirer

The mining Regulations fix the following limits on the number of shots to be fired by a shotfirer.

Underground coal mines

Deg. 1 gassy mines - 50 shots with single-shot exploder and 100 shots with multishot exploder.

Deg. 2 & 3 gassy mines - 40 shots with single-shot and 80 shots with multishot exploders.

If a mining sirdar or overman is working as a shotfire also and has more than 30 person working under his charge, he can fire not more than half the number stated above.

Opencast mines

60 shots with single-shot exploder or with ordinary detonators and 120 shots with multi-shot exploder.

The D G M S can put restrictions on the above or give relaxation.

BLASTING IN QUARRIES

Before describing the blasting practice in quarries we shall describe liquid oxygen explosives.

Liquid Oxygen Explosives

Oxygen gas liquefies at - 183°C. A given volume of liquid oxygen, when gaseous, is equivalent to 840 times at N.T.P. i.e. it has as much oxygen as would be available from 4000 times its volume of atmospheric air. If a combustible ingredient, made in the form of a cartridge, is soaked in liquid oxygen and then subjected to reaction takes place with such terrific speed that large volume of a gas is instantaneously released at high temperature so as to cause explosion. The velocity of detonation under suitable conditions of confinement can be faster than 5000 m/sec. This is the principle behind the use of liquid oxygen (LOX) as explosive.

LOX is used for removal of overburden as well as mineral in the quarries of coal as well as metalliferious mines. But its use is prohibited in underground coal mines. LOX is marketed by Indian Oxygen Ltd. in cartridges of two types.

i. Small cartridges of dia. 25 mm to 90 mm

ii. Large cartridges of dia over 100 mm upto 210 mm.

For small as well as large cartridges there is a "standard"cartridge. For small group the standard cartridge is 38 mm dia x 300 mm long. The standard cartridge of large group is called "Full cartridge" and it is 190 mm dia x 600 mm long. The other cartridges are specified in terms of volume of the standard cartridge. In blasting performance, a full cartridge of LOX according to LOX dealers and results in the field, is considered equivalent to 18 kg of conventional explosives.

For those areas where the demand of liquid oxygen is heavy, the Indian Oxygen Ltd. has established central depots equipped with storage tanks and other arrangements of preparing LOX cartridges to be supplied to consumers about an hour or two before charging into blastholes. Such depots are at Bermo, Kathara, Lohardaga, and centrally located centres in mining areas. Liquid oxygen is transported from factories in special vessels on trucks.

A LOX cartridge ready for blasting is prepared at the depots by soaking an absorbent cartridge in liquid oxygen. The basic ingredient of an absorbent cartridge is a cellulosic substance like crushed jutestalk or other agriculture product though other substances such as hydrocarbons or metallic powders, are used to impart to the soaked cartridge the properties of an industrial explosive. The absorbent cartridge for use in coal contains a special composition called "Loxite-C", with a view to reduce the temperature of the gaseous products after the blasting an such cartridges for use in coal are called LOX-C. The absorbent cartridge consists of the above ingredients wrapped in paper covered by cloth. Loxite factory at Ranchi manufactures absorbent cartridges which are kept in stock at the depots. LOX cartridges are not stored in colliery magazines, but are supplied by depots on 2 to 3 hours' prior intimation of the exact requirement for blasting.

For small consumers within 300 km of oxygen factories liquid oxygen is supplied in special containers of 26 litres by train.

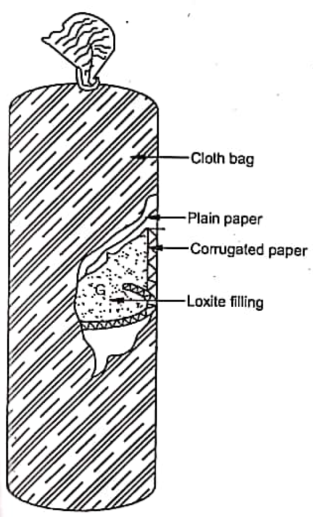


Fig. 8.18

During transit some evaporation, 5 to 10%, does take place. The small consumers have to obtain lixite absorbent cartridges, soaking boxes and other equipment to prepare LOX cartridges on the spot. Only small cartridges are prepared at the quarries in this manner for use in holes drilled by jack hammers or wagon drills.

LOX cartridges are inflammable and the flow of gaseous oxygen emanating from a cartridge will cause smouldering material, glowing coals, and cigarette stubs to burst into flames. LOX should therefore be kept away from such burning or smouldering material.

Characteristics of LOX are not constant. It depends much on the time that has elapsed between removal from the soaking vessels and firing. It is therefore not possible to specify the weight strength of LOX cartridge. LOX does not require a booster for blasting.

The LOX carrtridges should be used in the field without delay (within half an hour in the case of large cartridges) to prevents loss of absorbed liquid oxygen. Grease or oil should not come in contact with the cartridges at any stage. If the oxygen of a LOX cartridge evaporates, the absorbent, cartridge can be used again to prepare LOX cartridge at the depot. The 'Life' of a LOX cartridge in open is about one hour i.e. its full blasting power is available when used within that period but diminishes gradually after that period. The "Life" however, considerably increases in the confinement of the hole. The cartridge looses its oxygen by gradual evaporation if it comes in contact with water.

For use in watery holes only H-type cartridges wrapped in a polythene bag before lowering in the hole should be used. The toe will be not be sufficiently loosened if LOX is used without such precautions in watery holes.

LOX can be fired with or without the help of a detonator. A hole charged with LOX can be fired with safety fuse alone like gunpowder. In small-hole blasting, detonators are not much used as safety fuse is economic. Holes deeper than 3 m are fired by a detonating fuse.

Blasting Practice In Opencast Mines

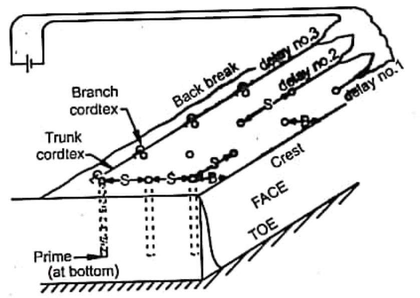


Fig. 8.19 S-Spacing; B-Burden

Efficient blasting should give such rock fragmentation as to eliminate the need for secondary blasting. The fragmented rock should be easily handled by the bucket of the shovel. For mineral which has to go to crushers, the size should not be more than the input size for primary crusher.

Quantity Of Explosive Consumption

The volume of rock broken by explosive is given by the following formula

Rock blasted per hole (solid m^3) = depth of hole × burden × spacing.

Rock blasted by a round of holes = depth × burden × spacing × number of holes in the round

The following table gives an idea of explosive consumption in quarries (O.C.G. or 80% special gelatine) for various rocks under Indian conditions (Solid m^3).

Rock		Explosive consumption kg/m^3
1.	Bituminous coal	0.1
2.	Medium hard shales, sandstones, gypsum	0.2
3.	Sandy shales, sandstone	0.3
4.	Massive sandstone, limestone, laterite	0.4
5.	Very hard shales, marbles, dolomites, limestones, magnesites, hematite	0.5
6.	Extremely hard limestone, conglomerate	0.7
7.	Gneiss, granite, amphibolite, schist	0.9-1

Depth and pattern of blast holes

The depth of vertical blast holes in coal is generally equal to the height of the bench. In hard rock like sandstone, laterite, hematite, etc. the depth should be 0.5 to 1 m deeper below the level of bench floor. This loosens the toe. In any case, the hole should terminate in hard rock and not in the soft one; otherwise the force of explosive is wasted.

Burden is the minimum distance from the face to the blasthole (the term usually refers to burden at the top of the face).

Toe is the projection of the bottom of a face beyond the crest. Sometimes the bottom edge of the bench is referred to as toe.

In hard rocks like laterite and hematite the spacing and burden vary from 0.3 to 0.4 times the height of bench. In less hard rocks like coal and sandstone, the spacing and burden vary from 0.5 to 0.8 times the height of bench. The exact dimensions depend upon the hole diameter, type of explosive used, type of the rock, nature of rock consolidation and the angle of cleats or laminations with the blast hole.

Preparation of primer cartridge and charging blast holes

For blasting with detonating fuse the primer cartridge of a booster, O.C.G. and similar gelatinous or semigelatinous explosive is prepared by pricking a through hole in it, 10-15 cm below the top with a brass or aluminium pricker and threading detonating fuse through it. A knot is made at the end of the fuse in contact with the cartridge to prevent the former from coming out. In the case of LOX, to prepare a primer cartridge no hole is pierced in the cartridge but the detonating fuse is coiled at the neck of the cartridge in the form of a loose knot so that some length of fuse is in intimate contact with the explosive along its length. Aquadyne and Superdyne slurry explosives which are in the form of cartridges closed at both ends by strong rubber bands can be made as primer cartridges by tying a detonating fuse round the cartridge. The short length of fuse at the knot end can be inserted into a small hole prieced in the cartridge by a pricker. The slurry is viscous enough not to flow out of the small hole.

The primer cartridge is always placed at the bottom of the hole in case of blasting with O.C.G. and similar explosives and also in case of LOX, ANFO, and slurry explosives. It is generally lowered by the detonating fuse tied to it. LOX cartridge have a loop of copper wire round the nec. A self detaching hook is attached to the copper wire and the cartridge lowered in the hole. After the cartridge rests in its place in the hole the self detaching hook comes out of the copper wire loop when the self detaching hook comes out of the copper wire loop when the rope is slack. Cartridges of other explosives including ANFO or slurry explosive are dropped or lowered into the hole. ANFO mixture or slurry which is not in the form of cartridges is poured into the hole. In the case of ANFO or slurry explosive a few cartridges of booster or O.C.G. in the middle of the total explosive column or at the top is essential for good blasting performance. The quantity of O.C.G. or other explosive as booster in the total explosive column if ANFO or slurry explosive is used is nearly 15-20%.

Deck loading *i.e.* separating the explosives charges into sections by placing a column of stemming between groups of cartridges is a useful technique for obtaining better rock fragmentation especially where soft and hard rock are encountered in alternate layers. A primer cartridge is placed at the deck charge also and the detonating fuse of bottom-most primer cartridge is threaded through the primer cartridge of the deck charge. (Fig. 8.20).

The drill cuttings in overburden are used for stemming after moistening them with water. The stemming is packed by tampers or a wooden tamping dolly.

LOX cartridges, if they have to be used in watery holes, should be enclosed in polythene bags before lowering. After LOX cartridges are lowered in the hole, sand bags are slipped on top of them to anchor them at the bottom of the hole and to prevent them from floating to the surface as liquid oxygen evaporates. The hole is then completely stemmed by overburden cuttings. Gas bubling through the water should not distract attention as the blasting efficiency is not affected if blasting is performed within a reasonable time.

The detonating fuse is cut after leaving nearly 1 m length outside the hole for attachement to truck line of fuse. The connections of detonating fuse of the blast hole with trunk line are shown in Fig. 8.21. Where two or more rows of blast holes are to be fired in one round, delay detonators are used. Short delay detonators or millisecond detonators are used. Short delay detonators or millisecond detonators give good fragmentation and reduce ground vibrations, the latter advantage being important where quarries are situated near areas having buildings and costly engine foundations.

The blasted rock pieces in mechanised quarries fly upto 300 m. All the workers within 300 m of the blasting site should withdraw to safe places. Warning is given by the shortfirer half an hour before the blasting and again 5 minutes before the blasting by an electric siren, or alternatively by a bugle. All the machinery like shovels, drills etc. should be withdrawn to a place where it is not likely to be damaged by blasted rock or should be suitably protected. The heavy blasting in mechanised quarried should take place during rest interval of workers.

The electrical circuit of detonators is tested for contintuity by a circuit tester which is essentially a galvanometer. When everything is in order, the cable from detonator leads is connected to an exploder and the explosives are blasted from a protected place.

Pump-truck arrangement of Maharashtra Explosives Ltd., for SMS System

In opencast mining where blasting is on a large scale, the SMS system is adopted as a standard practice, involving use of specially designed pump-trucks for transport to the blasting site ingredients required for SMS System. The Site-Mixed Slurry System (SMS), basically comprises a mother support plant where an intermediate non-explosive slurry is, initially, prepared for SMS application. This intermediate slurry subsequently, is transfered to a 10-te capacity stainless steel tank mounted on a 18-te capacity, three-axled 'Ashok Leyland' chassis, suitable for off-highway applications.

On rear side of the pump-truck, two small stainless steel tanks of 150-litres capacity each, are mounted which store cross-linker and sensitiser solutions. Each of these tanks is connected to a stainless steel screw pump powered by a hydraulic motor. The intermediate slurry, cross-linter and sensitiser are simultaneously fed onto a mixing hopper at a pre-calibrated rate, wherefrom it is pumped into blast-holes with the help of progressive cavity stainless steel pump. The pump delivery is connected to a 15 m ling, static-charge resistant chemical hose of 50 mm diameter. All the control, check, relief and regulating valves for the hydraulic fluid are mounted at the rear end of the chassis in a control console.

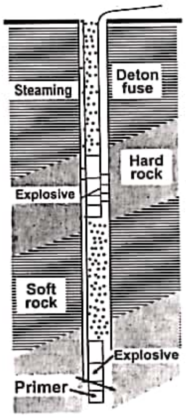


Fig. 8.20 Deck charging

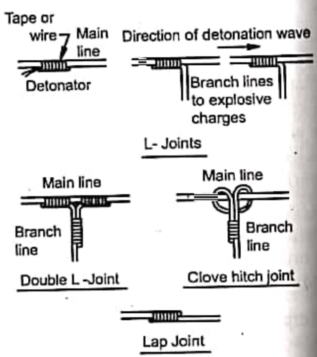


Fig. 8.21 Connections of detonating fuse

For safety consideration, all storage vessels are made of stainless steel. the discharge hose is antistatic. The exhaust pipe is fitted with a

flame-arrestor and a fire screen is provided separating the main body of the pump truck and the cabin.

Computer blasting model : The latest technique in blast design is to use computer simulations which take into account rock properties, blast geometry and explosive characteristics. One such computer model is SABREX, marketed by ICI (India) Ltd. SABREX stands for Scientific Approach to Breaking Rock with Explosives. The essential requirement for running Sabrex simulations is :

1. Five rock properties; Density, Young's Modulus, Poisson's ratio, compressive strength and tensile strength.
2. Explosive characteristics like shock energy, gas energy VOD, detonation pressure, density, etc. A companion programme called CPeX - Commercial Performance of Explosives - calculates these properties for a given composition and density.
3. Blast geometry in terms of hole diameter, bench height, burden, spacing, charge, stemming, delay pattern, etc.

Given these inputs, SABREX predicts fragment size analysis, throw, muckpile profile, damage envelope, flyrock and cost of drilling blasting. The results are displayed in colour graphics and tabulations. The blasting engineer can experiment with adjustments to many factors in a computer safety and achieve results of practical benefit.

1. SABREX accommodates variations of input to all elements of the blast design.
2. SABREX crack pattern is a view of the crack extending from each borehole yielding information on fragmentation, delay periods and backbreak.
3. SABREX gives a picture of backbreak damage.
4. SABREX calculates fragmentation in terms of the percentage of passing size.
5. SABREX calculates 'heave' and builds muckpile profile to assist in subsequent digging operations.

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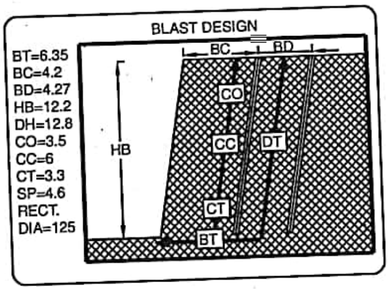

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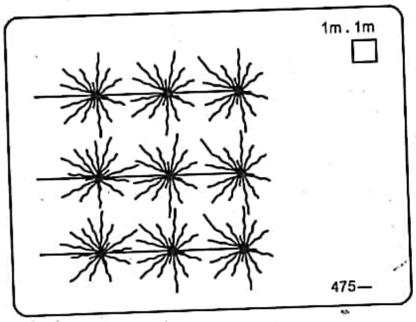

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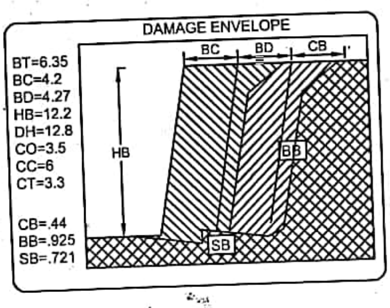

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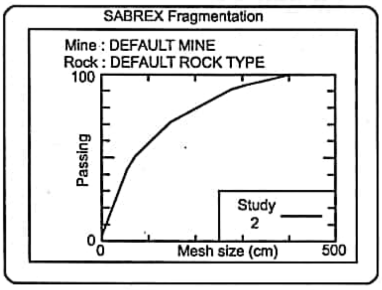

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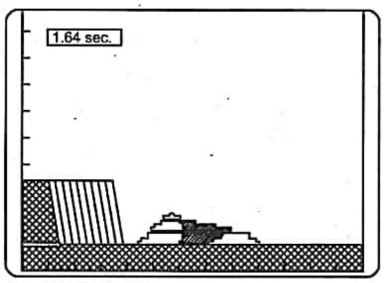


Fig. 8.22

Secondary Blasting

Secondary blasting is carried out in two ways :

1. Pop shooting
2. Plaster shooting

Pop Shooting. A hole is drilled by jackhammer for charging with explosive and blasting the boulder. Normally a depth of 0.3 to 0.6 m is sufficient for most of the boulder sizes. The explosive, widely used, is special gelatine in conjunction with safety fuse or detonators.

Plaster Shooting. A charge of explosive consisting of either a single primed cartridge or a few cartridges is laid on the surface of the boulder. It is then covered with a shovelful of plastic clay which is pressed into position by hand. It is advantageous to wet the surface of the stone before plastering and the clay should be well pressed down for good contact with stone round the explosive. Special gelatine, or Ajax G, can be used for the blasting. I.E.L. has developed an explosive known as "Plaster Gelatine" for this purpose. It is a high velocity, high strength, gelatine type explosive suitable for such work. Plaster gelatine is not in the regular products of I.E.L. but can be available on request I.O.L. also markets a LOX cartridge which is used for contact blasting.

In plaster shooting the explosive charge used is about four times that required for nop shooting.

The object of secondary blasting is to break oversize boulders produced during the first or primary blasting to a size suitable for loading.

Blasting Gallery Technique

This is a new technique for depillaring of thick coal seams.

Blasting gallery technique was adopted at GDK 10 incline of Singareni Collieries Co. Ltd. with collaboration of M/s Charbonnage de France. The method involves drivage of level galleries, and depillaring by drilling holes in a fan pattern around the gallery. The full thickness of the seam is extracted by retreating along the galleries, while blasting and loading out of successive slabs.

Due to the use of longer, the method requires special explosives and accessories. The explosives charge (of upto 3 kg) is distributed in the hole with the help of spacers, and non-incentive detonating fuse is used to ensure reliable of the explosives column.

ICI has developed special explosive - UNIRING - and a non-incendive detonating fuse - RINGCORD - which have passed special tests and have been approved for use with this method.

The details of depillaring by the method of Blasting Gallery Technique are given in the chapter on Thick Seam Working and the reader should refer to it. p. 431)

Coyote blasting : In this system a large quantity of explosive charge, nearly a few hundred tonnes, is packed in the large chambers inside the underground mine. These chambers are made by driving tunnels drifts or raises and are called coyote chambers. After packing explosives in the coyote chambers, the connected tunnels/drifts/raises are backfilled tightly with the muck excavated when forming the chambers and the charge is normally blasted with the help of detonating cord.

Cast blasting : Throw blasting or cast blasting is described as the controlled placement of overburden by drilling and blasting to achieve the optimum overburden removal cost. The technique is suitable for use with large or small draglines and can be adapted for shovel and dragline operations in conjunction with tract dozers to place upto 40% of the overburden in final position.

✧ QUESTIONS ✧

1. What is an explosive and how does it work ?
2. Write short notes on :

   Safety fuse, Delay detonators, Detonating fuse, detonating relay, deck charging, weight strength, booster.
3. State the principle of working of L.O.X. What is a full cartridge and a standard cartridge ?
4. What is the yield of coal per kg of explosive in underground mines and in opencast mines ?
5. What are permitted explosives ? State the approximate composition of cartridge type explosives and mention the factors which make the explosives safe in gassy coal mines.
6. What are the common causes of accidents from explosives in underground mines ?

✧ ✧ ✧

Chapter - 9

Rock Mechanics And Roof Supports

Rocks available in the coal mining areas consist of sand stones, shales, clay and coal seams. Rocks of sills or dykes constitute irregularities which are not present at all the mines. Rocks are called massive when bedding planes in them are more than 1.2 m apart, bedded when between 75 mm and 1.2 m, and flaggy when less than 75 mm.

Rock mechanics is the theoretical and applied science of the mechanical behaviour of rock and rock masses; it is that branch of mechanics concerned with the response of rock and rock masses to the force fields of their physical environment. This definition of Rock Mechanics is widely accepted by the US National Committee on Rock Mechanics in 1966 and subsequently modified in 1974.

Rock mechanics itself forms part of the broader subject of Geomechanics which is concerned with the mechanical responses of all geological materials including soils.

It is generally assumed that the weight of the rocks in coal mining areas exerts a pressure of nearly 0.2 kg/cm^2 for every metre depth from the surface. When working a mine and driving roadways in it, it is, however, the immediate roof, some 6 m above the seam, and the immediate floor, some 3 m below the seam, that matters. Timbering, stowing and to the artificial supports have the object of supporting the immediate roof and sides and prevent sagging of the roof and heaving of floor of the coal seams.

The Pressure Arch Theory

Before an excavation is done in an underground mine rocks in sit are subjected to vertical and lateral pressures which are in equilibrium. In Fig. 9.1 the vertical compressive force is P_1 is to compress the rocks in a vertical direction and as the strata area not able to expand sideways, there is induced a lateral compressive force along the beds denoted by P_2. When an underground roadway is made in the mine, the equilibrium of the pre-mining forces is disturbed and there is a redistribution of the vertical and lateral pressures. New kinds of forces also come into play. This redistribution of pressure is explained by the "Pressure Arch Theory" and is illustrated in Fig. 9.2. In this figure.

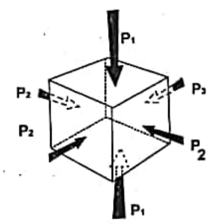


Fig. 9.1 Forces acting on a cube of rock in situ.

B	=	Bending forces whose magnitude increases with width of excavation.
P_2	=	Lateral forces, intensity of which increases with depth.
S	=	Sheaving forces; intensity is highest when coal and floor beds are strong and lowest when they are weak and excavation is narrow.
P_1	=	Vertical compressive forces, proportional to the depth.
C	=	Increased pressure : roadway abutment pressure.

In a narrow roadway, as in driven in the bord and pillar method of working, the vertical compressive forces which were acting in the area of roadway before it was formed, are deflected to the pillars of coal after the formation of the road. The redistribution of pressures takes place as shown by arrows. The rocks in the immediate roof bend downwards under their own weight and tend to separate from one another. This phenomenon is known as bend separation. The weight of higher strata is transferred to the coal sides as shown by the curved arrows CC. At the same time the side pressures P_2, P_2 tend to push the rocks into the roadway. The sides of the coal pillars formed are subjected to shearing forces SS. The immediate roof beds AB, LL, MM and NN act as beams which tend to bend downwards due to their own weight. The lower layers of these beds are, as in the case of any beam, under tension and therefore, liable to fracture. To prevent the lower layers from excessive sagging and ultimate fracture, supports have to be erected in the roadway.

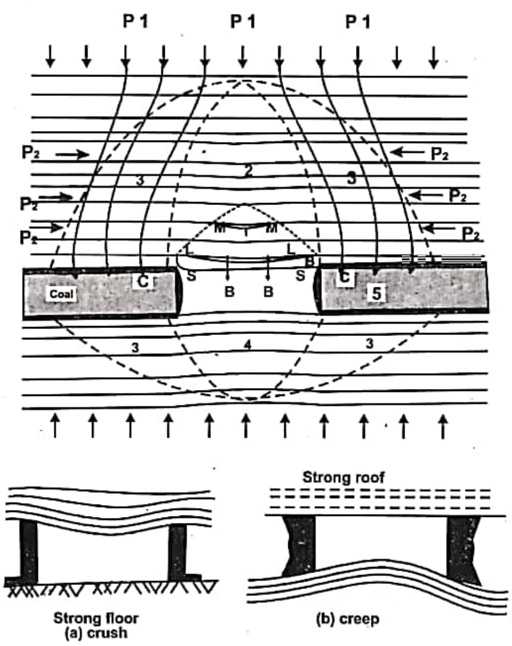


Fig. 9.2

Redistribution of pressures on formation of a gallery in an underground coal mine, resulting in five "zones of influence".

Zone 1 – Bed separation in immediate roof strata due to differential sag of beds.

Zone 2 – No bed separation through the strata sag slightly due to gravity.

Zone 3 – Horizontal and vertical pressures build up to their "pre-excavation" value.

Zone 4 – Floor heaves up but there is no bed separation in the floor.

Zone 5 – Pillar bulges towards the gallery due to release of horizontal stress at pillar sides.

The dotted line represents the "pressure arch" or "pressure dome" with its supports (or abutments) situated at the sides of the coal pillar. The weight of the strata outside this arch is carried by the solid coal and the abutment pressure concentrated at these points is much more than the static pressure, *i.e.* the pressure merely due to depth.

The pressure on the coal pillars is also transmitted to the floor of the coal seam and in the excavated gallery the floor, if not strong enough, may lift upwards and reduce the height of the gallery.

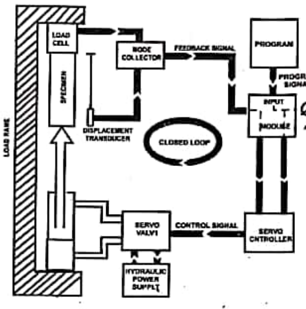


Fig. 9.3 Schematic of fast-response, closed-loop, servo-controlled testing system (Adapted from MTS manual 1970)

At greater depths the coal abutments are not able to withstand the increased pressure if it is more than the crushing strength of the coal and this causes the coal pillars to crush. The crushing commences at the edges of the coal pillars. The weight of the beds within the arch is carried or controlled by the roof supports.

With wide excavations the magnitude of the bending forces B increases and if these bending forces are not kept in check by artificial supports for the roof, the rocks in the immediate roof and downwards, develop cracks and in course of time the cracks become long and wide resulting in breakage of the roof rocks.

As the width of excavation increases, the span of the pressure arch also increases. Beyond some maximum width, which will vary with the nature of rocks and the depth from the surface, sagging of the immediate roof can no longer be prevented by artificial supports and the rocks in the excavation collapse, sometimes right upto the surface. This maximum width of roadway is known as "width of maximum pressure arch".

It will be obvious that a larger pillar will carry a greater load than a number of smaller pillars of the same size.

A number of headings in solid coal relatively close together, as in the bord and pillar method of working, releases the pressure inside the arch but the abutments formed by the extreme sides of the outer headings are under great pressure. In the bord and pillar method of development, as there are galleries along the strike and along the diprise, there is a pressure done spanning the galleries instead of the pressure arch.

Pressure Arch In Longwall Workings

When a longwall face is newly opened between two headings, or gate roads, say, 100 m apart, and the face advances from a solid coal pillar or coal rib, the higher strata in the roof form a sort of bridge (or arch) across the excavated area from solid coal of the face to solid coal of the coal pillar or coal rib behind. Timber or steel supports will support the immediate roof. As the face advances and the space of the arch increases the load due to upper strata increases over the abutments of the arch until the bridge breaks down and the roof comes down in the goaf.

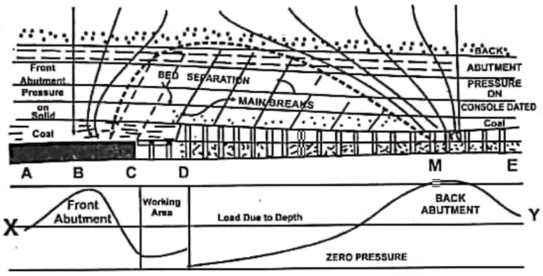


Fig. 9.4 Pressure arch in longwall workings

The redistribution of forces is shown in Fig. 9.4. The dotted arch represents the "Pressure arch". The weight of the higher beds is carried partly by the solid coal at some distance ahead of the face. (between B and C) and partly by the packs at some distance behind the face (between M and E). The pressure arch, thus, has it front abutment in the solid coal and shifts forward as the face advances. It has its back abutment on the packs in the goaf. This back abutment also moves forward as the face advances.

In the figure, the line XY represents the static load due to depth of the seam (called the dead weight of the strata). The front and back abutment pressures are much in excess of this static load. In the working area, CD, the load is greatly reduced and in the goaf the load increases till it reaches the maximum at the back abutment between M and E.

Just as there is a longitudinal pressure arch at right angles to the face, as described above, there is also a transverse pressure arch which spans the whole face length and has its abutments on the solid coal beyond the gate roads.

Experience shows that the pressure arch in longwall workings has its apex or crest at a minimum height above the seam at nearly 2 times the length of the face. If the depth of the coal seam exceeds twice the length the pressure on account of weight of the rocks outside the pressure arch is transmitted to abutments. If the depth is less, the static pressure of the rocks above the goaf are right up to the surface is to be supported by the props, chocks and the packs or stowing in the goaf. Perhaps for this reason longwall faces with caving have caused difficulties in roof control at depths of less than nearly 100 m.

It will be clear that failures of underground coal pillars and roadways are the combined effects of the induced stresses in the surrounding rocks and coal and the inability of the rocks and coal to withstand them. For example, a pillar fails when the applied axial stress in the roof rocks at the edges of a rectangular opening exceeds the shed strength of the immediate roof, it fails in shear. How a rock responds to possible state of stress as well as the reasons for its failure should be understood by engineers planning underground supports and workings.

Underground rocks are generally subjected to a load equal to the weight of the overlying rock. Machines for tests of rock samples brought out in cores of boreholes are generally designed to determine the compressive strength and the most common method of rock testing is the uni-axial compressive strength test. It involves setting a cylindrical (or cubic or prismatic) specimen between the upper and lower platens of the testing machine. The load generated by the testing machine is applied to both ends of the specimen through the platens. The physical properties of the rock determined in the laboratory generally do not present those of the rock mass in situ, be a use in-situ rock mass generally contains planes of weakness, which often break apart during preparation of specimens for laboratory testing. The specimens used in the laboratory tests represent generally the intact portion of the rock mass.

The compressive strength of most of the rocks is greater than the tensile strength. The compressive strength increases with lateral stress; so most test systems are also equipped with a pressurisation system and chamber designed to apply to confining pressure to the lateral surface of the specimen during compression test.

The underground conditions to which a rock is subjected in situ can be simulated to a great extend if the rock specimen undergroes a compression test when it is laterlaly confined by hydraulic pressure on all sides. This is achieved by triaxial load tests. The triaxial tests are

usually performed for cylindrical specimens. In a triaxial cell which is a cylindrical steel chamber (Fig. 9.3) large enough to accommodate a specimen with a length/diameter ratio of nearly two. The confining pressure is applied by using hydraulic oil while the vertical (axial) load is applied by the testing machine. To present oil from penetrating into the specimen, it is usually inserted into a tightly fitted tubing, which can be rubber, plastic or copper.

The modern machines used in the laboratories employ closed loop servo-controlled testing system.

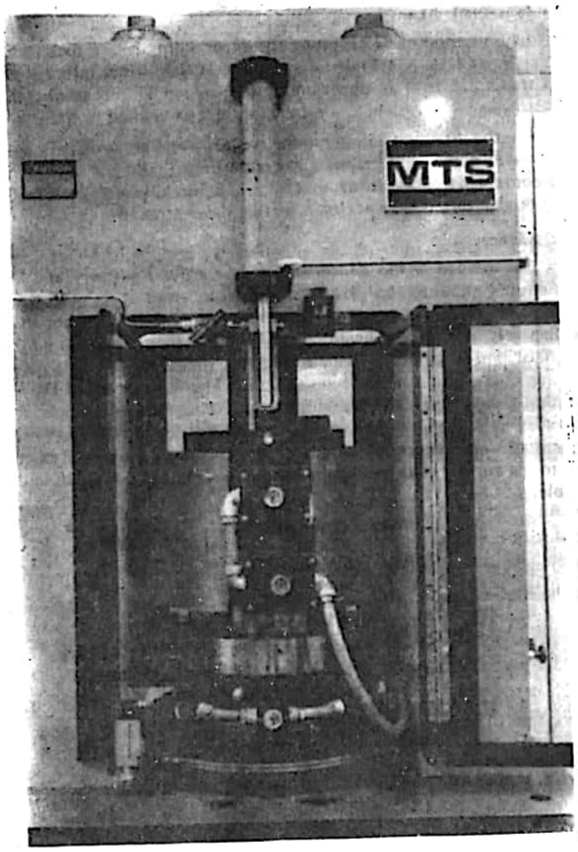
Plate shows a triaxial test cell for specimens up to 54 mm in diameter. It has an internal heater capable of bringing the specimen to temperatures as high as 200°C. (MTS model model 656.01)

This triaxial cell can test specimens upto 150 mm long, and is available with optional platens for testing specimens with diameters small or than NX size. The pressure rating of this triaxial cell is 20,000 psi (138 MPa) maximum confining pressure, and 20,000 psi maximum pore pressure. The unit's internal heater will heat specimens to 200°C typically in less than four hours.

The pressure vessel's internal diameter of 140 mm gives ample room for simultaneous mounting of optional MTS circumferential and axial rock mechanics extensometers. Inside, a Teflon insulating shroud and heat baffles are incorporated to reduce the effect of temperature on the load cell and instrumentation connectors.

The vessel is raised and lowered by hydraulic lifts for easy access to the test specimen area.

One method of determining the shear strength of a rock is the direct shear test. A cylindrical specimen is tight-fitted into a shear box that consist of an upper and a lower piece. During testing, the lower piece is fixed while a horizontal force is applied to the upper piece in the direction parallel to contact plane between the upper and lower pieces and gradually increased until failure occurs along the contact plane. The horizontal force at failure force divided by the cross sectional area of the specimen is the shear strength.



Fig

Subsidence

Subsidence is the depression of the ground surface (and the structures standing on it). In the mining areas it is caused by underground mining if the roof over the void is not supported effectively. The first indication of subsidence is small cracks on the surface which gradually widen resulting in depression of patches in course of time. The thinking of the DGMS and of many mining engineers in the past based on experience, was that there is a safe cover beyond which subsidence is tolerable for the types of structures mentioned in Table 1 as cracks may not extend upto the surface. The safe cover was considered to be as given in the table.

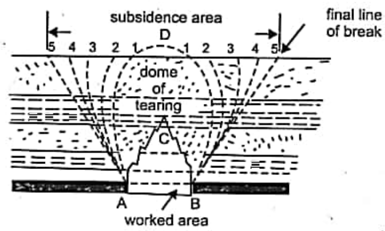
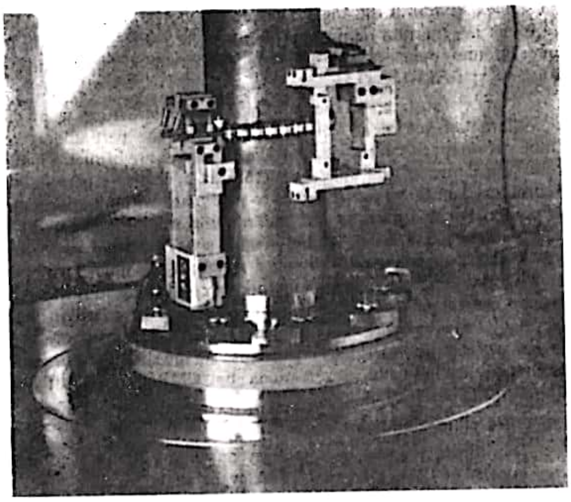


Fig. 9.5 Final line of break and angle of draw

The scientific studies and observations by CMRS have established that in the case of caving the theory of "Safe cover" does not hold good and that the area on the surface affected by subsidence is always more than the area from which coal (or mineral) has been extracted underground. In other words, the angle of draw is always positive in Indian mining conditions.

Angle of draw is the angle between the vertical and the final line of break *i.e.* the joining the point of zero subsidence with the underground point of solid mineral from where excavation of mineral started. (Fig. 9.5).



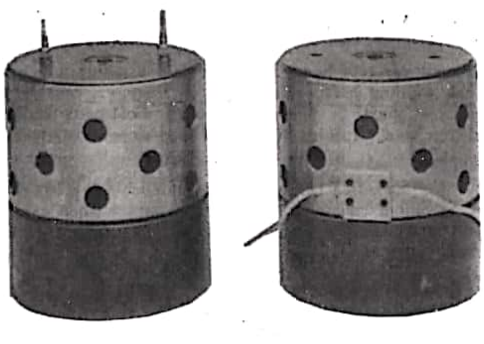


Fig.

Cracks of surface and the buildings or other structures result due to differential subsidence of the surface *i.e.* one part undergoing more subsidence than the other. If the structures are of such a nature that subsidence of the entire structure as a whole is permissible, subsidence can be so controlled as to take place without appearance of cracks. For this purpose the underground mining with caving or partial packing has to be planned in such a way that there is gradual but uniform subsidence of the entire surface as a whole. This is the idea behind harmonic mining, a method of controlling subsidence by proper control of underground mining, and is practised in West Germany for extraction of coal under towns, rivers and railway lines. In our country this idea may materialise in the distant future after prolonged studies and observations by CMRS under different conditions of mining.

Properties Of Various Types Of Roof

The immediate roof in a roadway or workplace in a coal seam may consist of coal, sandstone, fireclay or shale.

Coal roof is common in all seams more than 3 m thick if coal near the floor is worked. It is reliable and with occasional dressing down of the hanging portions will not prove dangerous when it stands for long periods, even a year or more. Compressive strength of coal varies but may be considered as 2.25 kg/mm^2 (1.5 te per square inch).

Sandstone roof bends slightly before breaking and will give enough warning before fracture. It is therefore, reliable and is considered "good" in mining terminology. Coal parts readily form standstone roof. Crushing strength of sand stone is nearly 13.5 kg/mm^2.

Shale roof is treacherous and most unreliable. It rarely gives any warning before collapsing. It weathers quickly when exposed to atmospheric action and spalls off, that is one layer after another comes down at intervals. It is a good practice to keep 0.6 m of coal intact near a shaly roof during development of a mine if the seam thickness so permits. The coal left intact in the roof may be completely extracted when depillaring the area.

Two terms, laminated roof and massive roof, are generally used in mining in case of roof consisting chiefly of standstone. Laminated roof is common in most of the coalfields. During depillaring operations, if the roof is laminated, local roof fall occurs 4 to 10 hours after props are withdrawn. This relieves the coal pillars of local weight and is, therefore, an advantage when extracting the pillars. Massive roof consists mostly of sandstone which has no laminations, e.g. roof in Giridih coalfield. Such roof will not come down easily and will not give enough warning before its break. In the depillaring area such roof will stand for a long period even after withdrawal of supports. When the roof fall takes place, it is generally over a wide area followed by a heavy air blast and crushing of pillars in the vicinity of goaf.

Table 1

A is surface with multi-storeyed pucca buildings.

B is the surface with one-storey building, main river like Lamodar and main railway line.

C is the surface occupied by *kutcha* buildings, jores or tanks.

D is the surface on which sidings and roads are laid.

T is the thickness of the seam being extracted in (in meters).

Method of extraction.	Safe cover for surface			
	A.	B.	C.	D.
1. with caving 2. With hydraulic sand-stowing (Assuming 15% shrinkage)	60 × T	52.2 × T	45 × T	37.5 × T
3. With dry packing (assuming 30% shrinkage).	9 × T	7.5 × T	6.8 × T	5.6 × T
4. Splitting of pillars to the extent that pillars left contain at least 40% of pre-mining coal content upto 60 m depth, and hydraulic stowing of decoaled voids. For every 60 m extra cover, the percentage of coal should be increased by 1%.	5.5 T	5.5T	4T	3.5T

Testing Of Roof

In many situations the roof of mine roadways and faces is required to be systematically supported at regular distances apart, whether or not the places have been tested and found to require supports. Where such systematic support is not considered essential, it is the duty of mine official to test regularly and systematically the places where workmen have to pass or work. This testing is usually done.

1. Visually with the help of a strong light,
2. By hearing the sound of the roof and side or a propset when tapped with a stick, and
3. By feeling the vibrations when the roof is tapped with a stick.

Visual test aims at detecting any cracks in the roof or sides or any signs of weight on supports such as bending of lids or burr of the tapered end of the existing prop, water percolation from roof, or increased flow of water percolation. These indicate the need for additional supports.

A good roof or side gives a ringing sound when tapped with a stick; dull or drummy sound indicates the need for supports. When testing the place, the official should stand to the rise side of the place being tested as loose chunks of rock of bad roof may give way when tapped. The roof above gallery height of 2.5 m may be tested with a bamboo with its testing end shod with iron. Caution is necessary when testing the shale roof, which emits a ringing sound even when the roof is bad, if the thickness of the shale bed is greater than 0.3 m. A prop, if already erected, may be tapped to get an indication of weight on it; a ringing sound indicates good condition but a dull and heavy sound gives a warning of bad roof condition, requiring additional supports.

Vibration test is done by experienced men by resting the palm of the hand against the roof when tapping it with stick and feeling the vibrations, which are different for bad and good roof conditions. This is possible for height not exceeding 2.2 m.

Visual test is carried out if the roof is upto 5 m high by a strong focussed beam of light from a torch or an electric lamp. For vibration test a bamboo end is kept pressed on the roof by one worker. A second worker taps the roof by another bamboo and vibrations are felt on the first. (Fig. 9.6)

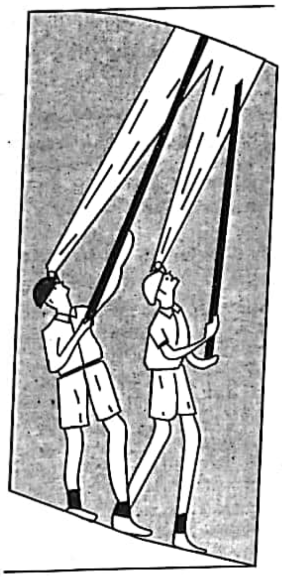


Fig. 9.6

Materials employed for support

The following materials are commonly used for mine supports entirely as such or in combination :

1. Timber, usually sal (and in some areas, teak) is used for props, bars, chocks or cogs, and laggings,
2. Iron and steel in the form of bars, props, arches, corrugated sheets and roof bolts.

3. Brick or building stone in masonry walls, or archings.
4. Reinforced concrete or precast concrete blocks as roadway lining.
5. Roadway ripping, dirt bands and shales as packwalls.
6. Sand, earth, boiler ash, washery rejects, mill tailings, slage from blast furnace for smelting iron and crushed stone as packing of goaf and filling or voids.

Research into strong, lightweight materials for underground supports has shown that the most promising one is glass fibre-reinforced plastics. These have not been introduced in our mines as yet. Tests in Russia have shown the CBAM framed supports or development roads weigh one seventh or one eighth as much as precast reinforced concrete and one third as much as timber frames. CBAM is a Soviet glass fibre product.

Precast concrete assemblies as support have been seldom used in our mines. They have the serious disadvantage of great weight and difficulty in handling.

The type of support to be built up depends on the importance of the place to be supported, the period for the support, its cost and availability.

TIMBER AS MINE SUPPORT

Timber is the most commonly used material for support in a mine as it is cheap and easily available. Dry and seasoned sal (Hindi-Sakhwa) props and bars are used in some parts of Maharashtra and Pench Valley coalfields where teak growth is plentiful. Sal props are available in lengths upto 9 m.

The size of sal props used for roof support are generally as follows

Height of gallery	*Diameter of prop at thick end*
1.2 m to 1.8 m	*100 mm 125 mm.*
1.8 m to 2.4 m	*150 mm 175 mm*
3 m and above	*175 mm 225 mm.*

A timber prop is sometimes called stull in metal mines.

Timber used in the mine is subjected to two main diseases during use.

1. Dry rot 2. Wet rot

The diseased timber is soft and weak in strength. Moreover it warps and is subjected to attack by fungus leading to decay. Decayed timber forming part of mine supports has to be replaced frequently. The unseasoned timber (also called "green") is prone to dry or wet rot. Timber treated with chemicals in special kilns is normally not used in mines and the common method is to dry it in the open for a long period. This causes the sap in the wood to evaporate to a great extent. In the colliery store-yard the timber is so stacked that it is kept clear off the ground, free from dampness and permits air circulation to individual members. Turnover of the timber at intervals helps in maintaining the conditions which are not conductive to attack by dry or wet rot.

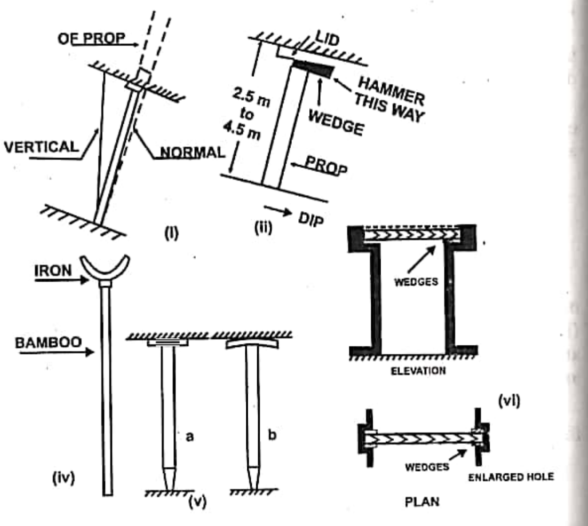
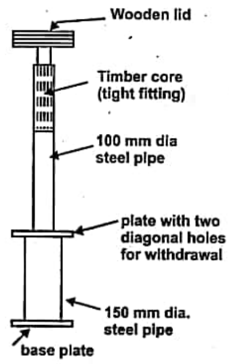


Fig. 9.7

i) Underset prop, ii) Setting a prop, iii) Yielding prop of steel pipe with timber core, iv) fork to hold a prop, v) tapered prop (a) when a erected (b) when under roof pressure, vi) bar for roof support.



A yielding pipe prop. with timber core

Setting Props

A timber prop when erected in a mine to support the roof should yield slightly under the roof weight. The timber prop is strongest when the load acts parallel to its length; a prop as such is almost unyielding but a certain yield is obtained by (a) tapering it at the foot or top, or (b) providing a lid on the top of the prop as a compressible cushion between the roof and the end of the prop.

In flat seams the prop is erected vertical, and in inclined seams, axis of the prop should be normal to the dip of the seam. The prop then offers the maximum resistance to the roof. A prop which is so set that its axis is between the vertical and the normal to the seam is known as an underset prop. Erection of underset props is not common as it is not possible for the timberman to judge whether it is underset or not (Fig. 9.7).

Props should be set on solid floor and not on loose packing or debris except in case of emergency. If the prop has to be set on loose rock, loose coal, or sand pack, it should be placed on a flat base piece not less than 5 cms thick, 25 cms wide, and 0.9 metres long. Such props on loose coal or rock should, however, be removed as early as possible and replaced by props erected on solid floor after cleaning up loose rock or coal. If the floor is soft, the base piece or foot lid mentioned here is used to prevent prop penetration in the floor. In our coal mines the floor is generally hard.

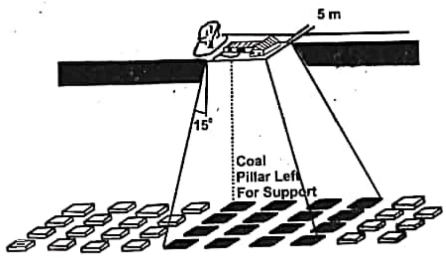


Fig. 9.8 Method of supporting surface structure

Leave 5m on all sides of the boundary of the surface structure and make due allowance for angle of draw to know the underground area where pillars should be kept in tack.

After late P.D. Nath from an article on Roof supports.

A timber prop is expected to support a roof area of about 0.4 m to 1 m all around.

The props are erected on hard floor in seams of varying thickness as follows :

(a) ***Roof height upto 2.5 m :*** Prop is held upright in the place where it is to be erected, a lid placed on top of it (not nailed), and a wedge is hammered between the lid and the prop to tighten the lid against the roof and the floor (Fig. 9.7). Generally for this thickness, the lid and wedge are not used as two separate pieces, but the lid is so made (slightly tapering along the length), that it serves the purpose of a wedge also. If there is any crack or slip which needs support, the lid should be placed across the crack or slip.

A lid serves to distribute the resistance of the prop over a slightly wider area than its cross-section, to prevent penetration of the prop into the roof, and to indicate the load on the prop. It also helps easy withdrawal of the prop. Wedges are used to tighten the prop against the roof and floor. The lids and wedges should be at least of the same width as the diameter of the prop, of a minimum thickness of 80 mm and of a length of 0.5 metre.

(b) ***Roof height of 2.5 to 4.5 m :*** There are two methods :

(i) The position of the bottom end of the prop on the floor corresponding to the place in the roof to be supported is marked by a plumb bob suspended by a bamboo. The prop is held upright by 2 or 3 timber helpers. The timberman standing on a high stool or a ladder places the lid and a wedge is hammered on a position.

(ii) The above method requires a stool or a ladder to be carried from place to place for prop erection. The more convenient method is : a lid is attached to the prop by nails and a hole is made nearly 25mm deep in the floor where the bottom end of the prop is to remain after erection. The prop, laid on the floor with the bottom end in the hole, is made upright, the hole preventing the prop from slipping.

The prop is held in position by timber helpers and one timber helper levers up its bottom end by a crowbar bringing the lid in contact with the roof. A wedge is then hammered between the prop and the floor to tighten the lid against the roof.

(c) ***Roof height above 4.5 m :*** The method described in (b) (ii) is used for roof heights beyond 4.5 m also. The lid is fixed to the prop by nails. The prop is raised upright in its position of erection by helpers, with the help of a fork of the type shown in the Fig. 9.7 (v) or by a rope tied to the prop with a special knot which releases the rope when pulled. The prop is tightened up by the timberman by hammering a wedge at the bottom in the same manner as stated in (b) (ii).

In inclined seams, steeper than 1 in 5, it is usual to fit the foot of the prop into a stamp hole in the floor.

Tapered props : Tapered props are not much used in Indian mines. About 200 to 500 mm length of one end is tapered, and the other end is provided with a lid. Tapered end should be in contact with the hard surface. These props are used where the floor is hard and the roof, soft. With increasing roof pressure the tapered end burrs and provides yielding (Fig. 9.7).

Timber Bars

Bars act as beams. A timber bar is placed in the holes of minimum depth of 500 mm in the side of a coal pillar if the sides are strong (Fig. 9.7). If the sides are weak, the bar is placed on vertical props. If the bar is circular in cross-section, the top end of the prop should be hollowed out to fit the contour of the bar. Alternatively the bar may be flattened at each end so that the ends can rest on flat ends of the props. The bar should always be tight against the roof and should offer the maximum area of contact against it. The side of the bar towards the roof is chopped flat for this purpose. Flat wooden laggings are also used if the roof is uneven.

Safari Supports

The conventional method of supporting galleries in coal mines is by means of wooden cross bars. For fixing these cross bars, holes are to be made in the coal pillars manually by crowbar. This is time-consuming and the whole operation of fixing one cross bar, this method takes about 2-21/2 hours. Therefore the supports lag much behind the working face. For quick setting of the cross bars, the manual cutting of holes in the coal pillars is eliminated by drilling holes with the usual coal drills and a support, known as safari support, is installed to support the roof. This support consists of a pair of clamps of mild steel on which a cross bar is placed to support the roof. Each clamp consists of an angle iron frame to which semicircular m.s. bracket is welded as a seat for the wooden cross bar and in the angle iron two holes, 35 mm. dia. and 175 mm apart, are provided for two m.s. rods of 32 mm dia. 700 mm long. The two m.s. rods of each clamp are inserted into the holes drilled in the coal pillar. The cross bar is placed in position over the two brackets and tightened against the roof with wooden wedges. The complete operation of setting the support is completed in 15-20 minutes for each cross bar. The clamps can be easily recovered and used again for several times. The support stands the effects of blasting and the freshly exposed roof is supported in a short time after exposure. One of its applications is to support the split galleries in depillaring areas in thick seams and extract the floor coal, heightening the galleries upto 5 m. During extraction of stooks, wooden chocks are provided in the spilt galleries and later on the clamps are recovered completely for re-use.

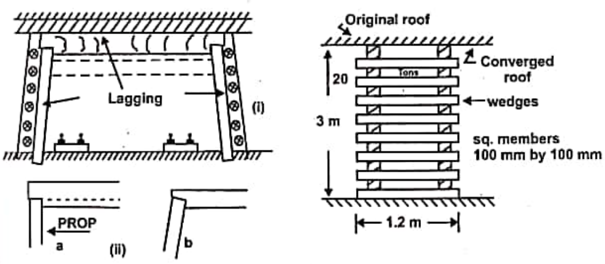


Fig. 9.9

i) Reinforced timber set or chockmate , ii) Notched props iii) Timber cog, chock

Safari supports require strong coal pillars. In a thin seam, the effective height of the gallery is reduced which is a disadvantage where coal-loaded baskets have to be carried on head for tub loading.

Side Support

Wooden laggings are placed tight between vertical props and pillar where the sides are weak and need support. Sometimes the timber set of prop and bar has to resist pressure from sides which tend to crush into the roadways. Notching is useful in such cases (Fig. 9.9). The props should be set at an angle of 14^0 to 20^0 off the vertical and the feet well sunk into the floor.

An alternative method of resisting side pressure is to sink the props well into the floor and to reinforce the timber-set by an additional bar or stretcher (Dotted bar in Fig. 9.9), which may be nailed to the props. As this reduces effective height of roadway, its use may not be advisable in roadways of less than 2 m height, used by basket loaders.

Support By Wooden Cog, Chock or Chockmate

A chock, cog or chockmate is a combination of sleepers above one another in a criss-cross manner as shown in Fig. 9.9. It supports a much larger stretch than a prop and is used in places where the roof is bad over a wide area and needs a substantial support. Cogs are also erected where main roadways have to pass through area having coal pillars of inadequate size. The term chockmate is generally used in metal mines.

Cogs are required under the Regulations at goaf edges, at junctions of splits and galleries in depillaring areas in bord and pillar workings, and at break-off line at the goaf on longwall faces.

Only rectangular sleepers, or alternatively, sleepers having their two opposite sides chopped flat, should be used. The minimum length of sleepers of a cog to support roof at a height upto 3 m may be 1.2 m but for a roof height in excess of 3 m it may be 1.5 m. The sleepers should have a minimum roof at a height upto 3 may be 1.2 m but for a roof height in excess of 3 m it may be 1.5 m. The sleepers should have a minimum cross section of 100 mm x 100 mm.

The cog should not be normally erected on loose floor or debris, but on natural floor or on secure foundation. The floor area over which the cog is to be erected may be excavated for a depth of 25 to 50 mm and should be made nearly flat in seams of mild inclination. Members of the cog are placed at right angles over the members immediately below in pairs. The chock is tightened against the roof by hammering wedges in it at a convenient height. When withdrawing a chock, withdrawal of such wedge loosens the chock and withdrawal of the sleepers becomes easy.

The chock should be tight against the roof and this may be tested by hammering for looseness at the uppermost sleepers.

In some cases a chock is erected in between four corner props. Such corner props are generally not necessary except (a) where the roof is very bad, (b) at goaf edges of galleries (c) at junctions of galleries and (d) if the floor is steeper 1 in 5. Where corner props are erected, one sleeper of the chock is always placed outside the prop. Withdrawal of the member helps dismantling the chock.

Support Of A Roadway

Where the roof of a roadway is bad over some distance bars resting in holes of coal pillars and tightened against the roof by wooden laggings may be erected at intervals of 2 to 3 m. If the coal pillars are not strong enough or the road is through a fault zone, the wooden bars are supported on timber props. This is a common practice. Where the roof pressure is likely to be heavy bars may be supported on timber chocks. Fig. 9.10 shows a method of supporting a wide junction by cogs and bars. No props should be erected at such a place where they are likely to be dislodged by moving or derailed and runaway tubs. The bars may sometimes be placed on bricks walls constructed on solid foundation of coal floor with 150 mm layer of concrete at the base.

Clearing Up Heavy Roof Fall

Such occasions arise sometimes in mines. As a result of roof fall the haulage rope and cables may be burried and if the debris blocks up the roadway right upto the roof, ventilation is affected if there is no other path for ventilating air. If the roadway is the only access to inbye workings men at the workings are stranded. It is therefore, necessary to clear up the roof fall speedily and only experienced timbermen should be entrusted with the job. Power from the buried cable should be switched off.

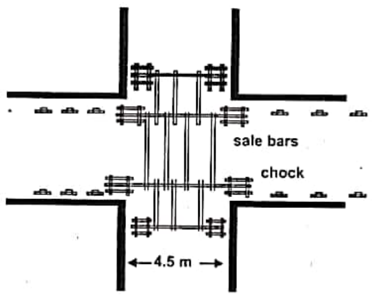


Fig. 9.10 Support of junction by cogs and bars.

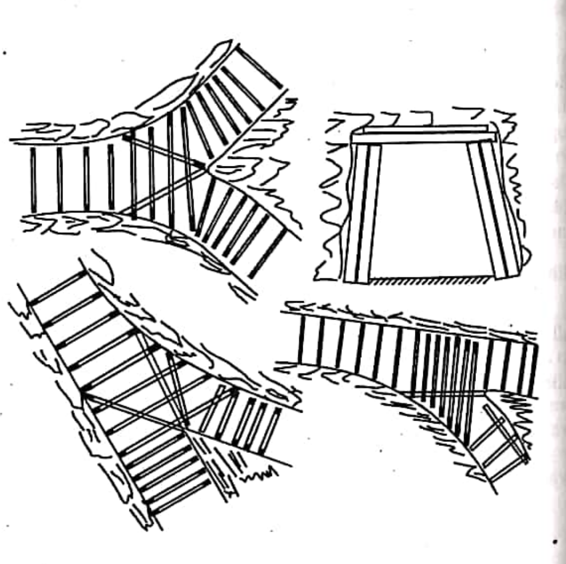


Fig. 9.11 Support of junction by props and bars

In all such cases where the roof fall has to be cleared up work must always be done from a safe place and as the debris is cleared, supports should be set so that the place outbye is always free from danger. In Fig. 9.12 sets of props and bars are erected at a, b, c after dressing loose roof rock and the approaches to the fall made safe. Cogs are erected at the junction as shown at d, and bars placed over the cogs. Standing at the safe place near the cogs, the loose or hanging pieces in the roof are dressed down by the timbermen with a button

The debris of roof fall is then removed by workers and packed up in the nearby galleries but where speed is essential, it may be packed along the sides of the haulage road itself. Temporary props with thick and wide sole plate and top lids are erected on the debris to support the roof temporarily for safety of debris-clearing majdoors. The roadway is thus partially cleared so as to establish ventilation and to remove the trapped men. If the haulage rope is also freed by clearing the debris it is used for expediting the operation by loading debris in tubs. The temporary supports are then replaced by permanent ones like cogs and bars for at least 15 m on all sides of the junction. If the place is important and life of the roadway justifies the expenditure, the place may be supported by brickwall over which girders and corrugated galvanised iron (C.G.I.) sheets are placed. The cavity between the C.G.I. sheets and the roof is packed with boiler ash, or cogs may be erected over the C.G.I. sheets to support the roof above. This permanent measure, however, takes quite some time.

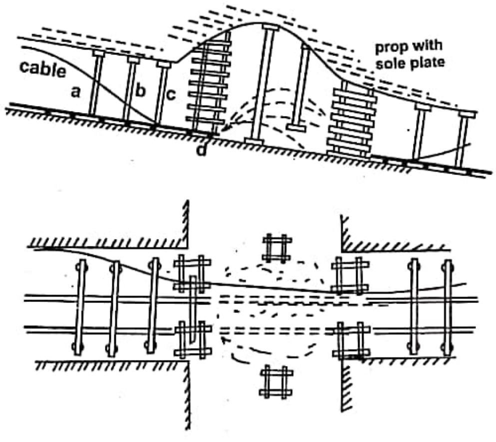


Fig. 9.12 Clearing up a heavy roof fall

Systematic Timbering

"Systematic timbering" is the term used for erecting supports in such a manner that the distances between supports are according to a specified pattern as laid down by the Manager and approved by the Directorate of Mines Safety. Systematic timbering is essential in the district of bord and pillar workings where splitting of pillars or depillaring is going on, on every longwall working face, in every working in a disturbed or crushed ground (*e.g.* fault zone) and at other place where the D.G.M.S. may so direct. The type of supports to be erected, whether cogs, props, or bars are also specified in the order governing systematic timbering. In every case of systematic timbering, it is essential that additional supports shall be erected as and when necessary. Manager has to hand over copies of systematic timbering rules to all the supervising officials and has to post such copies at conspicuous place in the mine.

Withdrawal Of Supports

When props, bars or cogs have to be withdrawn it is prohibited by law to withdraw them by hammering. Suitable safety prop-withdrawer like the sylvester prop-withdrawer has to be used.

It is an advantage to fix the chain C on the prop P G in such a manner before withdrawal that when the chain is pulled and is getting tightened, the prop receives a slight twist – on action which loosens the prop in its position. (Fig. 9.13).

In the depillaring areas, a long chain is often required in place of the chain C. To avoid such long length, a flexible wire rope with steel core (*e.g.* 15 mm dia. coalcutting machine haulage rope) having capels at both the ends may be used in conjunction with the usual 6 m long chain.

Where the roof is high, a suitable anchor prop may not always be available for operation of prop withdrawer. In such case a piece of rail is fixed in a half metre deep hole in the floor. It serves the purpose of anchor prop. Alternatively, a strong bar fixed in the coal pillar may be used to serve the object. Care should be taken to see that the anchor prop or other props which provide safety to the timbermen operating the prop withdrawer, should not be dislodged by the prop under withdrawal when it is released.

IRON & STEEL FOR MINE SUPPORTS

Iron and steel are used in mines in the form of rigid and yielding props, beams and girders, reinforcement in concrete, corrugated sheets and roof/floor bolts. Discussed rails of 36 lbs. section or heavier greatly reduces fire risk in mines.

Steel props used in the miners are (a) rigid props and (b) yielding props. An ordinary H section steel girder of suitable length with the web cut away and flanges turned over at one or both ends is the common type of a rigid prop. If the prop buckles in use it can be straightend by hydraulic pressure situated at the pit bottom. Another timber core through major length and extending 25 to 40 mm beyond the steel pipe at either end. The timber core yields to some extent to the roof pressure and gives indication of roof weight.

Advantages of rigid steel props over timber are :

1. a row of steel props offers uniform resistance if properly set.
2. A steel prop has a greater ultimate strength than timber.
3. Recovery of steel props and frequency of its re-use is greater. If buckled, it can be straightened and re-used but a timber prop once broken cannot be re-used except as lagging.
4. A steel prop does not deteriorate to the extent timber does; timber decays in a number of ways, particularly by dry rot.
5. They are incombustible.

Disadvantages of steel joist props are :

1. Large props are difficult to tighten against the roof and may, therefore be knocked out.
2. A large manpower is required for handling them.
3. Wooden props give warning of heavy weighting, which steel props do not.
4. A simple joist prop cannot be used in seams of varying thickness. Cutting of a long prop and turning its flanges cannot be done at the site of erection.

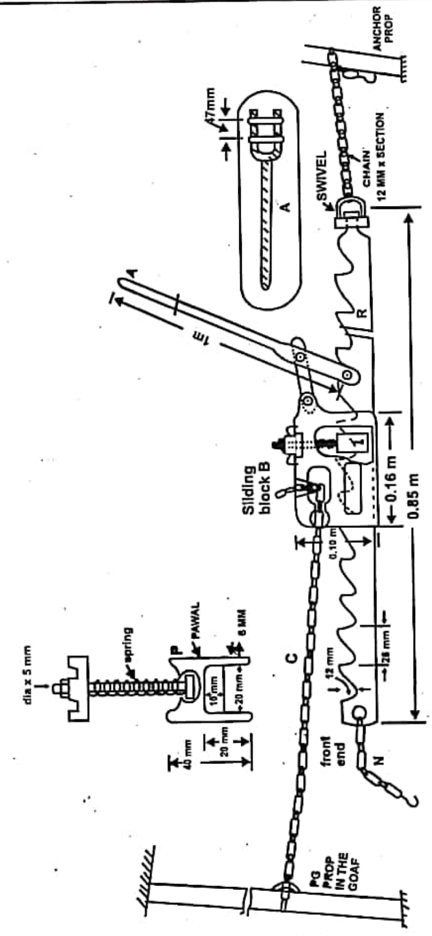


Fig. 9.13 Sylvester prop withdrawer

Yielding Props

Extensible non rigid props or yielding props are used in this country and are manufactured indigenously. They are of two types.

1. **Friction props :** The common example is the FP3 series (Tubular friction prop) manufactured by MAMC, Durgapur.
2. **Hydraulic prop :** The familiar example is the MAMC - Downty prop, TU-40 prop of Usha Telehoist Ltd., *etc.*

Before we describe these props, certain terminology must be explained.

The term *setting load* is used for that load which, by means of a setting device, is imparted to a yielding prop, purpose being to ensure that the support is firmly in place and can resist lateral pressure due to the passage of the face machinery. In other words, setting load is the reaction of offered by the prop to the strata. It is applied to a prop before its yoke is tightened and it is considered essential that it should not be less than 4 te in a seam of moderate thickness (that is, above 1.2 m). The setting load does not affect the load-yield characteristic of the prop.

The *yield load* is that load on a prop at which the upper member begins to slide. Hydraulic props are specified by the yield load *e.g.* a 20-tonne prop, a 30-tonne prop, etc.

In the friction type of yielding props the yield load depends upon the force with which clamps or wedges are tightened. But clamping load is not the same thing as yield load.

Load-bearing capacity of a prop is the load at which an axially loaded prop reaches its elastic limit or at which it begins to buckle.

Characteristic curve or load-yield curve is a graph in which yield is plotted along the X-axis and load along the Y-axis and represents the behaviour of a prop under load.

Early bearing props are yielding props which accept the maximum load with a minimum yield. An early bearing prop (one which is capable of building up of maximum load with a minimum of yield, thus reducing to a minimum both convergence and bed separation) should be used for a routine installation on the coal face while a late bearing prop should be used to stiffen the break-off line. It exerts a resistance which continues to increase with increase in yield. Hydraulic props are early bearing props.

Friction props rely upon the friction grip between two members, one telescoping into or sliding against the other.

If two bodies are held in contact and one moves with respect to the other, the force P for just moving the body in

$$P = \mu \times Q$$

where μ is the coefficient of friction and Q is the normal force. If the upper member is held by two friction clamps, resistance to sliding, P (or in other words, the bearing capacity) of the prop, is $= 2\mu \times Q$.P can be increased by increasing μ or Q or both. In a friction prop Q is raised by means of a clamp which is tightened in various ways, or by compressing parts of the prop by the use of wedge-shaped upper member or by drag wedges which come into operation as the prop yields.

When setting the prop, first operation is to extend the prop to the length required in the position for setting and then it is tightened against the roof or bar by a setting wedge or claw attached to the prop for this purpose. The clamp is then tightened up and the setting device removed.

MAMC Friction Props

The friction props type FP3 manufactured by M.A.M.C. consists of two seamless steel tubes of which the inner member is made captive to the outer member by means of a spring locking pin which prevents the inner member to come out completely from the outer member beyond the extended length. The clamp unit which is fixed at the top of the outer member provides the necessary friction grip by hammering two locking wedges alternately. The inner member and the clamp unit are provided with special coatings which give the required frictional characteristics of the props and at the same time prevent from corrosion. In the closed position of the props adequate finger clearance has been given to avoid the injury to miner's hand during operation. The important features of these props are as follows :

1. These are early bearing props and accept the roof and load very quickly.
2. These have constant load yield characteristics.
3. The weights of these props compared to their nominal load are less in comparison with other types of props.

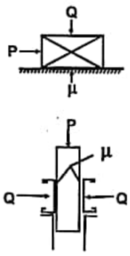


Fig. 9.14

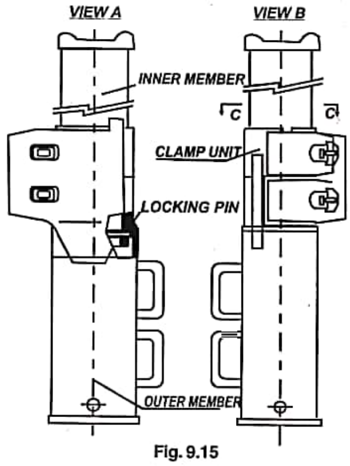


Fig. 9.15

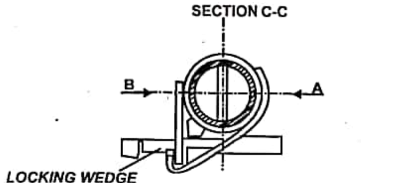


Fig. 9.16 Tubular clamp ring prop (M.A.M.C.)

Friction props are not much favoured these days though they were adopted on the longwall face with sand stowing using Anderton shearter (for the first time in India) at Chinakuri colliery, and later at Gidi A colliery for the experimental method of extraction by inclined slicing with French collaboration. They were also used at Jeetpur colliery on longwall faces with sand stowing.

Hydraulic Props

A hydraulic prop is simply a hydraulic jack. These props have been used at longwall mechanised faces in our country. A hydraulic prop can be set to take immediately three quarters of the maximum load (yield load), but when overloaded it will yield at the designed load after which the resistance is uniform and about 3/4th of the maximum.

A hydraulic prop (Fig. 9.17) basically consist of two oil-filled cylinders, the upper one telescoping into the lower one. A piston head is fitted to the lower end of the top member and this provides a seal with the inside wall of outer cylinder. The piston (and the top members to which it is fitted) does to slide down, unless the load on it exceeds certain limits. The resistance to downward movement of the upper member is provided by the pressure of the oil in the cylinders and this oil pressure builds up by the operation of a pump at the time of setting up the prop in piston. There are two ways of building up the oil pressure in the prop by a pump.

(a) Closed circuit system *(b) Open circuit system.*

In the closed circuit system a built-in pump is provided in the prop itself and forms an integral part of it. The pump is operated by a external detachable handle. In the open circuit system an external pump, serving a number of props from one central site, is connected to the prop by high pressure hose pipes and operation of the pump builds up the pressure of oil contained in the cylinders of the prop. Non-return valves provided on the prop retain the oil pressure. The largest manufacturer of hydraulic props in the country. M.A.M.C. Durgapur, manufactures props of both designs, *i.e.*, of the closed circuit system (Example- MAMC Duke hydraulic props) and also of the open circuit system (example-Salzgitter/MAMC hydraulic props).

The MAMC hydraulic prop consist essentially of an inner tube, a pump cylinder with oil, a guard tube, a release valve, a non-return valve, a main piston, a top extension fitting, and a pump and release shaft. The inner tube and pressure cylinder can be extended like a ram

by hydraulic pressure. The pump can be actuated by an outside key or handle. A large diameter steel tube connects the lower reservoir with a relief valve capsule. Action of the handle pumps the oil from the inner tube to the outer tube, thus extending the inner tube. After the prop has been so extended up to the roof, further operation of the pump handle provides the initial bearing pressure (or initial setting load) which is 5-8 tonnes.

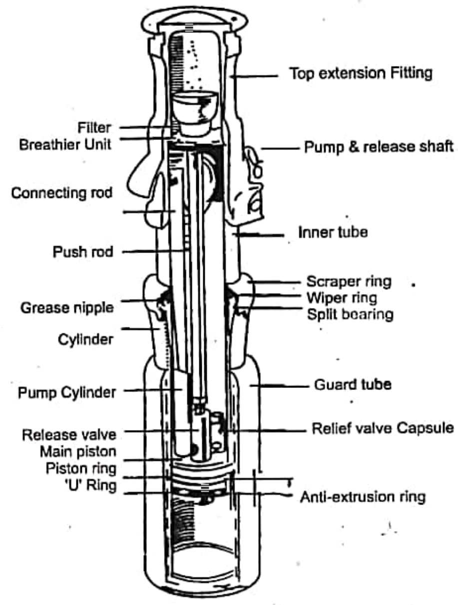


Fig. 9.17 MAMC-Dowty hydraulic prop

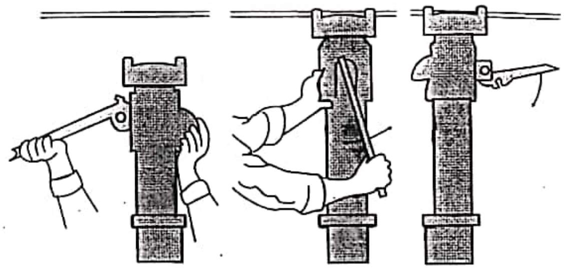


Fig. 9.18 Hydrulic prop., with built-in pump.
Left-extending the prop, Middle-setting the prop., Right-withdrawl of prop.

The pumps handle is then withdrawn. In course of time when the roof pressure on the prop gradually increases the inner tube will not slide down till the load is 40 te, (in the case of 40-te prop) as the hydraulic fluid is compressed till that load is reached. When the load on the prop exceeds 40 te, a relief valve operates. The relief valve is a capsule permitting adjustment and testing prior to insertion in the prop and it is set to the correct yielding pressure before the prop is assembled in the factory. During the yield, the oil pressure may be from 200 to 500 kgf/cm². The prop can be withdrawn easily by pulling the release shackle, which actuates a cam and lifts the valve assembly off its seating, allowing a free flow of oil back to the top reservoir. Withdrawal can be effected from a distance by attaching a chain to the relief valve shackle and pulling it. A hydraulic prop can be tightened in a few seconds to any length within a wide range. The pump handle is the only tool required for operation and the prop can be released and withdrawn in a few seconds. The hydraulic oil used may be water with 10% suitable oil. The oil is DTE light oil, servo system 311 supplied by Indian Oil Corporation. In has anticorrosive characteristics.

M.A.M.C. manufactures 40-te props of its own design.

A precaution which must be borne in mind in connection with hydraulic props is that they must not be left lying on the floor, and the prop must be withdrawn before full closure *i.e.*, before it becomes

"solid". On longwall faces having sand stowing, hydraulic props which were not used carefully, developed scratches due to sand rubbing on the inner cylinder thereby partially losing the oil-sealing capacity. Such neglect renders the prop ineffective.

Salzgitter/MAMC hydraulic prop.

This is a hydraulic prop of open circuit type, normally used with sliding roof bars (Fig. 9.10). The roof bar itself can be pushed or retrached, when in position, by hydraulic pressure. The Salzgitter/ MAMC hydraulic prop can be extended, according to prop type, by mounting of extension pieces.

Setting of Salzgitter/MAMC hydraulic props is effected by hydraulic pressure, which, produced by a central high-pressure pump, is delivered to the setting gun via a lead in the supporting system. The setting gun is put on the filling valve of the prop, mechanically fastened by one-hand locking device and operated by one hand.

The pressure medium flows under the piston, pushes out the inner ram tube and sets the prop safely with the setting load pre-set by the hydraulic pressure between cap and bottom. The hydraulic oil used is a 5% oil emulsion, and 95% water; pH value 5-8 pH.

The prop can take the roof load until the nominal load adjusted in the operating valve has been reached. As soon as the nominal load has been reached, the operating valve opens and some pressurised fluid spurts out until a pressure spring closes the operating valve again. This operation occurs whenever the nominal load has been reached. Thus the prop is always protected from over-load of rock pressure.

Large stroke admit large convergence as well. Due to the extension feasibilities according to prop type, the prop lengths can be easily adapted to seam heights underground.

When the prop is to be withdrawn at the goaf edge, the releasing valve can be actuated by means of a releasing key which is extended by means of a rope or chain from the miner's safe position effecting the flow-off of the pressure fluid out of the cylinder space. An installed powerful return spring rapidly retracts the inner ram tube. After this the prop is ready again for another setting process. Releasing keys are available in accordance with service conditions (service heights) in lengths of 150, 280 and 800 mm.

Technical Data **(of one of the props) ; Salzitter/MAMC prop**		
Type	:	Aluminium-ally prop, IIS40/L
Nominal Load (Tonnes)	:	40
Material	:	Aluminium-alloy
Min. Yield Strength	:	45 kgf/mm²
Inner Ram Tube (mm)	:	105 × 13
Outer Ram Tube (mm)	:	138 × 14
Piston Area	:	95 cm²
Stroke	:	800 mm
Foot plate	:	180 mm dia. (Area 250 cm²)
Head plate	:	40-prong crown
Length (extended)	:	2500 mm
(retracted)	:	1700 mm
Weight	:	62 kgf

Comparison Between Friction And Hydraulic Props :

1. A hydraulic prop is by far superior to any friction prop in so far as acceptance of a sufficiently high and uniform load in the underground application is concerned. The friction prop has irregular behaviour under load and it slides more than a hydraulic prop under a similar load.

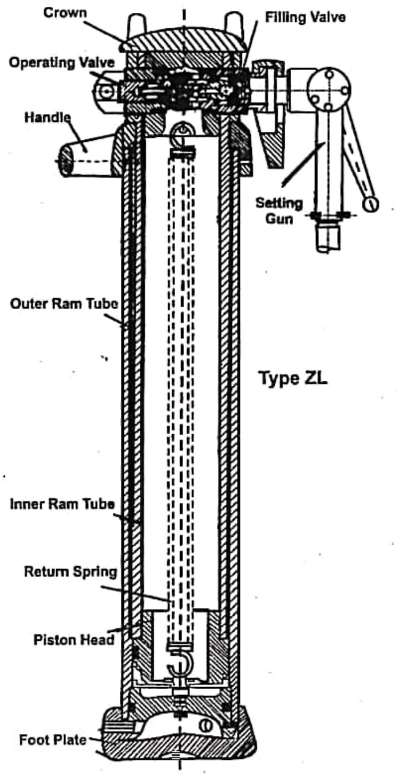


Fig. 9.19 MAMC/salzgitter prop.

2. A high setting load is obtained in a hydraulic prop.
3. Friction prop is lighter in weight and has a superior extensibility.
4. Friction props serve a long life of sand stowing faces.
5. Friction as well as hydraulic props can be easily withdrawn.

Wooden lids are not used with friction props and hydraulic props. Where such props have accommodated steel roof bars, the prop heads have prongs which prevent the prop from being joilted or forced out of position by an accidental heavy blow or roof pressure.

CMRS Designed Hydraulic Prop

Most of the hydraulic pros for use in mines do not incorporate any device (i) to indicate load coming on the prop at any particular moment, and (ii) to change the capacity of the prop without dismantling it. CMRS has developed a new type of hydraulic prop in which these two devices have been incorporated. Other notable features of this prop are : (i) if any part of the prop gets damaged underground, it can be easily repaired without taking it to the surface, and (ii) the load indicating system also acts as a safety release capsule and can be operated when the release valve gets damaged during working conditions.

Prop density is the term used to denote the number of props required to supports one square metre of the roof area. This is the practice adopted in Germany.

In U.K. prop density is frequently referred to as the area of roof supported per prop. The prop density is calculated under least favourable conditions.

Bars : A steel bar or beam is used when there is a prop at each end. A corrugated bar is stronger and offers more resistance to bending. On the prop-free-front system of roof support, cantilever bars are used, thereby reducing the number of props and the risk associated with their installation. They are supported by a prop in the middle or at one third length from one end behind the conveyor (on the goaf side), the other end remaining unsupported and therefore liable to bend slightly with roof weight. A cantilever bar has a length which is a multiple of the face advance per cut by coal cutting machine or shearer. Two or three bars can be connected Fig. 9.20.

Sliding bar is a long rigid light section steel girder which rests in special heads fitted to the vertical props and slides between the vertical slide posts of the heads. The bar is raised and tighetened against the roof or lowered or released from roof pressure prior to sliding into the next position by means of a sliding wedge.

Bar slide method is used in the prop-free front. At the start of cut, the bar resting on the heads of three props, is raised and tightened against the roof by means of the wedges. As coal is loaded away the bar is dropped from the roof by releasing the wedge and pushed forward. The bar is re-tightened in the new position and if necessary, temporary prop is erected under the free end of the bar next to the face. Conveyor is then advanced and a prop is fixed in the new position just behind the conveyor. During the period the bar is released and advanced, the roof load is carried by props as if there was no bar. In case of an irregular or weak roof sliding of the bar presents difficulties.

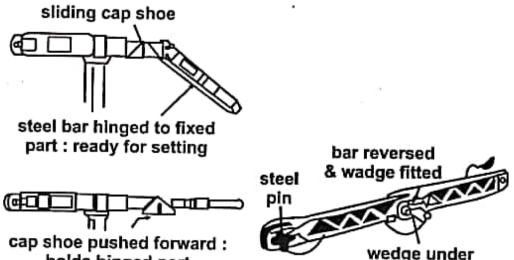


Fig. 9.20 Cantilevel bars of capped shoe type and (bottom) of wedged type

Though the hydraulic prop is a powered supports, the term "powered support" is confined to cover such supports as the hydraulic chock, shield or canopy.

Self-advancing or walking supports

These are open circuit type hydraulic chocks which, when already erected at a place, can be retraced by hydraulic pressure, pushed to the new site of erection by hydraulic shifting cylinders and erected by hydraulic pressure. The hydraulic pressure for chocks and for the operation of the rams is provided from a centrally located pump near the face. A workman is not required to handle the supports during any of these processes, except for guiding the roof bars supported by the hydraulic props. The self advancing supports are used in coal mines on prop-free front longwall faces and are introduced for the first time in India in Monidih colliery.

The walking ability of a powered roof support is provided as follows :

There are two distinctive sets of one to four hydraulic props connected by a common roof canopy and floor base. The sets are connected at the base by a horizontal shifting cylinder. The support moves or "walks" when one set of props or "legs" is lowered and the shifting cylinder is actuated, while the other set of legs remain firm against the roof. After the first set moves through a predetermined distance, approximately 0.6 metre, its legs are set against the roof. The same operations are now repeated for the second set, such that the whole support is self advanced through an approximate distance of 0.6 metre. The face conveyor is advanced by the double acting rams set between the supports, usually in alternate support units.

4-Leg And 6-Leg Canopy Supports

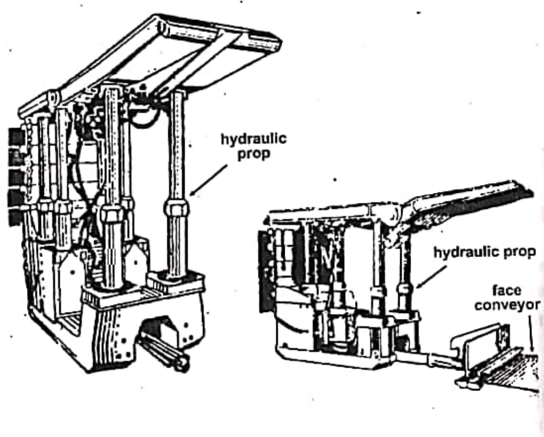


Fig. 9.21 Gullick Dobson supports. Left-6 leg chock; Right-4 leg chock

A combination of 2,4 or 6 hydraulic props on a common rigid base is manufactured by some companies to provide support which can withstand very heavy roof pressure. Fig. 9.21 shows a 6-leg, 240 te support and a 4-leg, 310 te support manufactured by the firm of Gullick Dobson. The 6-leg support has particular application in coal seams between 1450 mm and 2360 mm in thickness. The legs may be single telescopic or double telescopic. A 4-leg support is bulkier than 6-leg support and is available for yield load of 310 te; 590 te, and 728 te. These supports provide a canopy to the armoured chain conveyor and the coal cutter or shearer mounted on it. Between two units of such supports there is a gap of only 70 to 150 mm so that the entire face is well supported and there is no exposed roof along the length of the coal face. Such heavy support are retracted, advanced and are re-erected by hydraulic pressure with the help of rams, (one ram serving one support) by the workers standing under the adjacent support. Operation is effected by a single lever 'Dead Man's Handle' type control valve.

The support is attached to the armoured flexible conveyor (A.F.C) by a double acting hydraulic ram which gives a 6.3 tonne differential push load for conveyor advance, and 7.9 tonne pull load for support advance and has a 787 mm stroke.

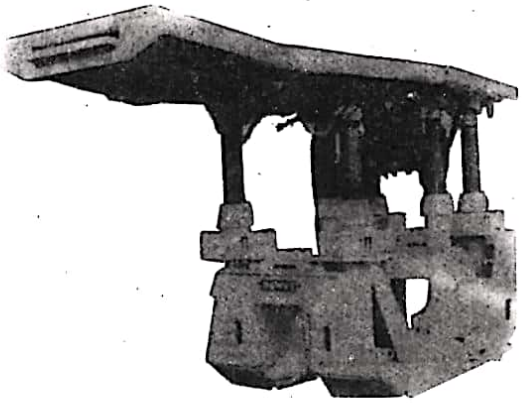


Fig. 9.22 Dowty-4-leg, 280 te chock

RANGES AVAILABLE UPTO 2800 mm EXTENDED

Anti-slew equipment

Such supports can be fitted with anti-slew equipment and provision is made for attaching brackets to the base for it. Sliding links act on a guide rail between two chocks to maintain alignment. The guide rails are attached to the conveyor between alternate chocks.

Table

FAULT FINDING

If a support develops a fault, it is important to locate it and rectify it, or change a defective component with the minimum interruption to normal working. The following paragraphs provide a guide to the possible causes of failure under various operating conditions. It is assumed that the power pack is functioning correctly and that all main pressure and return line stop cocks between the support and the power pack are open.

1. Failure of Advance	(a) Canopy in contact with roof : Lower (b) Front cantilever tip in contact with roof : Lower further before advançing. (c) Front cantilever fails to retract : (d) Bad floor cutting (steps). (e) Disconnected or broken piping : Re-current or renew. (f) Rotary selector valve defective : Change valve (g) Leaking ram gland seal : Change ram. (h) Piston rod bent : Change ram.
2. Failure to Set (Auto set)	If all legs fail to set, possible causes may be : (a) Faulty rotary selector valve : Change valve. (b) Faulty release/yield valve : Change valve. If a pair of legs fail to set on either adjacent or manual setting the possible causes are : (c) Defective piping or connections ; Find source of leakage and tighten connections or renew defective components. (d) Faulty release/yield valve : change valve.
3. Failure to maintain leg pressure after setting	(a) Leaking bonded seal or hose connection : Tighten or renew if leakage persists. (b) Leg gland sela failure : Change leg. (c) Faulty release or yield valve : Change release/yield valve.
4. Failure of legs to lower	(a) Corrosion or dirt on inner tube : Clean off without damaging plated surface. Change leg if corrosion is servere. (b) Insufficient pressur to operate release/yield valve : 48 bar (700 p.s.i) required. Operate release/yield valve handle manuaily. (c) Leaking leg piston gland : Change leg. (d) Bent leg or sized leg bearing : Change leg. (e) Faulty release/yield valve : Change valve.
5. Ram fails to advance conveyor	(a) Excessive floor steps for conveyor to climb. (b) Defective pipeing or connections : Find source of leakage. Tighten loose connections and renew damaged components. (c) Leaking ram gland seal : Change ram. (d) Ram piston rod bent : Change ram. (e) Faulty rotary selector valve : Change valve.
6. Cantilever fails to retract	(a) Cantilever tip in contact with roof : Lower legs. (b) Cantilever ram piston rod bent : Change ram. (c) Leaking ram gland seal : Change ram. (d) Faulty rotary selector valve (support also fails to advance) : Change valve.
7. Cantilever fails to extend	(a) Cantilever ram piston rod bent : (b) Leaking ram gland seal : Change ram. (c) Faulty control valve : Change valve.
8. Unit sluggish on all operations	(a) Stop cocks in return line partially closed and low pressure relief valve blowing : Open fully. (b) Stop cock in main pressure line partially closed : Open fully. (c) Trapped pressure or return line hoses : Clear.

Shield Supports

The shield supports provides a continuous cover all along the face by canopies placed side by side. In this respect it is like the multi-leg hydraulic chock support except for a little difference in construction at the rear side *i.e.* the goaf side. The hydraulic powered chock has a venetian blind type flushing shield suspended from the rear of the main canopy to protect the chock from the falling debris of the goaf side whereas a shield has a sloping rigid one-piece roof of adjustable inclination on the goaf side. This results in elimination of side loading on the legs.

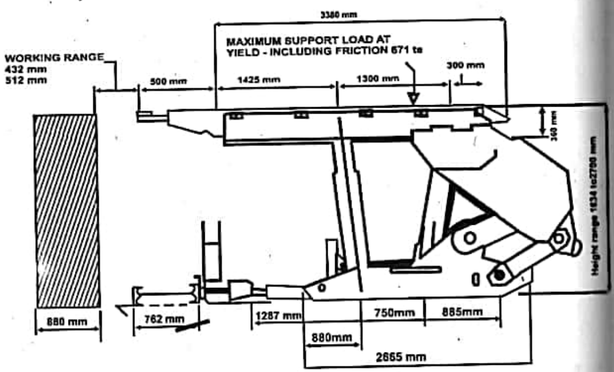
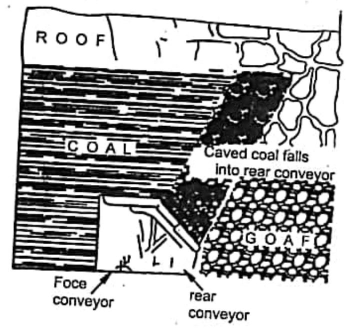


Fig. 9.23 A 4-leg, 630-te Chock Shield (downty)

The shield supports are with 4 legs (not with six) which react direct with the canopy. The base of the shield is of rigid construction incorporating fabricated sockets for locating the legs. The lemniscate linkage formed by the shield structure and links limits the movements of the canopy in the face towards goaf. The linkage also resists forces exerted on the canopy by horizontal movement of the roof.

Where mining conditions are suitable, *i.e.* sound roof with minimal lateral roof movement, then a chock type support could be the answer at a considerable cost saving over a shield.

One type of shield support marketed by Downty is 4-leg, 400 te sub-level caving shield support. It is suitable for sub-level caving in thick seams or for conventional operations in longwall installations.



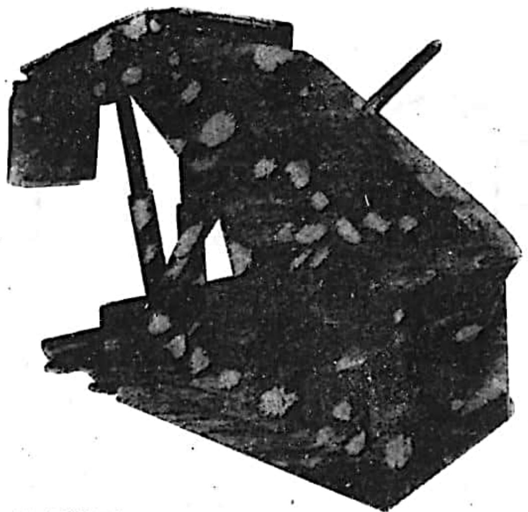


Fig. 9.24 Basic system of Sub-level caving with shield support. Typical seam thickness upto 10 m.

(Fig. 9.24). Apertures are provided in roof canopy of the shield for drilling of large coal lumps and subsequent blasting. The sloping shield has an integral coal loading door which is hydraulically controlled from a safe position within the support and can be closed at any time during coal entry. The shield also protects the rear conveyor from falling debris.

A hydraulically operated agitator arms is fitted to the rear shield to assist the flow of coal, if it does not fracture readily. Hydraulically operated canopy side flamps effectively seal the gaps between the supports and further exclude the dirt and dust from entering the working area. The support provides a clear traveling way in front of the legs from which all major operations can be controlled.

Hydraulic shields are costlier than chock supports.

Steel Arches

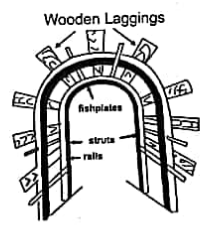


Fig. 9.25 A steel arch

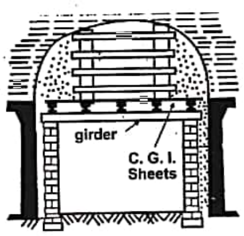


Fig. 9.26 Support of a high roof

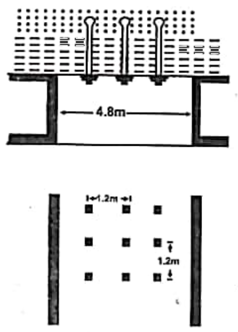
These are used for supports of permanent and semipermanent roadways. Heavy section rails in two parts are suitably shaped in workshop to form an inverted U when assembled together. (Fig. 9.25) The two sections, after installation underground in the place of erection, are joined by fishplates and four bolts and nuts. The legs are generally placed in holes made in the floor. Normally no sole plate or lid is placed at the foot of arches and the arch has no yielding property. Where a roadway has to be supported by a number of steel arches struts are placed between adjacent arches to prevent lateral shifting

To keep arches in contact with the roof and sides wooden laggings are placed on the top and sides in the same manner as for timber supports.

In our mines steel arches are not erected in places where strata may have a tendency to descend, and such arches with yielding property are not required.

C.G.I. sheets are used in the mines mostly to cover bars of girders over which cogs may be erected for support of a high roof, or sand/obiler ash may be filled to pack up a cavity in the roof in a mine as shown in Fig. 9.26.

Roof Bolting



The term **roof bolting** is applied to the practice of drilling vertical holes in the roof and fixing steel bolts into them in such a manner that the bolts grip the strata and support the immediate roof. The bolted roof strata behave as one thick beam capable of supporting not only their own weight but the weight of the strata above Fig. 9.27 shows the general scheme of supports by roof bolts.

Fig. 9.27 General scheme of support by roof bolts

There are two types of roof bolts in common use :

Slot and wedge bolt. It consists of ms rod 25 mm dia., nearly 1.3 m long, split at one end for nearly 160 mm, and threaded at the other end for about 125 mm. Into the slot is fitted a wedge 150 mm long. To install the bolt in position a vertical hole, nearly 125 mm less than the length of the bolt, is drilled and the bolt, having the wedge fitted at the split end, is inserted into the hole and hammered in position. The split end having the wedge somewhat widens in the hole and grips the rocks. A bearing plate of mild steel, 150 mm × 150mm is placed on the bolt and a nut is tightened.

The better the grip, the sounder is the installation.

This type of bolt is popular as it can be easily made in the colliery workshop, is cheap in the first cost and can be installed in holes drilled by the usual coal drill. It cannot be recovered.

Expansion shell type bolt.

One of the expansion shell bolts works as follows :

Fig. 9.28 shows a forged head or stud type bolt complete with expansion shell and plug of Nanda Manufacturing Company. Completementary taper of plug and shell provides full contact with the wall of hole. Its installation is as follows :

1. Drill hole of recommended diameter at right angles to bearing surface, to a depth larger than the bolt to be used.
2. Place washer on bolt thread. Plug on bolt four to eight turns.
3. Insert in hole until washer and bolt heads are against mouth of hole.
4. Apply torque as per recommendation. This expands the serrated shett against the wall of hole.

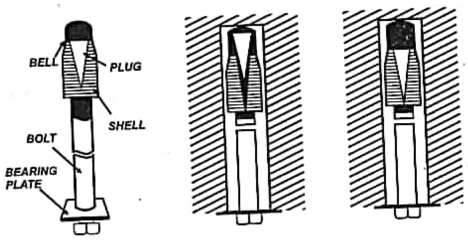


Fig. 9.28 Dexpansion shell bolt

Perfo Bolts (N.M.C.)

A ribbed/torsteel rod is threaded at one end and chamfered at the other end for easy insertion. Sleeve of recommended diameter made of flexible galvanized wire mesh (size 5 mm x 1 mm dia.) is used to insert the sand cement mortar into the drill hole.

Two semi-cylinders are manufactured from 0.63 mm thick perforated steel sheet; alternatively wire mesh cylinder is manufactured from 5 mm mesh x 1 mm dia. They are usually 3 to 4 mm less than bore hole dia. It is advisable to use perforated semi cylinder tubes for lengths longer than 1500 mm.

For installation of a perfo bolt, drill hole longer than bolt to be used. Take wire sleeve and fill it with mortar (cement, sand and water 1:1:0.6 by vol.) For the remaining procedure see Fig. 9.29.

Roof bolts

(a) Slot & Wedge type
(b) Expansion shell type
(c) Perfo type
(d) Resin Bolt with Capsule
(e) Split set Bolt
(f) Recoverable Bolt
(g) Quick Setting Inorganic Cement Capsule

Roof bolts manufactured by Nanda Mfg. Co.

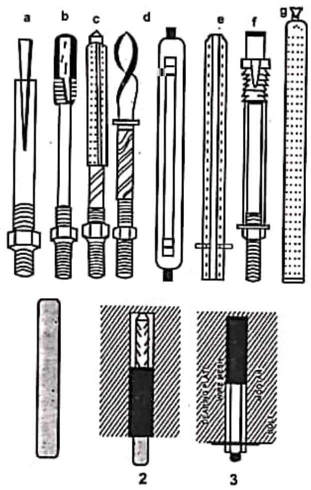


Fig. 9.29 Perfo bolt

Principles of action of roof bolt

There are three theories put forward to explain the action of roof bolt.

1. Beam theory
2. Keystone theory
3. Suspension theory

The beam theory is widely accepted. According to this theory, the roof bolt assists in joining together the rock beds through which the bolt passes. The joined beds form a sort of thick beam which supports the weight of the strata above. For example, if a single wooden plank, say 6 mm thick and 6 m long is supported at its two ends, it will bend considerably in the middle under its own weight, and may even break, but if several of such planks are clamped or bolted together, the composite beam so formed will be much stronger than the single plank and will support not only its own weight, but also some heavy weight over it.

For deficient roof support bolts in sufficient number should be fixed immediately after the roof is exposed, *i.e.* immediately after loading of coal in an advancing heading is over. Roof bolts are not very successful in watery holes as the grip of bolts is reduced.

During roof bolting instead of a plate, a girder or a bar may be used to support the roof. The girder or bar provides supports to a crack or slip in the roof.

Advantages of roof bolting

1. It is simple to apply, easy to mechanise and moderately cheap in cost; manpower required for fixing supports is less than with conventional supports. In the U.S.A., the complete roof bolting equipments are mounted on a single self-propelled trolley.
2. It gives greater headroom and clearance in the roadway. This facilitates easy manoeuvring of the machines, e.g. shuttle cars, coal cutters, mechanical loaders etc. and offers least resistance to the air flow ;
3. The hazards due to accidental dislodgement of conventional supports caused by derailed tubs, blasting, etc. are reduced if systematically carried out. It results in greater safety and less accidents due to roof falls. The supports are fireproof.
4. Handling and transport of heavy support materials involved in conventional supports are eliminated;
5. Storage space required is small;
6. The protrudings ends of bolts can be used to suspend water or air pipes and cables ;

7. Stability of the support does not depend upon the condition of the floor and this is a considerable advantage where floor is soft;
8. Where excavations are wide, e.g. open stopes in metal mines, underground engine rooms, pit bottom excavations, etc. roof bolting is useful.
9. Bolts can be used to secure sides in headings and sinking pits.

The disadvantages are that

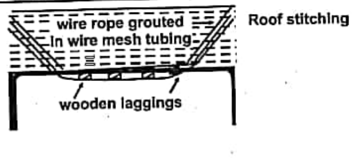
1. It cannot be applied in all cases;
2. It gives no warning of impending failure;
3. Some types of bolts are not recoverable e.g. slot and wedge type.

One Company in U.S.A. Climax Molybdenum Company has successfully used in the late sixties fibre glass rock bolts in varied ground support applications. Two-metre bolts cemented in place within resin have been installed in brows at drawpoints and are also used in supporting under-cut sub-drifts. Since then fibre-glass rock bolts have gained wide popularity in metal mines.

Application Of Floor Bolting

In many roadway supported by conventional methods, having of the floor is of such magnitude that the roadway is completely disrupted. The erection of conventional supports may promote such floor having in some cases because the roof weight is transmitted to the floor through the props or arches and these then pierce the floor, breaking it up and setting it free to move.

Floor bolting, like roof bolting, has as its object the formation of a strong compound beam of clamped floor beds which will be strong enough to resist the lifting forces acting upon it. In some cases, it is found sufficient to concentrate the holes near the middle of the roadway where maximum movement is liable to occur. But the best pattern of floor bolting to prevent heaving of floor can be found by experiments.



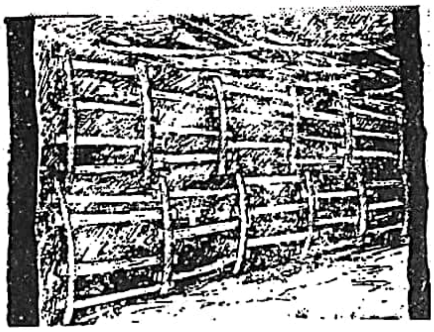


Fig. 9.30 Roof and side stitching with old wire ropes

Roof Stitching

Roof stitching is a method of roof support which has been tried in our mines with encouraging results. In a gallery two slanting holes are drilled by the usual coal drill in the roof. The holes are about 1.4 m to 1.5 m deep. To fit the hole a wire gauze (10 mesh) is made in the form of a tube open at one end. In this tube of wire gauze viscous cement mortar is filled up to three fourth length. The wire gauze is then inserted into the hole in the roof.

Old wire rope 18 mm to 22 mm diameter is cut into suitable length to cover the depth of the two holes and the distance between them. Each end of the wire rope is prevented from untwisting by typing it with binding wire. The wire rope is then inserted into the cement mortar of the wire gauze tube held in the holes. As this is done, some of the cement mortar in the wire gauze is squeezed out of the container and it fills up the hole. The wire rope is held in position by hammering a wedge at the mouth of the hole. The assembly is allowed to set for

24 hours and thereafter the cement mortar becomes sufficiently hard so that the wire rope does not come out of the hole even when the sagging middle portion is pulled by hand. The pairs of holes for roof stitching are drilled at 1 to 1.3 m intervals and the roof stiching serves as a substitute for the conventional support by the set of timber props and bar. The roof gives an appearance as if it has been stitched. Wooden laggings are placed between the roof and the sagging portion of the wire rope. Where the roof is bad sometimes an extra holes (vertical) is drilled into the middle of the gallery and a wire rope inserted into it in the same manner as in the inclined holes. The principle of roof stitching is the same as that for roof bolting, viz. the immediate rocks are held together as a composite stratum of large thickness (Fig. 9.30).

Roof stitching was tried at a number of coal mines as well as in the metal mines at Balaghat and Seetharama mica mine. This method of support was found to be not only effective but also the cheapest. At Ballarpur mines a fault zone, over 60 m length, was successfully supported with the ropes. To make the technique effective it is recommended that *(i)* the face should not advance more than 2.4 m from the last tensioned rope, *(ii)* short ropes should be used in between long ropes for further reinforcement and to prevent bed separation, and *(iii)* ropes should be tensioned while using good wooden sleepers. Where the roof condition deteriorates immediately after blasting, roof stitching cannot be tried.

Plastic perforated sleeves, 35 mm dia. 1.2 m long, with 5 mm dia. perforations have been tried in place of the wire mesh sleeves by CMRS and the results were satisfactory. Preparation of the wire mesh sleeves at the mine takes time and the workers complain of the scratches on the hands due to handling of the wire mesh. With the plastic perforated sleeves tests conducted on coal roof indicated anchorage strength of 10 te and wire mesh sleeves also offered the same anchorage strength.

Resin Capsule Bolting

Roof bolts of the wedge-and-split type an expansion shell type are not suitable in soft rocks due to their poor anchorage in the latter. The wire mesh grouted bolts or roof stitching, though suitable for soft rocks, require about 24 hours for the cement mortar to set. The use of styrene resins instead of cement grout reduces the interval between the moment the bolt is installed and the moment it is ready to carry a load. After a lapse of 15 minutes the bolt can support a 5-te load, and 20 minutes thereafter, a 10 te load. Styrene resins have been used in USSR and some foreign countries. A system of roof bolting which can be adopted in soft rocks and which does not require a long time for the cement mortar to set to make the bolt effective in anchorage, has been developed by the CMRS and is known as **resin capsule bolting.** The resin capsule is a polythene cartridge, 28 to 35 mm dia. and 330 mm long. The components of a resin capsule are resin, a hardener and a filler. The resin is a synthetic polyster resin which has the consistency of a jelly at ambient temperatures. The filler is coarse sand or small sized dolomite chips. The hardner is kept in a separately sealed glass tube which is placed at the centre of the polythene cartridge of resin and filler. The composition of the resin and the hardner are trade secrets. If the glass tube is broken and the hardener allowed to come in contact with the resin and the two are mixed well, the mixture hardens within 30-40 seconds and becomes solid.

In actual application vertical or inclined holes are drilled in the soft rock of roof with the usual coal drill fitted with a slightly larger dia. drill rod for 40 mm dia. hole. The roof bolt to be used is 22 mm dia. m.s. rod, with a length of 230 mm at one end in the form of a scroll (like an auger). The other end of the bolt is threaded to receive a nut. A thin circular washer of 40 mm dia. is provided at 250 mm below the bolt head to prevent falling of resin while mixing inside the hole. The roof bolt is then attached to the coal drill through an adaptor. The resin capsule and the bolt are inserted in the hole, the capsule leading. The bolt is pushed as far as it can go and then the drill switched on. As the bolt starts revolving it is pushed inside the capsule. The rotation of the scrolled end tears open the polythene wrapper and breaks the thin-walled glass tube of the hardener. The bolt is rotated only for 15 seconds. The hardener and the resin, coming in intimate contact with each other, result in a solid compound within 30-40 seconds with the scrolled end firmly embedded in it. The solid compound that is formed grips the side of the hole. The solidifying process inside the hole is indicated by the gradually diminishing speed of the drill. The roof bolt is thus anchored in the hole. A bearing plate and a nut are then fixed to the threaded end outside the hole. The anchorage strength is 8-9 te even with boundage for a length of only 300 mm. Wooden dowels 36 mm dia. made of hard wood, capable of taking more than 10 te tensile load, could also be used as bolt material.

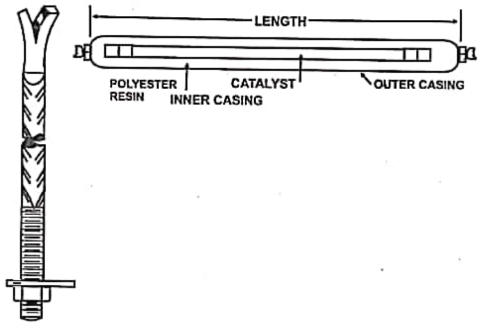


Fig. 9.31 Above - A capsule for resin capsule bolting.
Left - A roof bolt used for resin capsule bolting (N.M.C)

Capsule : It contains separated pre-measured quantity of filled polyestered and resin catalyst. Outer skin of capsule is made of PP, nylone or special paper. Filler is crushed rock-silica, barytes, soap-stone, etc. usually of 250 mesh. Catalyst tube is usually made of glass tube with cork on both sides to prevent mixing of catalyst with resin unless desired.

For a capsule the gel time is the time between the start of the running operation and the onset of the solidificaiton process. This time varies from 2 to 5 minutes, depending upon the type of bolt used. The curve time follows immediately after the gel time and is the time for resin to harden completely and become load bearing. The bolt should not be loaded for the first half an hour.

Under normal conditions the capsule can be stored for 3 months from the date of manufacture.

Cured resin has properties as follows :

Compressive strength - 125 MN/m^2 (12.5 kg/mm^2)

Tensile strength - 50 MN/m^2 (5 kg/mm^2)

The capsule has a length of 310 mm but the diameters are suitable for two hole sizes, 40-43 mm and 31-33 mm. The sp. gr. is 1.9.

For using the resin bolt the procedure is as follows :

1. Air blow or water flush the hole to remove the dust.
2. Check that the bolt is free to rotate in the hole and that the hole is of correct depth, *i.e.* 75 mm less than bolt length.
3. Insert the capsule into hole, push gently home with bolt.
4. Drive bolt home, spinning it all the time with drill. Do not try to rush this stage.
5. Spin bolt for further 30 seconds once it is home. Continue longer if an impact wrench is being used and bolt is rotating very slowly.
6. Removes drill and can nut disturbing bolt as little as possible until 5 minutes after "gel" has occured. Gel time can be detected when the bolt becomes too stiff to rotate slightly between fingers.

Fit plate washer and nut over bolt and tighten up.

The system was tried at Sudamdih colliery for friable coal roof and proved quite effective. It can also be used for side bolting in coal or other corcks if the sides have a tendency to spill off and has been observed to be suitable in holes having water percolation.

Wooden Downel :

It comprises wooden bolt bonded along its whole length with polyester resin. Immediate resistance to strata or coal movement is offered without the necessity of tensioning because the bolt is of wood and can easily be cut; cutting machine can operate freely. They are light, easy to use and positive results are quickly obtained with a minimum of equipment and supervision.

Bolts are manufactured from specially selected timbers which are straight grained, knot and resin free. One end is chamfered to 23 deg. taper to allow easy perching of the capsules, and the other end is left square to permit the attachment to a suitable box section spinning adaptor fixed into the chuck of a drilling machine. The surface of the bolt is left roughened to give good adhesion between the resin and the bolt.

Bamboom Bolting

CMRS has experimented with a system of roof bolting known as bamboo bolting, as a temporary roof support in places such as splits in depillaring districts. A bamboo slightly longer than the depth of the

hole for roof bolt is split longitudinally along the full length. The two pieces are then tied together by a thin wire at 4-5 places after inserting a wooden wedge only partially at one end. Care is taken to see that the knots on the two pieces of the split bamboo are staggered. The wedged end is inserted into the hole and at the other end another wooden wedge is fitted. The bamboo, with wedges at either end, is then hammered into position in the hole. The hammering pushes apart the two splits of the bamboo and breaks the thin wire. The bamboo splits grip the hole sides and the gripping effect is enhanced by the knots of the bamboo.

Sometimes a bearing plate is fitted on to the bamboo before inserting the wedge at the exposed end. A nut cannot be used for obvious reasons.

The bamboo bolting is being tried only as a substitute for the conventional timber supports required for short periods in the splits and during extraction of stooks in depillaring.

Table 3

Common types of roof bolts manufactured by Nanda Manufacturing Company

Type of Bolts	Bolt dia. mm	Recommen-nded hole size mm	Yield load of bolt shank in te	Ult. Load of bolts in te	Weight in Kg/m
1. Slot wedge	24	31-33	10.8	19.0	3.55
	25	31-33	11.7	20.1	3.85
2. Expansion shell Bolts	20	23-33	8.1	13.2	
	20	34-36	13.3	15.5	2.47
	22	40-43	9.1	15.9	2.98
3. Perfo Bolts	20	30-33	13.3	15.5	2.47
	22	39-43	16.1	18.8	2.98
4. Steel Bolt for Resin system	20	30-33	8.1	13.2	2.47
		39-43	...	do	...
5. Wooden Dowel For Resin	32	39-43	–	5	–

For "behaviour of roof during depillaring" and "bumps in coal mines" see the chapter on Pillar Extraction in Board and pillar.

✧ QUESTIONS ✧

1. What are the properties of different types of roofs met within coal mines ? Clarify the terms "good roof" and "bad roof".
2. What are the materials used for support in underground mines? What are the advantages and disadvantages of timber as support material in mines ?
3. How is roof tested in a coal mine ? What are the indications of a "bad" roof ?
4. How are props erected in roadway of varying heights ?
5. Write short notes on :

   (a) early bearing prop (b) link bars (c) subsidence (d) harmonic mining (e) roof stitching.
6. What is the principle of action of a roof bolt ? In which case is a roof bolt is not effective for roof of support ? How is a slot and wedge type bolt fixed in the roof ?
7. Explain the pressure arch theory in respect of bord and pillar workings and for longwall workings.
8. Describe a hydraulic prop and its principle of working. Compare it with a friction prop.
9. In a mine a heavy roof fall has taken place at a junction of a main roadway. Describe how you would clear it ?
10. Describe a suitable prop withdrawer and the manner of withdrawing timber supports in a depillaring district.
11. What is systematic timbering and state ? where it has to be enforced. ?

✧ ✧ ✧

Chapter -10

Stowing Practice

When mineral is extracted from an underground mine the void or goaf is packed with sand or other packing material wherever it is conveniently and cheaply available in sufficient quantities. The process is known as "goaf stowing" or "goaf packing" and often simply as "stowing". The material employed for goaf packing may be stone or shale obtained from bands of stone or shale if they are present in the coal seam under extraction. They may be used for erecting pack wall or for stowing in the goaf, in the crushed form. Other material which is sent down the mine for goaf stowing may be sand, earth, boiler ash, crushed material available from quarry overburden, shale pickings at the screening plant, washery refuse, mill tailings, or slag from blast furnace for iron ore smelting. The material should be free from carbonaceous matter.

The packing material may be packed at site by -

1. dumping it with baskets with the help of human labour as in hand packing.
2. transporting the material with water in pipes and allowing the water to percolate through bamboo matting or similar perforated barricade erected at site. This process is known as hydraulic stowing.
3. throwing it with the help of a high speed belt conveyor as in mechanical stowing.
4. introducing the material in a stream of compressed air as in pneumatic stowing.

The main advantage of stowing is that goaf is packed solidly and the strata over the goaf are supported. The advantages resulting from this are :

1. These are no chances of gas accumulations in the goaf in coal mines.
2. Leakage of ventilating air is a eliminated.
3. Roadways and working places are rendered safe.
4. Better roof control results and effects of surface subsidence are minimised. Very often, there is no subsidence.
5. High extraction percentage is attained. It makes possible the extraction of a thick seam, a seam below fire area, below water bearing strata and below railway, rivers, towns, *etc.* Panel barriers also can be extracted and wastage of coal underground is reduced. Full extraction (nearly 95%) is possible in a seam thicker than 4.5 m only in conjunction with stowing.
6. Depillaring of contiguous seams in any order is rendered possible.
7. No danger of inundation during depillaring.

Sand packing is adopted to stabilise and workings where pillars are small and galleries are high or wide. It is adopted sometimes to form a barrier underground against fire area.

Hand Packing

Hand packing is simple and does not involve capital cost but is limited in scope and depends on human factor for efficiency. It can advantageously be used in places where hydraulic stowing is not much effective, as in flat and shallow seams. Tubs can be conveniently handled any where in flat seams and this facilitates hand packing. It is also useful in seams steeper than 25° when arrangements can be made to tip the tipping tubs and the material packs itself. The process is slow and heavy on manpower; the most efficient operation gives an average of 10 m³ per manshift when materials are close at hand.

HYDRAULIC STOWING

This process is widely used in India in those collieries which are situated within 16 km of rivers giving plentiful supplies of sand, the commonest stowing material in our mines.

The following factors have made stowing possible in many Indian mines :

1. Availability of sand from rivers flowing near the collieries within 16 km.
2. Roof and floor of seams are not affected by water.
3. Seams not being very deep, humidity is not a major problem.
4. Mines are usually at depth exceeding 100 m and the seams are inclined. Hydraulic sand stowing is not successful where the seam is at a low depth from the surface and is flatter than 5°.

From the stage of collecting sand at river and till the sand is packed in the goaf, the following operations are necessary :

1. Gathering of sand at the river bed.
2. Transport of sand from river end to the bunkers on surface at the colliery.
3. Transport of sand hydraulically from the bunkers to the underground stowing site through pipes.
4. Stowing of the sand in the area from where coal has been extracted.

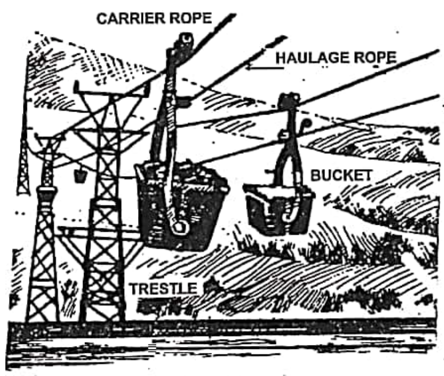


Fig. 10.1 Aerial ropeway

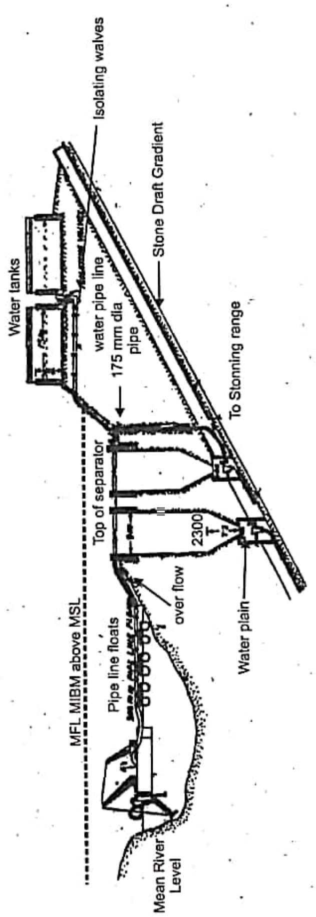


Fig. 10.2 Sand gathering and stowing plat of Chinakuri Colliery

Before introducing sand stowing arrangements at any colliery it is necessary to make bore holes in the river bed to ascertain the depth of sand, to estimate its reserves and to explore the possibility of continuous supplies its reserves and to explore the possibility of continuous supplies to meet the demands. One the of coal extracted needs theoretically nearly 1.3 te of sand in a virgin area. Where old workings have to be stabilised and the pillars to be extracted, the ratio may extend to 2.5 te of sand for every te of coal extracted from old workings. In workings which are developed by bord and pillar method, if the depillaring is to be in conjunction with stowing, nearly 1.8 te of sand per the of coal extracted during depillaring will be essential. In longwall advancing workings, nearly 1.4 te of sand per te of coal raised is required. Wastage or losses account for additional 10-15%.

Sand is brought to the sand storage bunkers on the surface in any one of the following ways :

1. By trucks or wagons.
2. By aerial ropeway (Fig. 10.1)
3. By tipping tubs pulled by haulage if the river bank is near.
4. By pontoon mounted sand pumps discharging into separators. (Fig. 10.2)
5. By sand slushers and scrapers which feed the sand to a river bank bunker and from the bunker the sand is supplied by aerial ropeway to the sand storage bunker at the colliery.
6. By dredgers discharging sand into bunkers.

The sand storage bunker is always situated to the rise side of the underground are to be stowed.

Fig. 10.4 shows the general layout of the stowing arrangement from bunker to the pit-bottom. The bunker is situated on one side of a shaft and a drift inclined at 1 in 3 or 1 in 4 is driven from the mixing chamber (situated directly below bunker) to the shaft carrying the stowing pipe range. Sand from the bunker drops through a chute into a "mixing cone" (in fact, an inverted cone) fitted below the floor of the bunker. Its opening is controlled manually by a rack and pinion arrangement. Sand and water are mixed in the mixing cone in predetermined proportions. The water is supplied by 125 to 175 mm dia. water pipes from the surface reservoirs, situated close to the mixing chamber and provide sufficient head of water for flow through the pipes leading from reservoir to mixing cone. In practice, sand bunkers have a capacity of 2 days' requirement of sand, and water tanks, a day's requirement.

Mixing Chamber

The place where the mixing cone is located is called mixing chamber. The size of the chamber should be sufficient to accommodate the desired number of mixing cones and pipe ranges. The chamber has an access either through an incline from the surface or by a cage operated by a small hoist in the case of a stowing shaft. On the mixing cone there is a screen to prevent pebbles or stones larger than 25 mm size from going into the shaft range with the sand-water mixture. These rejects have to be picked up and collected in the chamber from where they are removed to surface. The chamber should have sufficient lighting and as the work goes on in humid conditions all fittings and cables should be moisture proof.

Signalling or telephone arrangement is provide for communication between the mixing cone operator and the underground stowing supervisor so that supply of sand, water, or both can be stopped or adjusted according to the underground requirements. In the mixing cone, a vertical free fall of sand to a minimum of 600 mm from the bunker chute is believed to avoid heaping up of sand on the screen. The water is admitted to the mixing cone by surrounding it with a circular perforated pipe (garland pipe). It is preferable to line the mixing cone with rubber, e.g. old conveyor belts, to save iron from abrasive sand.

To measure the input of sand and water in the mixing cone with a view to have control on the stowing operations, the operations, the mixing chamber is equipped with ;

1. Water metre : Kent velocity meter is commonly used on some installations ; alternatively a V notch or a venturimeter may be used; and
2. Lea Recorder : The sand input may be calculated in terms of the area of chute opening which can be recorded by a Lea chute opening recorder.

Stowing pipes and their layout

The sand water slurry pipe, from the mixing chamber downwards, may be installed in a borehole, in a steeply dipping stone drift or in a shaft.

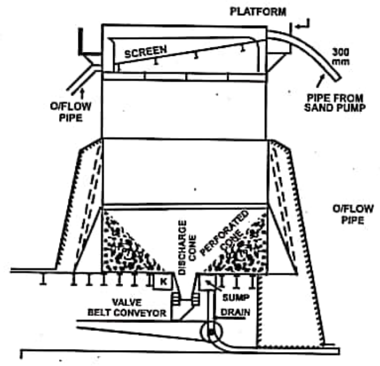


Fig. 10.3 Separator at Bhaladih

The pipes used for sand stowing range area of C.I. mild steel, hot rolled seamless tubes or alkathene. They have flanges for joints (except in boreholes). C.I. pipes are heavy, have a low tensile strength of only 15 kg/mm² and are used for more or less permanent installations in drifts, shafts, main cross-cuts, etc. Their use is not favoured at the face. Sizes in use are 125 mm bore, 3 m long, and with 20-25 mm thick walls. For facility of turning, the marks I, II, III & IV and an arrow for direction of turn, are cast on the pipes during manufacturing. The life of C.I. pipe is observed to be 4 lac te of sand for 20 mm thick and nearly 8 lac te for 25 mm thick walls.

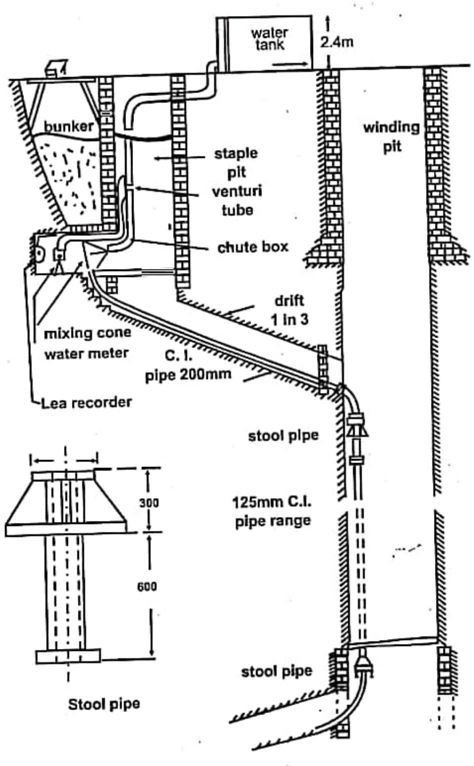


Fig. 10.4 General arrangement of a sand stowing plant

M.S. Pipes are lighter than C.I. pipes and have a tensile strength of 45 kg/mm². M.S. pipes are preferred to C.I. pipes for face pipe range due to their comparative lightness. M.S. pipes are usually with 12 mm thick walls, and of 5.5 to 6 m lengths. Their life is 6 lac te approximately.

Alkathene pipes are much lighter than M.S. pipes and greatly resist wear by sand or siliceous mill tailings. They can be threaded for fitting flanges and pipes lengths can be joined together by heating with a special heating -cut-joining device. Jaduguda mines have replaced M.S. pipe range from the surface to underground stopes by Hasthi pipes (trade name of one make of alkathene pipe) some years ago with encouraging results.

Stowing pipe range consists of three main portions :

1. Shaft range, drift range, or borehole range.
2. Underground roadways range.
3. Face pipe range.

The shaft range is carried in a shaft which is not the main coal winding shaft with a view to avoid interruptions to raisings when pipe ranges have to be changed, repairs to be done, or pipe jams to be cleared. The ropes are badly affected with leakage of sand, and specially when pipes burst, as sand adheres to the greased surfaces of ropes and abrades it. Main intake shafts and main intake roadways are also avoided as water leakage increases humidity.

Roadway pipes are, as far as possible, kept off the main haulage roads as :

1. Leakage of water may wash away tram line packing.
2. Clearing up of pipe jams introduces quantities of sand into roadways, and
3. Dangers to pipe range arise from derailment of tubs.

Pipes are laid out such that the line is dipping all the way and stepped or staircase layout is preferred. Sharp bends are avoided and standard bends of 90°, 45°, 15° are kept at hand and used instead of bent pipes. Smaller deviations of 5° to 10° may be covered by suitably adjusting thickness of packing at the joints. Instead of one large bend, 2 or 3 small bends are preferred, as replacement of a worn-out bend is then easy.

Valves are not used for controlling branches in stowing ranges as they will be easily worn away. Y pipes are used for branching off and the range that is not needed is blanked off. Bifurcators of either "plate type" or "hinge type" may sometimes be used, but they are somewhat uncommon.

Pipe Joints

Fibre or asbestos washers are used between flanges. A ring of lead sheet or of signalling wire, having hessian cloth wrapped around it and soaked in coal tar is a good pakcing between flange joints. Joints without coal tar rot after some time and start leaking. The flanges should be screwed upto the last thread on the pipe end. As the pipe wall thickness is less at the threads, the threaded portion is the first to leak due to wear. Pipe leak is the common cause of interruption in the flowing of sand and of pipe jams. The joints should be occasionally tested to a pressure of 15 to 20 kg/cm^2 of hydrostatic head. This can be easily done by putting in a blind flange at the end of the pipe and filling up the pipe with a known quantity of water as measured by water meter. In very long ranges, the pipes should be tested in different stretches for their ability to withstand pressure.

Where working are on the rise side of a shaft, entry of stowing range by the shaft will not be helpful for stowing sand in the goaf on the rise and stowing ranges can be installed in bore holes drilled on the rise side. Pipes with screwed sockets are lowered down the borehole and are suspended by clamping them to wire ropes anchored at the surface.

Support Of Pipes

In the shaft the pipe is supported on stool pipes placed every 30 to 40 m apart (Fig. 10.5). The stool pipes are supported by double duntons set into the side of the shaft. At intermediate points, the shaft ranges are steadied by single buntons set 8 to 10 m apart in between the double buntons.

At the bottom of the range in the shaft the change of director of the pipe range is effected by a stool bend. It is usual to fix a stool pipe on a double bunton at the bottom of a shaft range and to attach this to a stool bend. Along the roadways the range rests on the floor or on sleepers or on brick-work wherever necessary for smooth gradient. It is essential that the range be fixed to the ground securely.

Wear in Pipes

In a shaft range there is more wear on the upper portion of the range than at the lower. The upper portion of shaft range is generally changed after transport of 2,00,000 to 4,00,000 te of sand (C.I. pipe, 25 mm thick walls). Lower portions are known to pass even 8 lac te of

sand. To economise and to utilise the shaft range to the full extent, it is advisable not to change the whole length of shaft range at a time but in parts, say, length between two stool pipes at a time. The safe wall thickness in C.I. pipes is 6 mm with a factor of safety of 3.5 for a depth of 400 m. It is not necessary to rotate the pipes of shaft range as the wear in uniform throughout the bore of individual pipe length. Where the pipes are in inclined position, e.g. in the drift and in the underground roadways, maximum pipe wear is in the bottom portion covering an arc of 90^0. Hence rotation of pipes should be completed in four stages for uniform wear of the pipe walls. AT one time 10 to 15 pipes of a range could be turned. After turning the joints should be checked, because, those with taper, become slack.

Hydraulic Profile And H : L Ratio

It is essential that the pipe layout in a hydraulic stowing installation should conform to a correct hydraulic profile. Incorrect profile will cause cavitation and then the full available head cannot be pressured to use; high local velocities will cause considerable wear on the pipes and the entrapped air will set up pulsations in the system.

The length (L) of the pipe range through which sand water slurry can be transported depends mainly upon the vertical head or height between the mixing chamber and the point at which slurry is discharged (H). The ratio H/L (height/length) available in practice for a stowing range indicates its efficiency. Generally this ratio is 1/7 for reasonably good slowing rate.

Underground Stowing Arrangements and Operations

Before a goaf is packed a boxing (a barricade of bamboo matting) should be constructed as near the face as possible, leaving place for conveyor path, coal cutting machine and roof supports. On a longwall face the width between previous boxing of sand pack and new boxing under construction is generally 4 to 5 m. With smaller widths of packs, cost of boxing becomes high and face work gets disrupted frequently. Materials used for boxings of narrow packs 1.5 m to 1.8 m in width should be stronger than those for packs of larger widths, say 3.5 m to 4.5 m, as the pressure developed on boxing in narrow packs due to sand is more. Bamboo matting or hessian cloth is generally used, and in some installations coir matting has been tried. The latter, though costly, can be used 3 to 4 times and has thus an economic advantage over the bamboo matting has been tried.

The latter, though costly, can be used 3 to 4 times and has thus an economic advantage over the bamboo matting or hessian cloth which can be used only once. Wooden props or telescopic rails are erected along the new boxing line and the hessian cloth or bamboo mating is fixed to it by wire nails or strings from the floor to the roof. In some cases props are placed slightly inclined. For this, two chalk lines are marked on the roof by stretching strings to indicate position of the top and bottom ends of props. Ideal inclination would be the angle of repose of sand, but due to limited space it is not practicable and the prop has in inclination of 10^0 with the vertical, the top end leaning towards goaf. In such cases recovery of props and coir mattings is easy. If the boxing is made vertical, the sand buries the coir matting which is, therefore, difficult to recover when props or rails are withdrawn. The dip side boxing is made stronger with double bamboo matting, and sometimes, with wooden planks or coir matting for a simple reason; the dip level may be drainage level or it may carry gate belt or tracks for tubs and bursting of the packing may foul the level road. Chocks are preferred to props on such dip side packing. When the new boxing is complete, the old boxing is dismantled and props, coir matting, planks, *etc.*, are recovered from it, unless the old boxing is requires to protect equipment of gate roads.

Telephone is extended to a convenient place near the new boxing when stowing of goaf is to be undertaken.

At the commencement of stowing the stowing range is extended so as to keep the end of stowing pipe nearly 7.5 m away from the dip side boxing. The velocity of sand : water mixture would depend upon the pressure head available. A nozzle is attached to the pipe end to increase the throw in special cases. Without nozzle the throw in one case was 3 m but with the help of a nozzle it was increased to nearly 7 m. The nozzle is of mild steel and tapers from 125 mm dia. to 85 mm dia. within 1.2 m length.

To stope the stowing, the flow of sand into the mixing chamber at the surface should be stopped first and then, after 3-4 minutes, the flow of water should be stopped with the discharge through the nozzle is only of water. This avoids pipe jam.

Concentration of sand and water

In most of our installations for depths of 200 to 300 m the average ratio of sand : water may be 1 : 1.5 by volume though it varies with individual installations.

Sand		water (by vole)		sand		water
1	:	1.5	...	1 te	:	935 litres
1	:	2	...	1 te	:	1100 litres

For deeper mines the sand : water ratio has been more favourable, nearly 1 : 1.3 by volume.

Rate of stowing

For a given stowing pipe range there is a maximum quantity of the mixture which can flow; if more quantity is fed, it will overflow the mixing funnel. If less quantity than this maximum is fed, the vertical range will not run full bore; as the mixture flows down the pipe, a partial vacuum will be created and the air will be drawn in, leading to a pulsatory flow and consequent jamming of the pipes. High velocities will be created in the upper section of the vertical range causing excessive wear on pipes. Thus it is desirable that just the maximum quantity of the mixture should flow in the range, ensuring a full bore flow in the vertical range. Rate of stowing varies from installation to installation. A rate of 80 to 100 te of stowing (sand only) per hour is common in many miners at 200 to 300 m depth with nearly 1100 litres of water per te of sand.

Discharge of sand is given by the formula :

$$Qs = \frac{Qm \times S}{S+W}$$

Where

Qs = sand (m^3) discharged/hr

Qm = sand & water mixture (m^3)/hr.

S and W are proportion of sand water respectively (by vol.)

CMRS, during its experimental work at Ballarpur colliery (depth of coal seam 45 m from surface) has arrived at the conclusion that a high rate of stowing at a water ; sand ratio of less than unity (nearly 0.9 water : 1 sand) by volume can be obtained even with 125 mm dia. pipe range. During the experiments carried out at Pure Jambad colliery CMRS claims to have achieved a water : sand ratio of 0.9 : 1 in pipe range of 125 mm and 150 mm bore and the H : L ratio was 1 : 11.4.

Pipe Jams

Pipe bursts take place when pipes are worn and they are not able to withstand high pressures. If extra pressure is created due to water ram, that may burst a worn-out pipe. When the valves of water pipe are not closed well after stowing it is possible for an extra pressure to be created due to water head.

In the mixing chamber heaping of sand on the screen, overflow of water at the cone and froth on top of the mixing cone are some of the indications of pipe jam. If it is of minor nature, sand is stopped and clear water flushed; the jam may clear. Hammering the pipe at suspected places of jam may help in some cases. If the methods do not help, pipes are opened out at suspected places and the sand deposition scraped.

Pipe jams are completely avoided if (i) air removal devices are incorporated in the installation and the full-bore flow is obtained, (ii) the pipe layout is in correct hydraulic profile, (iii) the size of the largest particle of the solid does not exceed 1/3rd D (preferabely 1/4 to 1/5th D), where D is dia. of the pipe, and (iv) the average velocity of the mixture is above the critical velocity of deposition.

Installation of booster pumps for increasing the pressure of sand-water slurry to stow on the rise is not very common. As a thumb rule, for every 10 m length of a horizontal pipe range there is a loss of 1 m head and the velocity of slurry has to be kept at a minimum of 2 m/sec to avoid sand deposition during flow.

Depillaring with stowing (Bord & pillar)

As in the case of depillaring with caving, pillars which have been "robbed" should be stabilised in the depillaring district by sand stowing. The pillar to be extracted is split into 2 or 4 stocks depending upon its size. Coal cutting machines may be used for the purpose.

The operations are conducted from dip to the rise in a panel. In each stood splits are driven either with coal cutting machine or manually and the size of split is normally 3 to 4.8 m. Between two adjacent splits a rib of coal, 1 to 1.5 m wide, is kept solid. At one time only one split (and sometimes two) is started in one stook and when that split is about to be completed, the second split is started in the same stook with a view to keep coal production nearly nonfluctuating. The split which is completed is then stowed with sand.

When the second split of the same stook is completed and the place is ready for stowing, coal from the partition-rib is extracted to the maximum possible extent before stowing. The operations are so conducted that not more than 80 m² of roof area is exposed in any working place at a time and not more than two such areas are left unstowed at any time. Fig. 10.6 shows the general arrangement. It may be noted that sand stowing pipes are taken to the depillaring area from the main pipe by branches at two or three places for facility of stowing. The figure shows a diagonal line of face, but a line of face parallel to the dip-rise may also be followed. A line of face parallel to the dip-rise is common and this helps stowing operations which are commenced at the dip end of the boxing. Fig. 10.7 shows a line of face parallel to the strike.

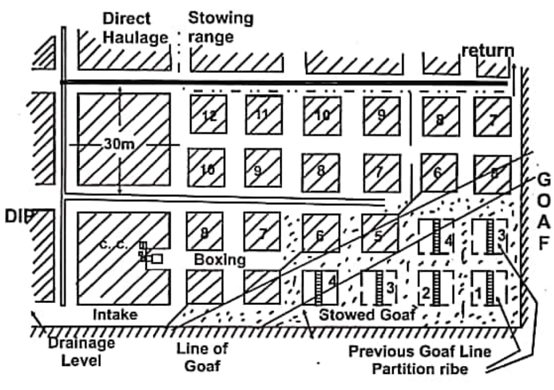


Fig. 10.6 Pillar extraction with hydraulic stowing

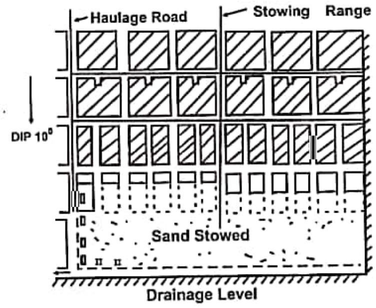


Fig. 10.7 Line of face parallel to strike

PNEUMATIC & MECHANICAL STOWING

These methods of stowing are not practised in India. They are definitely costlier than hydraulic stowing, but will have to be adopted at some collieries in the near future for nearly complete extraction.

Pneumatic Stowing

The stowing material may consist of washery refuse, boiler ashes, surplus pit rubbish, picking from the screening plant, or shale bands from the coal seam. Sand alone is not used as it is heavy and abrasive. A mixture of sand and washery dirt may be suitable. Debris containing much clay material is also unsuitable because it clogs the pipes. very dry material should be dampened to allay dust. Damp material is less abrasive and gives tight packing in the goaf.

The crushed material passes from the crusher through 65 mm to 75 mm aperture wire screen to a storage bunker. Tubs are loaded at the storage bunker and then lowered underground when coal-winding is slack. There should be sufficient storage room underground to stock these tubs and adequate haulage facility so that during stowing shift

the stowing operations are continuous and materials are supplied to the stowing plant without interruption. Underground, the loaded tubs are taken to a tippler which tipples the material on the feeding hopper of the stowing machine which is situated at a convenient point near the goaf to be packed.

The compressed air required for the stowing operations is supplied from the surface through pipes of 200 mm to 300 mm dia. Smaller size pipes are rarely used. The surface air compressor is multistage, has a large capacity of 55 to 85 m^3 of free air/min., consuming 400 to 600 H.P., and compresses the air to nearly 6 kgf/cm^2. The stowing pipes are of high carbon steel 10 mm thick, in lengths of 3 m. Roadway pipes may be with flange joints but pipes at the stowing range have quick release couplings (victaulic or hamecher type) with rubber seals. The compressed air ranges in the shaft and galleries are fitted with automatic water drainage devices and pressure gauges at intervals.

At the goaf the material used for boxing is corrugated sheets, hessain cloth, wooden planks, wire netting or paper reinforced with very thin wire netting.

Stowing Machines

The machines are of two types :

(a) Chamber machines, *e.g.*

(1) Torkret single chamber.

(2) Double chamber or automatic.

(3) Three chamber or vollautomat.

(b) Compartment or pocket wheel machines :

(1) Beien (2) Brieden (3) Blast stower.

Fig. 10.8 illustrates the Markham blastower. It consists of a rotating paddle wheel or drum driven at 40 to 50 r.p.m. by an electric motor of 15 to 20 H.P. The drum is divided into chambers which are filled with debris from the feed hopper above. As the drum rotates, each chamber in turn reaches a position where it forms a continuation of the compressed air inlet on one side and the delivery outlet on the other side. The compressed air thus flows through the chamber in the rotating drum, carrying the debris with it through the chamber in the rotating drum, carrying the debris with it through the stowing pipes to the face. The machine has a capacity of 1.5 m³ min with an air consumption of 70 to 85 m³/min. Allowing for stoppages, stowing rate may average to 42 m³/hour. Pressure gauges are provided to indicate air pressure at the inlet to the machine and the back pressure at the outlet due to resistance set up in the stowing pipes. The air pressure at the machine is regulated down to 1.8 to 2.8 kgf/cm².

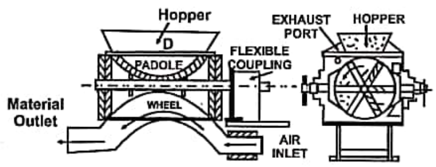


Fig. 10.8 Markham blastower

At the discharge end of the stowing range a detachable deflecting nozzle is fitted to enable the operator to direct stowing material as required. The air velocity in the pipes is to about 60 m/sec. The average velocity of discharge of stowing material is about 15 m/sec. The packing in the goaf is nearly 70% as compact as the solid coal. Convergence of the order of 20% takes place in course of time. The compactness of the packing depends on the type of material, its size, density, moisture content, length of stowing pipe and the velocity of discharge.

In this method the face becomes dusty due to air-borne dust of stowing operations and there is much noise by the machines. Danger of electrostatic sparks due to high discharge of compressed air also exists. Proper earthing is therefore essential. The work can proceed during coal winning and loading shifts also.

The method is adopted in Germany, Britain and other foreign countries.

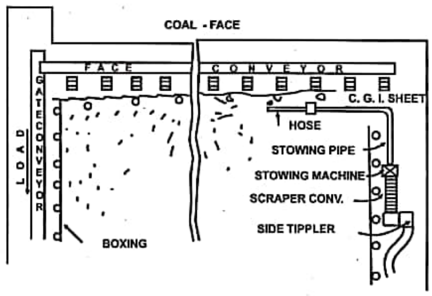


Fig. 10.9 Face arrangement in pneumatic stowing

Mechanical Stowing

In this system a high speed belt is used to propel the stowing material into the goaf to be packed. The stowing material is transported to the thrower belt in the same manner as for pneumatic stowing. The material has to be deflected on the thrower belt by deflection plates known as scraper ploughs, which drops the material into a hopper above the high speed stowing belt. The thrower belt is mounted on a travelling carriage and a 660 mm wide belt is considered adequate. The driving drum rotates at 1000 r.p.m. giving the belt a velocity of 600 m/min (nearly 36 kmph). Driving motor is 25 to 40 H.P. and stowing of 50 to 60 te/hour of stowing material is possible. Good results are achieved if the stowing material is delivered to the thrower belt at a high speed and in the direction of belt travel. Barricade of wire netting is required is in other methods of stowing.

Of all the methods of stowing, it is the cheapest (less capital cost) and the number of manshifts required for roof control per 100 te coal produced is the least. Energy costs are a fraction of the costs for pneumatic stowing (consumption 0.3 to 0.4 kWh/m^3 material). It however does not have the adaptability of pneumatic stowing. Its disadvantage is that a prop-free front cannot be planned; the installation of a belt solely for throwing requires careful consideration from the stand-point of economy and roof control; it is only applicable in thin flat seams or slices of 1.3 to 2.3 m thickness; the roof convergence is more compared to that in pneumatic stowing; coal getting and stowing cannot take place at the same time; installation of a conveyor in the tail gate for continuously feeding the machine is unavoidable. At the face, the operation creates a lot of noise and airborne dust and the risk of electrostatic sparks exists in this machine as the material is discharged at a high speed.

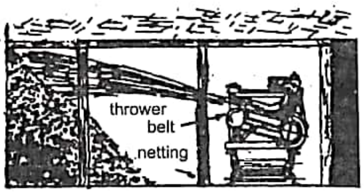


Fig. 10.10 Mechanical stowing

The method is popular in Germany and is used on a limited scale in Britain.

The pneumatic stowing, compared to this method, needs high initial cost for compressors, pipe lines for compressed air, and excessive power and air consumption.

✧ QUESTIONS ✧

1. What are the advantages of stowing ? Why is stowing not followed on a large scale in our mines ?
2. Give the layout of stowing pipes from surface to the goaf. What steps should be taken to get maximum life from the pipes ?
3. Describe the process of hydraulically stowing a goaf on a longwall face 100 m long.
4. Show by a sketch how pillars are extracted in conjunction with hydraulic stowing if the mine is developed by bord and pillar method.
5. Write notes on :

   (a) *Pneumatic stowing*

   (b) *Pipe jam in hydraulic stowing*

   (c) *H/L ratio.*
6. State the advantages and disadvantages of mechanical stowing.

✧ ✧ ✧

Chapter -11

Bord & Pillar Method of Working Coal Development

Methods of winning coal in underground mines are classified into two main categories :

1. *Boar and pillar (also known as pillar and stall)* and its modifications.
2. *Longwall*
   (i) Advancing, (ii) Retreating,

The following methods which are derivatives of the above principle systems have since acquired distinctive nomenclature :

(i) Room and pillar, (ii) Level mining,
(iii) Horizon mining (iv) Slicing,
(v) Sublevel caving.

As stated earlier in the chapter or Drivage of Roadways, development of mine by the method of working known as bord and pillar consists of driving a series of narrow roads, separated by blocks of solid coal, parallel to one another, and connecting them by another set of narrow parallel roadways driven nearly at right angles to the first set. The stage of formation of a network of roadways is known as *development or first working*. The coal pillars formed are extracted after the development of the mine leasehold and this later stage of extracting coal from the pillars is known as *depillaring*. Fig. 7.1 shows the general scheme of development by bord and pillar in an incline mine and Fig. 11.1, in a pit mine.

The bord and pillar method is adopted for working.

1. a seam thicker than 1.5 m,
2. a seam free from stone or dirt bands. Stone or dirt bands, if present in a seam, can be easily disposed of for strip packing in longwall advancing method of mining.
3. seam at moderate depth,
4. seams which are not very gassy,
5. seams with strong roof and floor which can stand for a long period after development stage is over,
6. coal of adequate crushing strength.

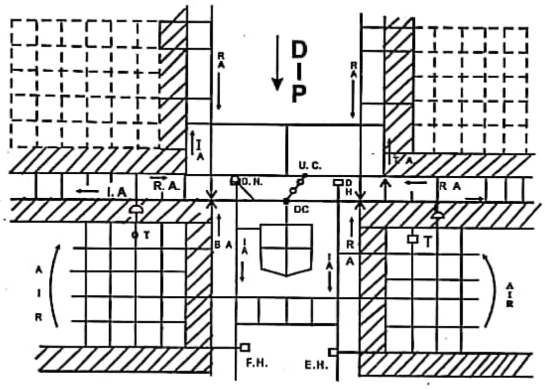


Fig. 11.1 Bord and pillar layout in panels in a pit mine. Firm lines in the panels indicate galleries under development and the dotted lines indicates proposed development.
T-tugger haulage; E. H. - endless haulage; D.H. - direct haulage; I.A.-Intake airway;
R.A. return airway.

In our country most of our coal seams satisfy the above conditions and therefore bord and pillar method of mining has been commonly adopted in a large number of our mines. It possesses the following advantages.

1. Roads and airways are in solid coal and their maintenance cost is low throughout the life of the mine.
2. Coal output is obtained while roadways are being made during the development stage, and naturally during the depillaring stage, thus providing a continuous flow of coal after the seam is touched.
3. Unlike in longwall mining no unproductive work of dining, strip packing, etc. is involved.
4. The development stage reveals the geological disturbances enabling the management to plan accordingly.
5. The working team is usually small at working faces. This helps in simpler methods of calculation of work performance, smoother and more co-ordinated work. The effect of absenteeism is not significant.
6. Surface features like railways, important buildings, rivers, etc. which should not be disturbed by underground methods of mining can be well supported during the development stage by the solid pillars of coal and later by only partially extracting the pillars if stowing is not practicable.

The **disadvantages** of the method are :

1. Ventilation is sluggish, as compared to longwall method, at the working places.
2. The extraction losses are generally higher than in other methods of mining.
3. Work is carried on at a number of working places creating problems of supervision.
4. At great depths, the working by this method becomes difficult as effects of roof pressure are not easily controllable; heaving of floor and creeping of roof may result in loss of roadways.
5. The effects of subsidence and interaction on other seams are not even and not easily predictable or controllable.

If coal from the full seam thickness is extracted the percentage extraction during development is neary 20% in deep mines and 35% in shallow mines. The percentage extraction is calculated as follows :

In Fig. 11.2 a, b, c, d is a square pillar formed during development. Consider that pillar size is 30 m centre to centre and galleries are 4 m wide. In the excavation formed by drivage of galleries the excavation enclosed by lines A,B,C,D, may be considered to belong to the pillar a,b,c,d, and the remaining excavation may be considered to belong to adjacent pillars. The original block of coal A,B,C,D, has therefore resulted into a pillar of coal a,b,c,d after driage of galleries.

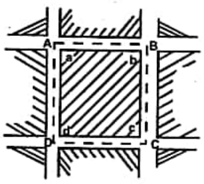


Fig. 11.2

Extration of coal = Area ABCD - area abcd

$$\% \text{ extraction} = \frac{\text{Area ABCD} - \text{area abcd}}{\text{Area ABCD}} \times 100$$

$$\frac{30 - 26 \times 26}{30 \times 30} \times 100 = 25$$

Panels

In a coal seam which is liable to spontaneous heating development is carried out in panels. A panel is a district separated from other districts by a barrier which may be of solid coal or of brick stoppings. Between two panels only essential galleries required for passage of men, ventilation, drainage or stowing are driven. In case of an emergency arising from spontaneous heating, out-break of fire in the panel, heavy gas emission, crush of pillars, etc. the panel can be sealed off by brick stoppings and isolated from other workings. The coal barrier between adjacent panels is usually of the same thickness as the thickness of pillar. A dyke or a fault plane may sometimes serve the purpose of a panel barrier. The number of pillars in a panel is such that during the depillaring stage the coal from the pillars can be extracted to the maximum percentage within the incubation period which varies from seam to seam and is usually between 3 and 10 months. The number of pillars varies from 12 to 30.

A panel is normally longer along the strike and shorter along the dip-rise direction. Long panels are unavoidable where the development is by conveyors or by locomotives and such long panel is sub-divided into small panels during depillaring by construction of isolation stoppings. Fig. 11.3 shows the general layout of workings in a panel where the panel is approached by an incline.

Where depillaring operations will be in conjunction with stowing, formation of panels is not necessary but may sometimes the demanded by the DGMS.

Opening Out a District

The drivage of roads has already been described. In a new mine a set of headings is driven in the dip-rise and level directions. This proves the thickness and gradient of the seam and gives an idea of the rate of gas emission, watery nature, roof condition and the geological disturbances such as faults, dykes, etc. Districts are opened from such set of headings. The manner of opening out districts depends upon :

1. Mode of entry into the seam, whether by an incline or by a pit.
2. Type of transport system used for the districts.
3. Gradient of the seams.
4. Nature of coal, whether liable to spontaneous heating or not.

In an inclined mine using rope haulage the direct rope haulage is the main transport and it is along the true dip except in steep seems.

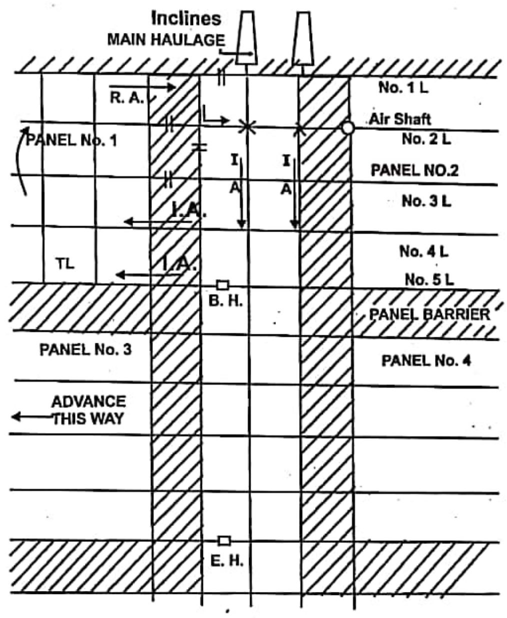
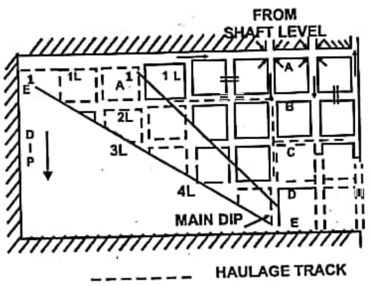


Fig. 11.3 General layout of workings in panels in an incline mine, with two intake airways. TL-tailrope haulage

From the main haulage road districts are opened the strike by use of endless haulage or by conveyor as shown in Fig. 11.3.

Fig. 11.4 (a) and 11.4 (b) show two methods of opening district on the strike from a main dip in a gassy mine (degree II and higher). It should be noted that when opening a district, transport and ventilation considerations usually conflict. It is a good practice to have the least number of air crossings in the roadways leading to the district and it is also desirable to avoid doors or brattices on the haulage roads and to arrange for ascensional ventilation. These considerations have partly to the compromised for transport arrangements.

In Fig. 11.4 (a) when main dip advances to A, level in the direction AA^1 can be opened. By the time the main dip advances to C (2 pillars) the level from A advances to A^1 (3 pillars or more) as progress in levels is easy. The lie of face may therefore be CA^1 and in course of time, EE^1. Such line of face which is nearly diagonal to the dip has the advantage of offering a number of faces and therefore increased output, soon after the main dip reaches the point A, the entrance to the district, and is not uncommon in many mines. The main disadvantage of such diagonal line of face is transport. If tubs are used, at E no hand tramming is necessary but to bring tubs of E1 to main dip hand tramming is involved upto 6 pillars - quite a long distance.



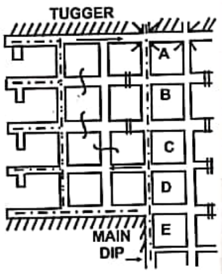
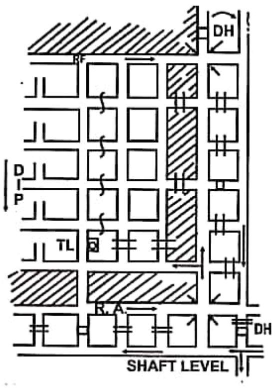


Fig. 11.4 (a) · Fig. 11.4 (b)

Opening a district from the main dip. Arrows show airways.

An endless haulage in the level AE_1, or in the level off B, has only limited utility for the 2 risemost levels and the lower levels still involve hand tramming or carrying loads uphill to take advantage of endless haulage. A small direct haulage (tugger) along dip rise in No. 1 or 2 level is not much utilised unless it is situated inbye, and if situated inbye, it cannot serve the lower levels.

Fig. 11.5 (a) and 11.5 (b) show 2 methods of opening of district on the rise side of shaft level. Fig. 11.5 (a) indicates a line of face diagonal to the dip or rise and the transport of tubs by a tail rope haulage TL. The return pulley can be extended as the main rise advances. In Fig. 11.5 (b) which is a better method of development, the main rise and companion rise are advanced upto panel barrier to the rise using tail rope haulage for transport as shown in the figure. It is later replaced by a direct haulage installed at the rise-most level. The workings are then opened out along the strike so that the line of face is nearly parallel to the dip-rise direction.



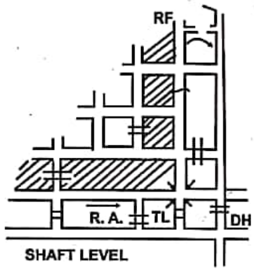


Fig. 11.5(b) Fig. 11.5 (a)

Opening a district on the rise side from the shaft level. Arrows show airways

In Fig. 11.3 which shows development in an incline mine with two intake airways at one time, No. 1 panel may be opened out from the dip headings so that 3 dip headings and the 5 strike headings of No. 1 pane advance at a time. If sufficient machinery's available No. 2 also may be opened out. In such case the endless haulages of No. 1 panel and and No. 2 panel are staggered. The capacity of the main direct haulage decides the number of panels to be opened at a time. The endless haulage may extend upto the panel barrier or a long distance, say 600 m along the strike. As each panel extending over such a long distance will contain a large number of pillars, small panels have to be formed by isolation stoppings prior to depillaring.

If conveyors have to be employed in the district a belt conveyor is installed in the middle of the district, i.e. in 3rd level (Fig. 11.6). A belt conveyor can work for 1000 m along the strike, but it is unlikely that the strike of the seam will be a straight line over such a long distance. Belt conveyors of smaller lenghts are therefore employed in underground mines. It is fed by scraper chain conveyors and the layout of of the district is then as shown in the figure. As a scraper chain conveyor of 15 to 20 HP transports loads satisfactorily upto 100 m along level or dip gradients and upto about 70 m on rise gradients, drivage of 5 headings in a district is advantageous for conveyor within shovable distance at the face is not always possible as coal cutting machines have to flit from one side of the conveyor to the other at the face. In the dip or rise working faces shown in the figure installation of chain conveyors would be an extravagance. It is a common practice to load coal in baskets on a chain conveyor and the lead normally does not exceed 30 m. Extension of the chain conveyors is carried out by additions of pans and chains whenever necessary. The discharge point of chain conveyors on belt conveyors should be staggered to avoid congestion at the transfer point. The belt conveyor of the district is called 'Gate belt'. It may discharge coal into tubs drawn out by main direct haulage or it may feed a belt conveyor installed in the main dip upto the surface (known as trunk conveyor). The output per loader with basket loading on to a chain conveyor is 3 te to 5 te.

Where the entrance to the mine is by pit the main haulage system is in the shaft levels which extend on either side of the pit. The main haulage may be endless rope or locomotive. Rise and dip districts are opened by first driving a set of 3 or 4 headings off the shaft level in the rise or dip direction respectively.

In deg. II and higher gassy mines to reduce the number of air crossings it is a good practice to drive 4 dip headings, out of which the middle 2 serve as intake roads and the other 2 as return roads. An intake and its adjacent return serve half of the panel and the other intake and return serve another half of the panel. (Fig. 11.7).

A tail rope haulage installed in the shaft level serves the rise headings and a direct rope haulage, also installed in the shaft level, serves the dip headings. Such headings are extended as long as the capacity of the haulage permits. Level galleries on either side are opened off the headings as the latter advance upto the panel barrier as shown in the figure. It shows such arrangement where a set of 4 headings is

driven to the dip or to the rise from the shaft level. The direct haulage A is used for development of No. 1 panel and if it has adequate capacity it is utilised for development of No. 2 panel also. Normally it may not be able to deal with output of a panel to the dip of No. 2 panel because of the length. Therefore the haulage is shifted to the position B for development of No. 3 panel and subsequently, No. 4 panel. If depillaring permission to extract No. 1 and No. 2 panels is possible after development of these panels, the haulage at A is used to depillar the 2 panels and another haulage is installed at B for development of No. 3 and No. 4 so as to keep them ready for depillaring by the time No. 1 and 2 panels are depillared.

The development of rise panels followes similar arrangement of haulage (tail rope type) with slight modifications to suit individual conditions.

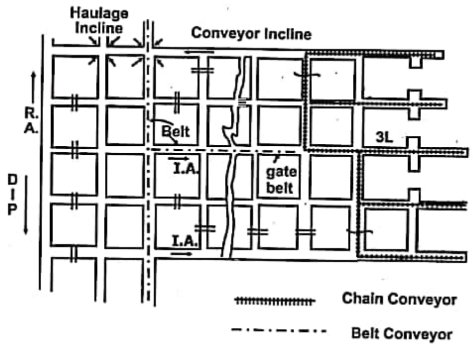


Fig. 11.6 Development by conveyors.

Development by cross cuts

Development of the area to the dip side of the shaft level or prominent haulage level by cross cuts is a common practice in many mines as one direct haulage can work in the main dip as well as in the cross cut. The seam should be inclined and the gradient in the cross cut should not be less than 1 in 12, the limiting gradient for direct haulage. On milder gradients the empty tubs, when lowered, are not able to pull the rope of the haulage engine and trammers have to drag it, often on their shoulders. This seriously affects the output.

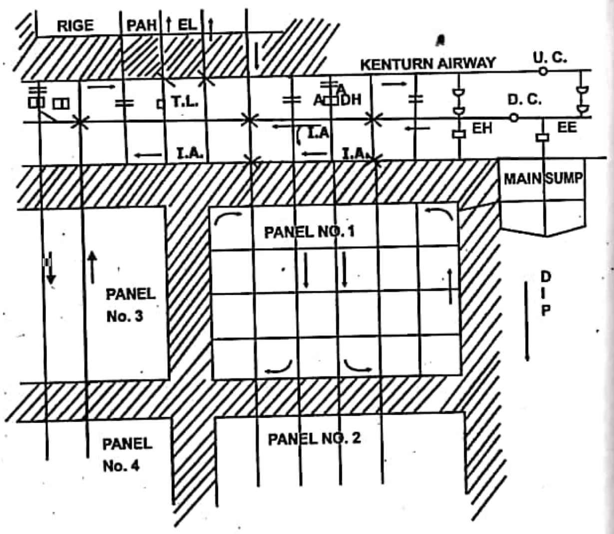


Fig. 11.7 Layout in panels in a pit mine with two intake airways. Thick lines indicate main headings and arrows indicate air travel

Cross cuts not only save the number of haulage engines but also help to develop the area on the strike when the main dip is advancing and for this reason a cross cut should make a sufficiently large angle with the main dip, provide the gradient of the former is suitable for operation of the direct haulage. In a seam of mild gradient cross cut is not an advantage. Cross cuts to the rise side are uncommon for haulage purposes. In Fig. 11.8 one direct haulage works along the main dip and the same haulage operates for the crosscut. A panel is opened from the main dip and another from the cross cut so that development on the strike in the panel off the crossout takes place at the same time when the main dip and cross cuts also progress and prove a large area in advance. Two cross cuts, one going East and the other going West, branching off from the main dip at more less the same point are to be avoided as they weaken the pillar formed at the branch-off point.

In the district opened on the strike from the cross cut the upper levels have to advance a good distance before a tugger or tail rope haulage can be installed in a dip-rise gallery. It is, therefore, common practice to have only a few pillars (usually 3 or 4) in the dip-rise direction in a crosscut district.

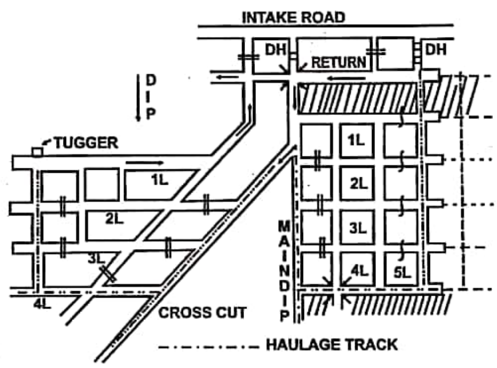


Fig. 11.8 Development by crosscut

Development in steep seams

The methods of development by bord and pillar, so far stated, are suitable for working seams upto a gradient of 1 in 4. Special arrangements have to be made in working seams steeper than 1 in 4.

The special arrangements are as follows :

1. The main haulage or conveyor should be, not along true dip, but along apparent dip to reduce the gradient of the road (Fig. 11.9)
2. The coal pillars should have smaller dimension along the dip and larger dimension along the strike.
3. The level galleries should be joined, not along true dip but along apparent dip. The pillars, therefore, become rhombus shaped. The floor area of each pillar should be the same as the area required of a pillar at that particular depth. The acute angle of the rhombus pillar should not be small, otherwise the corners may crush.

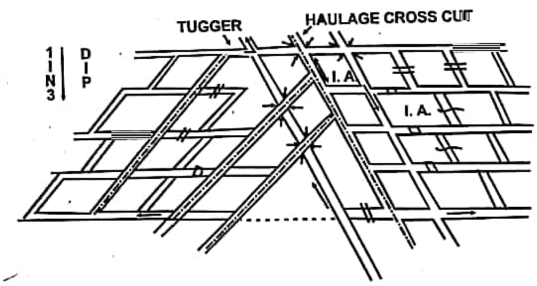


Fig. 11.9 Development of a steep seam.

4. Except the haulage dip the cross-connections between levels should be staggered. Otherwise a sleeper or other material rolling down a cross-connection would roll down to the lowest level without any hindrance.

5. From the main haulage road, tram lines should be taken to each level.

6. Steps should be provided not only in dip galleries, but also in level galleries as the floor of the seam is steeply inclined and not suitable for walking.

7. In the dip or rise galleries where the loaders have to carry coal in baskets to the tramming level, in addition to steps, hand bars should be provided.

8. Face pumps should be trolley mounted.

9. If chain conveyors are used at the face, the conveyors in the level galleries should be on the rise side. The skid mounted coal cutters and mechanical loaders have a tendency to skid to the dip side during filtting in the "level" gallery. Other equipment like drill transformers, gate end boxes, etc., should also be kept on the rise side in the level for the same reason.

10. The coal tubs or mine cars should be of special design to prevent spillage when hauled along the haulage plane.

11. When layout a haulage track in the level gallery, the floor should be blasted as it is inclined and the blasted rock may be utilised for packing of the tram line. Movement of the track towards the dip side of the level gallery is sometimes prevented by placing long timber props as structs between the rail and the pillar.

12. Use of skid mounted coal cutting machines should be avoided and solid blasting with suitable explosives may be adopted.

Output Available From One District

It is assumed that coal cutting machines are used. The average number of faces in a district with a number (n) of headings is (2 n-1). With 3 headings, 5 faces and with 5 headings, 9 faces are normally available for working. Out of these, one or two faces are generally stopped for roof or side support. Each face receives one cut per shift under normal working conditions with skid mounted coal cutters. If the haulage track or chain conveyors are close to the face (say, within 6 m), it is possible to get 4 cuts in 4 shifts at the same face with well planned arrangements. With a 1.8 m long jib of coal cutter the average depth of cut comes to 1.5 m under good supervision. In a gallery of 4.2 m width and 2.1 m height the output per cut is 17 te under ideal conditions, but rarely more than 14 te are available because of the floor coal left during operations of coal cutter. The output in a day of 3 working shifts may therefore be calculated as follows :

Main headings

3 headings provide 5 faces. Assuming 4 faces available for cutting, output from them will be :

4 faces × 3 cuts × 14 te per cut = 168 per day. From 3 main headings output of 150 te per day is reasonable.

District Headings

5 headings provide 9 faces. Assuming 8 faces available for cutting, output from them will be :

8 faces × 3 cuts × 14 te = 336 te per day. From 5 headings output of 300 te per day is reasonable.

A reasonable output per day from a district with 3 main headings and 5 district headings would therefore be 150 + 300 = 450 te. In practice it has been found that this output is less by 25% and in some cases even by 50% in mines with rope haulages and coal cutters. The main reasons are usually as follows and the reasons suggest the remedies.

Mains Reasons For Low Output

(1) At the face

(a) Face is not properly prepared for the coal cutting machine. Floor coal is not removed and the effective height is less. Similarly width of galleries is narrow at the face than at the outbye.

(b) The depth and width of the cut are less due to bad manner of sumping in and cutting. This is particularly noticeable where machine drivers are paid an incentive only on the number of cuts.

(c) It is a good practice to start coal cutting from one end of the panel and proceed from face to face to the other end. This reduces the time wasted on flitting.

(d) Insufficient attention to blasting. Sufficient coal is not dislodged in blasting due to less depth of holes and their bad spacing.

(e) The time lost after blasting for the smoke to clear up. This is a common cause for the delays in loading blasted coal in bord and pillar method where the ventilation is generally sluggish as compared to longwall.

(f) Efficiency of manual loaders. Bad lighting and bad ventilation, congesion of the galleries with brattices and equipment, reduced height of the roof due to cross bars, etc. affect the efficiency of manual loaders.

2. On the track and haulage planes

(a) Hand tramming wastes the energy and time of the trammers and loaders. Moreover turn-over of tubs becomes low.

(b) Proper gradient of the tracks, their packing, smooth curves, check rails, etc. do not receive enough attention with the result that derailments take place.

(c) Supply of tubs. Often the number of tubs available to a district is limited due to poor turnover of tubs in circulation. This may sometimes be due to long haulages. In a bad runaway of tubs and consequent derailment the smash may bulge some tubs which cannot enter the cage or tippler and such tubs have to be depth out of use till repaired. On the endless haulage track a tube which has bulged too much in a runaway smash may hamper movement of tubs on adjacent track. The bulged tubs have therefore to be kept away for repairs.

(d) Sometimes D-Links or couplings on the tubs are missing and the tub has to be kept out of use or the train length is reduced.

(e) Bad planning at loading ends underground. In a well planned arrangement, when one set of tubs is loaded the loaders should be able to load tubs standing on the other track till the loads are withdrawn and emptiy tubes are supplied.

3. On the surface

(a) When wagons are not available for loading, tipplers discharging on the screening plant are not used unless storage bunkers are provided. In such cases loaded tubs are often emptied by hand at dumping yard. This process is slow and turnover of tubs is seriously affected.

(b) During manual emptying the tubs which are tipped end-on or side-on, sometimes roll down the coal bank. They have to be brought back on the track. In an ill supervised mine such tubs are known to get buried under coal and this reduces their number in circulation.

Output Available With Solid Blasting

The availability of faces is the same as stated above for coal cutting machine. Solid blasting gives a progress of 1 m with each blast and with three shifts working, 4 blasts are possible in 24 hours, thereby giving a progress of 4 m per face. In a gallery of 4.2 m width and 2.1 m height the coal available per blast is 4.2 m × 2.1 m × 1 m = 8.82 cub metres (solid) and 8 cub. m. with minor losses. Therefore 32 cub (solid) coal or 32 te, is available with 4 rounds of solid blasting per face in 24 hours. Considering some minor losses it is reasonable to expect 30 te per face per day.

Thus the output available per day with solid blasting in a district as stated above is as follows :

(i) Main headings :

3 headings provide 5 faces. Assuming four faces available for solid blasting, output from them will be 4 faces x 30 te = 240 te.

Therefore 120 + 240 = 360 te of coal can reasonably be expected form new district of the size stated above during development. A round figure of 350 te of coal is adequate. For an output of 1000 te/ day, 3 districts in working condition and one district under preparation for working have to be provided in the development stage.

One contributory factor for low output is of administrative nature and not of technical nature. The piece rated workers like miners/ loaders have hardly any incentive to put in adequate efforts for production as they are entitled to fall back wages applicable to their category when they do not complete their normal work load.

Layout For Locomotive And Mine Cars

Fig. 11.10 shows a layout in a nearly flat seam using locomotives, mine cars, gathering area loaders and trolley mounted coal cutter. The galleries are nearly 4.8 m wide and the pillars rhombus-shaped (60°/120°) for easy turn round of the locomotive at corners. The junctions with the main heading are staggered to minimise roof exposure. One of the galleries with track is used by locomotive as siding for keeping the emptiy or loaded mine cars. The skid mounted coal cutter is not a matching equipment where locomotives are used as the yield of coal by such coal cutter in a shift due its slow speed of flitting and cutting is insufficient for transport by locomotives. Indeed a coal cutter which gives a face advance of less than 2 m per cut is not considered a matching equipment for transport by locomotives. The track of mine cars is always kept nearly touching the face for the coal cutter. If the face advance per cut is 2m, track length (simple multiples of 2m), such as 2m, 4m and 6m, are depth ready as extension pieces. The mine cars may be loaded by manual loaders or by gathering arm loaders.

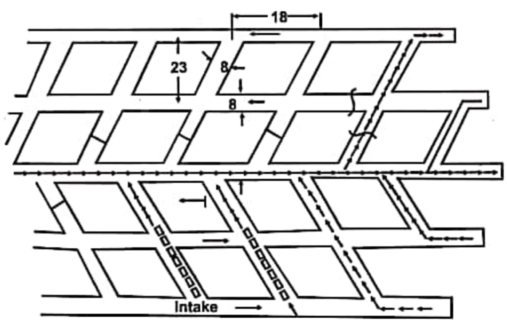


Fig. 11.10 Layout for a locomotive, gathering arm loader and mine cars.

Nearly similar layout with rhombus shaped pillars is adopted where shuttle cars are used in combination with mechanical loaders or continuous miners and a belt conveyor provides the main transport. The shuttle car does not discharge directly on the belt conveyor, but into a small bunker or hopper which releases the coal on the conveyor by a chute. The operating distance of the shuttle car is limited to 180 m due to restriction on the length (90 m) of the trailing cable. Where the distance between the face and the hopper exceeds 90 m, the driver has to disconnect the trailing cable from the gate end box during the midle of the run and reconnect it to cover the remaining distance. In a developing panel it is possible to give 8-12 cuts per shift (trolley mounted coal cutter) yielding about 200 te in the shift.

Continuous miners and tunneling machines are used in very few coal mines (hardly 1%) in India. The machines employed are AM-50 Alpine Miner and Dosco Dintheader. In United Kingdom DOSCO MK IIA Roadheader is by far the most predominant model operated by National Coal Board and over 550 of them are currently employed (in 1982) for underground drivage. Description of one such tunneling machine, Alpine Miner AM-50, is given in Vol. 3 of this series which also gives a general layout at the face. The machines can be used for development of a mine on bord and pillar system.

Some Restrictions On Development Of A Mine

Under the Regulations, working in a coal mine is prohibited in some areas unless permission is taken from the D.G.M.S. and other authorities. The places are :

1. Vertically below and within 45 metres of a railway, road, public works or buildings and permanent structures not belonging to the mineowner.
2. Vertically below a river, canal, tank or surface reservoir and also within 15 m from either bank of a river, canal or boundary of a lake or tank.
3. Winin 7.5 m of the common boundary of a mine. If the boundary is disputed, within 7.5 m of the boundary claimed by owner of a neighbouring mine.
4. Within 60 m of any disused or abandoned workings containing water.
5. Seams lying above or below a fire area.
6. At any place with a cover of less than 15 m.

Working Of Continguous Seams

If two seams are separated by a parting of less than 9 m they are known as contiguous seams. Their working, if both have to be worked at a time or one after another, needs prior approval of the D.G.M.S. A thick seam may be worked in two sections, one at the floor and the other near the roof and the parting between any two such sections of a thick seam or between two contiguous seams should be not less than 3 m thick. If the seams are within 9 m of each other, the pillars and galleries in one seam should be vertically above or below the pillars and galleries in the other seam if they ar not steeply inclined. This condition is not applicable if the parting is more than 9 m. Where the seams are not contiguous but separated by a parting upto

20 m thick, their development in panels is often planned in such a way that the panel barriers are vertically above or below those in adjacent seams but the pillars and galleries inside the panels are not vertically coincident. This is a good practice. To ensure verticality of pillar over pillar it is recommended that top and bottom seams should be under the charge of the same overman. The thickness of the parting should be ascertained from time to time by drilling boreholes at junctions of galleries after alternate pillars. In a thick seam if there is any band of stone, shale or inferior coal which is not workable economically, such band is allowed to remain in the parting between adjacent sections so that the thickness of parting is brought up to 3 m or more. Such bands help to maintain definite floor and roof gradients and nearly uniform thickness of section under development. They also help in reducing the roof convergence.

When working contiguous seams development of bottom-most seam is usually in the dip direction for water consideration and development in other seams is in other direction, *i.e.* towards the rise or strike so that the property is proved during development stage.

One haulage may serve both the seams through a drift. In such a case output from the seams can be brought to one seam, preferably the lower seam, from where it is hoisted up. A pit is normally sunk upto the bottom seam and coal of top seam is lowered by jig arrangement to the bottom seam.

It is a common practice to keep the bottom seam advanced at least one pillar ahead of the top seam and water of the top seam is allowed to gravitate to bottom seam though bore holes or a drift so that top seam workings are dry. From the bottom seam the water is pumped to the surface or to the main sump.

Ventilation of contiguous seams may be through a staple pit or through a drift.

✧ QUESTIONS ✧

1. Under what circumstances is the bord and pillar method of working adopted ? Why is it generally followed in our country ?

2. Show by diagram how a district in a mine is opened off the strike if the entrance to the seam is (a) by an incline, (b) by a pit. Assume a gassy mine (deg. 2) and rope haulages.

3. What is the meaning of the following terms :

(a) Projection plan,	*(b) gate conveyor,*
(c) district,	*(d) crosscut,*
(e) development	*(f) panel,*
(g) incubation period,	*(h) spontaneous heating,*
(i) contiguous seams,	*(j) caving.*

4. Which are the places in a mine where workings are prohibited without prior permission of the DGMS or other authorities ? State the distances specified in connection with each restriction.

5. State the considerations to be borne in mind when developing a steep seam with the help of a coal cutting machine.

6. Give the layout of a district of a seam dipping at 1 in 20, if it has to be developed using trolley wire locomotives, trolley mounted coal cutting machines and gathering arm loaders. Mark on the plan the operations at different faces at any instant.

✧ ✧ ✧

Chapter -12

Pillar Extraction In Bord & Pillar

After pillars have been formed on the bord and pillar system, consideration has to be given to the extraction of coal from the pillars; the operation is known as pillar extraction. It is also referred to as *depillaring, pillar-cutting or broken working.*

In a method of depillaring, known as the caving method, the coal of the pillars is extracted and the roof is allowed to break and collapse into the voids or the decoaled area, known as goaf. As the roof strata about the cola seam break, the ground surface develops cracks and subsides, the extent of damage depending upon depth, thickness of the seam extracted, the nature of strata, thickness of the subsoil and effect of drag by faults.

Depillaring with stowing is a method of pillar extraction in which the goaf is completely packed with incombustible material and In generally practised where it is necessary to keep the surface and strata above the seam intact after extraction of coal. The following circumstances would require adoption of depillaring with stowing :

1. Presence of water-bearing strata above the coal seam being extracted. e.g. Kamptee series in Wardha Valley coalfield of Maharashtra. Enormous quantities of water beyond the economic pumping capacity may enter the mine through cracks in the strata.
2. Railways, rivers, roads, etc. situated on the surface, which cannot be diverted.
3. Presence of fire in a seam above the seam to be extracted.
4. Existence of one or more seams of marketable quality extractable in the near future.

5. Restrictions imposed by local or Government authorities for the protection of the surface.
6. Extraction of the full thickness of a seam thicker than 6 m, as thicker seams cannot be extracted fully by caving method.
7. Extraction of seams very prone to spontaneous heating, of very gassy nature or liable to pumps.
8. Surface buildings which cannot be evacuated.
9. Tanks, reservoirs, etc. which cannot be emptied.

Where it is not practicable or economic to adopt complete stowing of goaf but all the same it is essential that the surface and strata above the seam remain intact, a method known as "splitting" the pillars is practised as a final operation. Maximum possible coal is extracted form the pillars by driving galleries through them so that the portions of the pillars left in situ support the overlying strata.

Caving is the general method adopted in majority of our mines at it does not involve the high cost of stowing arrangements. For depillaring with stowing availability of stowing material at a reasonable cost is a pre-requisite. River sand is the abundant stowing material commonly used and its hydraulic conveying is the general practice.

Preparatory Arrangement Before Depillaring

The depillaring operations are so planned and conducted that (a) there is the least danger to the safety of the mine and the men employed underground or on the surface, (b) maximum possible extraction is achieved (c) extraction of other seams or sections of seams is not rendered difficult, (d) the extraction is economic and (e) dangers from crush or collapse of pillars, fire, influx of noxious gases, or inundation are avoided. Effect of fire or water-logged areas in the adjacent mines has also to be considered.

Before undertaking depillaring operations, permission has to be obtained from the D.G.M.S. in all cases, from the Railways if the operations are likely to affect a railway, and from the district authorities if District Board roads or other buildings are affected. Diversion of railway, stream, road, power transmission lines, telephone lines, aerial rope way, etc. has also to be considered. Rehabilitation should receive attention. Surface land likely to subside has to be purchased. All these arrangements take 1-2 years.

We shall consider here the arrangements preparatory to depillaring by caving method :

(1) If there is a seam above, developed and filled with water, or "goafed" and water logged, it has to be dewatered. In case the top seam is developed, and abandoned temporarily, pumps can be installed in it for dewatering. Top seam may be dewatered through a shaft if it is on the dip side and is accessible on the surface. The shaft may be equipped with a submersible borehole pump. In most of the cases of depillaring of top seam the shaft pillar might have been extracted and the access road to it on the surface subsided. Under such circumstances pumps cannot be installed in the top seam and its dewatering is by boreholes drilled from bottom seam workings. Such borehole drilling calls for accurate surveying. The boreholes should be made by a drilling machine like the VOLSAFE drilling machine which can control flow of water during drilling. Dewatering of the top seam from the bottom flow of water furing drilling. Dewatering of the top seam from the bottom seam may be done in stages if the depillaring in bottom seam is to proceed from rise to dip.

(2) Plans. The plans are brought uptodate showing all the new constructions, position of boreholes, staple pits, etc. The position of fault planes and dykes in the leasehold and also in the area of an adjacent colliery should be shown. The goaf area of all these seam in the same mine and the goaf areas of the adjacent mines also should be indicated. History of mining operations in the adjacent mines should be ascertained from role experienced Managers, overmen or surveyors.

Sufficient allowance should be made for inaccuracies in the plans which may be intentional or unintentional. Plans and history of the adjacent abandoned mines will be available from the office of the D.G.M.S.

The thickness of barrier in adjacent mines that have been depillared should be treated with caution as the barrier might have been "robbed" during retreating though the plan shows adequate thickness.

Plans of the area to be depillared are prepared on a large scale showing the panels, the pillars to be extracted, the sequence in which pillars have to be split and extracted, and the pillars to be left intact. Pillars underground should be numbered with red lead and the pillar to be left intact should be marked all round the perimeter with a band of limewash 300 mm wide at a height of 1.5 m from the floor.

Tracings of such large plans should be given to the under managers, overmen and also to the mining sirdars.

(3) The adjacent mines likely to be affected by depillaring e.t.; mines on the dip side that will have to pump extra water, have to be informed.

(4) Adequate number of supervisory staff, workers and specially the mines who have experience of depillaring, have to be appointed; strength of timbermen may have to be increased,

(5) There should be adequate stock of the following materials.

(i) Timber : The demand is heavy during depillaring. In a thick seam the size of props normally not required during development will be in demand during depillaring. Construction of cogs requires logging sleepers in a large number.

(ii) Fire fighting equipment and fire sealing equipment e.g. stone dust, bricks, lime, cement, C.G.I. sheets, flame proof brattice cloth, etc.

(iii) Protective equipment like helmets, hoes, etc.

(6) Production during depillaring is likely to be high and the magazine capacity, number of tubs in circuit, haulage capacity and strength of trammers may have to be increased.

(7) Capacity of pumps should be increased to deal with extra water coming underground through surface cracks.

(8) Outbye pillars have to be stabilised by sand stowing or substantial timber supports, preferably chocks.

(9) Isolation stoppings must be constructed to form artificial panels of convenient size within which any heating or fire caused during depillaring may have to be confined and sealed. Sized of artificial panels should be such as to complete the extraction within the incubation period.

(10) Systematic timbering rules are to be framed and explained to the supervisory staff and timbermen. A copy of the rules, suitably illustrated, has to be posted at conspicuous places in the mine. Systematic timbering is compulsory in depillaring areas.

(11) In a mine every working place, where practicable, should be provided with at least two ways affording means of escape to the surface (Reg. 69). The fact should always be borne in mind for depillaring districts where the operations are usually hazardous. It is, however, not always possible two escape routes e.g., during splitting.

Depillaring By Caving

We shall consider the operations in a seam of

(A) Thickness upto 3.0 m (B) Thickness 3 to 6 m

(A) Seam thickness up to 3 m

The pillars formed during development are split into small pillars called "stooks" which are then extracted one by one. A gallery driven in the pillar for this purpose is called a "split". At shallow depth, below 100 m depth, pillars during development are small and they can be extracted without splitting. General, a pillar greater in size than 18 m × 18 m is split up into two stooks. In a large pillar of 30 m × 30 m size between centres, one split 3.5 m wide in the middle of the pillar along the strike and another split of similar width in the middle of the pillar along the dip, are driven so that the pillar is divided into 4 stooks. (Fig. 12.1). The splitting may start from 4 middle points of the pillar simultaneously.

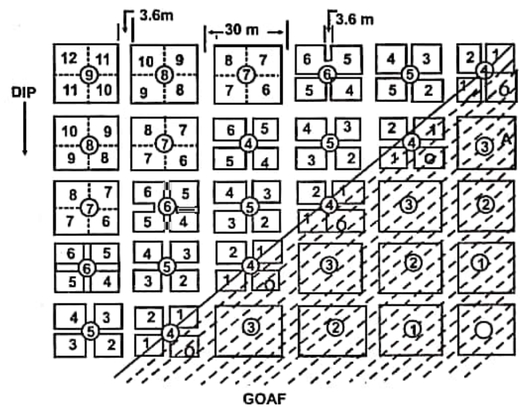


Fig. 12. 1 Sequence of pillar extraction with diagonal line of face.

The Coal Mines Regulations permits splitting of a maximum of four pillars when pillar extraction is about to begin. The width of split gallery should not exceed the width of development galleries at that depth (Reg. 100). Once the pillar extraction is commenced, the splitting of pillars may be done upto 2 pillars (maximum) ahead of the pillar under extraction.

During development, height of a gallery does to exceed 3 m. If the seam is thick, the development galleries and the split galleries are heightened so that all the coal upto the roof is removed using portable wooden stools to stand on. If the roof consists of shale, 0.6 m thickness of coal is left intact in the roof as shale has a tendency to part from roof rocks when exposed to atmosphere for some days. Props are erected generally 1.2 m apart and cogs 3 m apart for roof support.

In Fig. 12.1 the figures in circels indicate the pillars which will be attached at a time. The plain numbers (uncircled) indicate the stooks which will be extracted simultaneously to maintain a diagonal line of fracture.

Due to weight of roof on the pillars their coal is loosened and can be obtained easily by miner's picks without blasting.

A pillar 30 m size from centre to centre is generally 26 m corner to corner. Splits of 3.5 m width give stooks of nearly 11 x 11 m. The miners attack the stook from the side remote from the goaf or barrier so that the coal in the stook always stands between the miner and the goaf and protects him from the falls of roof in the goaf.

Fig. 12.2 shows the manner of attack and sequence of extraction of the stooks.

The arrows indicate the direction of attack by miners. The smaller pillar M indicates the fender or Chowkidar, that is, the solid coal of stook or pillar left intact during extraction of the stook or pillar. Roman numerals indicate the sequence of extraction of pillars and stooks.

(a) Roof is good and reliable.

(b) "Half moon method". Roof reliable.

(c) Roof is bad.

(d) and (e) – Roof is bad and unreliable. Least area of roof is exposed.

The loaders or miners form a team of 3 to 4. They drill the holes the shot-firer blasts them and the miners dress down the coal and load into tubs or mine cars. One team requires a span of about 4 to 4.5 m on the side of the stook.

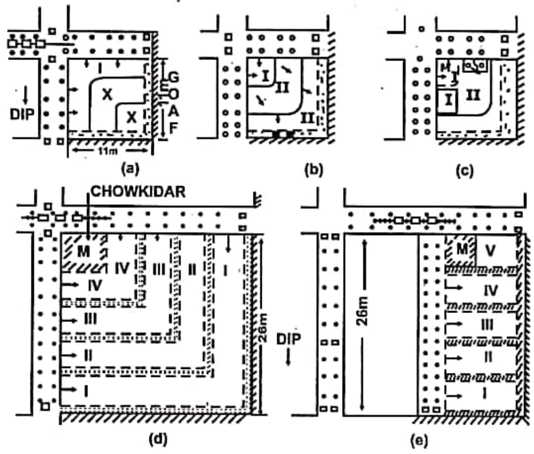


Fig. 12.2 Extraction of stocks under good and bad roof condition

At one time 3 to 4 such stooks are under extraction and the other miners are employed on splitting and gallery heightening jobs.

During extraction of coal a rib about 1 metre thick is left against the goaf. When all the slices are removed, and the area is to be abandoned, the rib coal is "robbed" to the maximum extent possible. When no more coal is extractable from the stooks, without endangering safety, props and roof supports are withdrawn and the area is then abandoned. Chocks are erected at goaf edges and the area is fenced off.

Depillaring of seams from 3 m to 6 m thick by caving is given in the chapter on "Thick seam working".

Behaviour Of Roof And Its Control During Depillaring

When the galleries are widened or heightened, or splits formed, the equilibrium of pressure that had stabilised during development stage in the strata overlying the seam and immediately below it is affected and the re-adjustment of pressure causes roof movement. When the stook or pillar is extracted, such roof movement is more pronounced due to the removal of the support so far offered by the solid coal of stook or pillar. Sudden transfer of pressure to any pillar or stook may result in crushing and the aim during depillaring is to avoid sudden accumulation of pressure over standing pillars or stooks and the supports erected outbye the goaf edge. Once the roof movement sets in, the kinetic energy is difficult to control by supports.

During depillaring, no splitting or reduction of pillars or heightening of galleries should be done beyond a length of two pillars ahead of the pillar under extraction. Galleries which are heightened or widened should be supported by rows of props, 1.2 m apart and by chocks 3 m apart. Chocks are necessary at junctions of the galleries, or splits and at goaf edges. Systematic timbering rules specify the manner of supports and the spacing of supports.

Supports should be erected without delay in the void formed due to pillar extraction, leaving enough room for movements of workers, swing of tools, movement of tubs, etc.

Even when supported by timber the area exposed should be kept to a minimum. With a good sandstone roof, the area explored may not be more than 40 m^2 at one stook extraction. With 3 or 4 stooks under attack at one time, the total area should not exceed 900 m^2. Each case, however, should be decided on the experience gained about the nature of the roof and its behaviour. The roof strata are relieved of the accumulate pressure energy when the pressure breaks up the roof. Once such energy is dissipated in the breakage of roof the pillars are not under stress and working conditions are comparatively safe. It is therefore, necessary to see that the roof in the goaf area is induced to fracture at regular intervals. The Regulations require that the depillaring operations should be conduced in such a way as to leave as small as area of uncollapsed roof as possible, and there possible, suitable means shall be adopted to bring down roof in the goaf at regular intervals.

Such suitable means consist of :

(1) Withdrawing all the roof supports beyond the goaf edge so that the goaf is unsupported and the roof breaks due to pressure of superincumbent strata.

(2) Drilling a few holes in the roof near the goaf edge and blasting them, or

(3) Changing the line of goaf or line of face.

The collapse of roof in the goaf which is deliberately brought about by withdrawal of roof support is called normal collapse as distinct from a *premature collapse*. Such *normal collapse* is an everyday incident in the depillaring area and steps can be taken to prevent danger to workers or property when the collapse or roof fall is anticipated. Premature collapse, as the adjective implies, occurs at a place where collapse is not expected and is therefore dangerous.

A strong thick stratum of massive sandstone or a sill in the roof, immediately above the seam, will delay the fall of roof and when a sufficiently large number of pillars are extracted the main roof will ultimately break with the build-up of pressure. The weight of the strata is transferred, before the fall takes place, over the pillars on the outbye side. If the pillars are not strong enough, they may be crushed due to such sudden heavy pressure. This phenomenon is known as overiding of pillars and stooks and results ultimately into premature collapse in the area. This shows the importance of stabilising weak pillars outbye of the area under extraction in a depillaring district and such stabilisation is an essential step before commencing depillaring.

Ensuring Regular Roof Fracture

The supports are withdrawn when all, or nearly all, the coal in the stook is extracted, or when the roof shows signs of heavy roof pressure.

The indications available of roof weight are :

1. The prop, when tapped with a stick, produces a dull and drummy sound.
2. The lid over the top of the prop bends.
3. The prop buckles in the middle or at weaker points in its length.

4. If the floor is soft, the prop may penetrate it.
5. There may be rumbling sound in the goaf.
6. If a small crack existed in the roof and a wedge is hammered in it when supports a erected, the wedge becomes loose and falls as the crack widens when the roof pressure increases and roof tends to come down. This method has been used to get indication of roof pressure at Giridih Colliery and at other collieries.
7. In some cases water percolation from the roof takes place as the cracks widen. This is the first indication of roof movement. With more roof movement the crack may widen further with increased flow of water, or the flow may cease altogether as the entrance of water into the crack may be closed by movement of rocks. This had been observed at Giridih Colliery.
8. Intermittent fall of loose rock piece from the roof in goaf increases, and small coal pieces shoot off the sides.
9. Booming sound of roof movement increases.
10. Heaving of floor is noticed.

    The time of withdrawal of props has to be decided by experience. If the withdrawal is delayed, the increasing roof pressure breaks them and the weight has to be borne by the outbye stooks which may get crushed and be eventually lost.

All the props and chocks have to be withdrawn only by mechanical prop withdrawer or similar contrivances. Hammer should not be used to dislodged a prop from its position. The furthest row of props in the goaf should be withdrawn first and the withdrawal should proceed towards the safer place. As far as possible, no supports should be left in the goaf in erected position as they retard the regular roof break. If a prop cannot be withdrawn it may be destroyed. The chocks should be erected at the goaf edges before prop withdrawal and the openings outbye the goaf edges should be strengthened by erection of props and bars. Such goaf edges should be fenced off after all the timber is withdrawn. It is not always possible to withdraw all the props and cogs, particularly those at the goaf edge.

Local Fall

After timber has been withdrawn the unsupported roof tends to fracture. The lower strata of the immediate roof separates and fills in the goaf. The fall which takes place soon after withdrawal of supports is called local fall. Where the stratas are well stratified sandstone or shales, the local fall takes place within 24 to 48 hours of the withdrawal of timber. Such local fall does not extend to the surface and rocks within a height of only a few metres above the seam is affected. The intensity of the booming sound in the goaf due to roof movement gradually increases after withdrawal of supports and ceases after the local fall.

Air Blast

When a roof fall takes place, the air in the goaf area is displaced as the latter is filled by the broken roof rocks. The displacement is so quick that the air is pushed out at the high speed resulting in air blast if the air has limited outlets to escape. The intensity of the air blast depends upon the volume of the air that is displaced. If the immediate roof is massive sandstone or has a sill, the local fall does not occur for a long period even after withdrawal of supports. The volume of air that is displaced in such case when a roof fall takes place is large. Similarly the volume of air displaced in a thick seam is also large. The air blast in these cases is strong and dislodges roof supports and ventilation doors, damages isolation stoppings and injures workers in the path of the air blast. The dislodged roof supports may initiate some additional roof collapse.

Precautions against air blast

1. Steps should be taken to see that an extensive area of uncollapsed goaf does not exist at a time. Where possible, steps should be taken to bring down the roof at regular intervals.
2. In any depillaring district, all the workers should be withdrawn to a safe place when the usual indications of imminent roof fall are observed. Where immediate roof is well stratified sandstone and shale, the work has often to be suspended for 2 to 3 hours or more before the roof fall takes place. If the roof consists of massive sandstone, the sound of roof-movement in the goaf lasts for 8 to 16 hours and coal-getting operations in the area have practically to be suspended for such a long period. The mining sardar or overman should warn the workers by a whistle to retreat to a safe place which is not directly in the path of air blast.

3. Apart from the entries for ventilation and haulage, additional roadways should be kept open in a thick seam.
4. After every blast there is usually a thick cloud of coal dust in suspension in the air. The electric switches should be put off before such anticipated air blast and put on only after coal dust has settled down.
5. Construction of a few isolation stoppings with an easily breakable zone (by air blast) should receive attention.

After the roof fall has taken place all the supports and the roof should be examined before workers are allowed to the working place. Special attention should be given to the goaf edges which should be well fortified by supports after the roof fall.

It has been observed in some collieries where the immediate roof is a massive sandstone that some districts may have areas of uncollapsed roof as extensive as 60 m × 50 m. One or two props erected furing pillars extraction are left erect in the area when all other roof supports are withdrawn. Such props serve to indicate roof weight and the mining sardar observes the condition of the props from the goaf edge by a powerful torch. The roof fall in such extensive goaf area is a serious matter.

Main Fall

The local does not extend to the surface as the broken shale and sandstone occupy a large volume after breaking and fill up the void caused by coal extraction. It has been observed that when the area of extractoin is nearly equal to or more than, the square of the depth, the cracks due to depillaring extend upto the surface. The roof fall which affects the surface is known as main fall and takes place long after the first roof fall (local fall) in the depillaring district. If the seam is at a shallow depth air leaks through the cracks to the coal buried in the goaf and is responsible for spontaneous heating. Such spontaneous heating has been the cause of fires in the areas near Jharia and Katras towns.

The main fall is expedited if (a) area is near a fault plane and (b) if the upper seam has been depillared. At Bhagaband colliery, Jharia field where the top seam is depillared, it is observed that during depillaring of bottom seam, main falls takes place much earlier than the period normally observed during depillaring of top seam.

Depillaring operations in the vicinity of faults need more than normal vigilance and precautions for roof control. Water seepage or emission of gas may take place near a fault. The line of goaf should not be parallel to the fault, but nearly at right angles to it. Heavy roof fall and even premature collapse may take place if line of goaf is parallel to the fault. The unsupported roof may slide along the fault plane. It is a sound practice to leave some barrier of solid coal near the fault.

Line Of Goaf

A line passing through all the corners of stooks under extraction at a time is called line of extraction. It is sometimes loosely called "line of face" or "line of goaf". A line of extraction which is diagonal, i.e. nearly 45° to the dip direction is common as shown in Fig. 12.1. With such line of extraction each pillar contributes some portion of it towards roof support.

The step diagonal line of goaf is as shown in Fig. 12.3 . In this case one solid stook intervenes between two stooks under extraction at a time. The line of goaf is more inclined towards the strike than in the earlier case. Where roof is bad, the presence of one solid stook between two stooks under extraction gives good support to the roof. As stooks are extracted, the intervening stook may be subjected to heavy pressure and its crushing indicates roof weight. Steep diagonal line of goaf may be so arranged in some cases as to have two solid stooks in between the stooks under extraction.

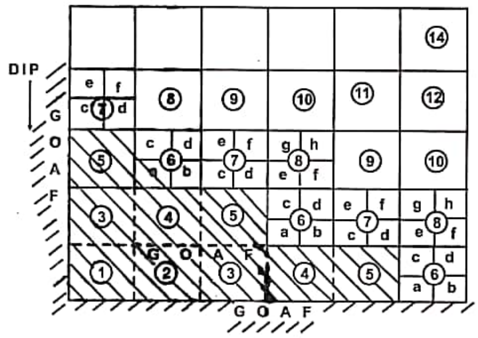


Fig. 12.3 Step diagonal line of face

A line of extraction that forms as arrowhead in the goaf as shown in Fig. 12.4 should be avoided. The pillar or stook at the arrowhead is under excessive pressure and is liable to be crushed.

Where conveyors are used in a depillaring district, it is necessary to have a straight line of extraction so as to reduce the number of conveyors and lead for manual loaders. In Raniganj field some mines using conveyors in depillaring district have experienced no trouble with line of goaf parallel to dip or strike doing to favourable direction of cleavage planes. A line of goaf parallel to the strike has the advantage that the goaf can be submerged in water if the depillaring is proceeding from dip to rise and the seam inclination is not mild.

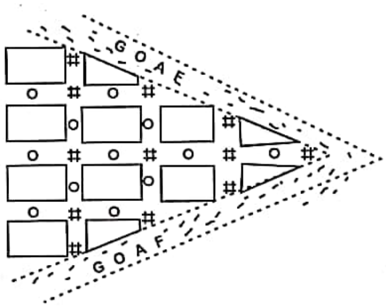


Fig. 12.4 Line of face in the form of arrowhead in the goaf. Avoid it

Precautions Against Fire During And After Depillaring

Where a roof fall takes place in the goaf after withdrawal of supports, roof coal and the coal of fenders, unextracted ribs and partially extracted stooks, is buried under the roof fall. The conditions conductive to spontaneous heating are then created and are present after the first roof fall as the heat of oxidation of the buried coal is not dissipated. The period that elapses from the first local fall till the signs of heating in the goaf is called the incubation period. It varies from seam to seam. AT Churi Colliery (North Karanpura), it is observed to be 3 to 4 months; in the Kajora seam in Raniganj coalfield it is nearly 4 months. In one mine in Raniganj coalfield working Poniati and Koithi seam, Poniati is known to show signs of heating in 9 months but Koithi is quiet even after 2 years. The top seam in the Pench Valley Coalfield has an

incubation period of 9 months; lignite at Neyveli, when stocked at surface is known to catch fire due to spontaneous heating in one month. It is observed that some seams are not liable to spontaneous combustion at all.

It is necessary to take steps to prevent spontaneous heating in the goaf and to prevent the spreading of heating or fire.

The steps are :

(1) Depillaring operations should be carried on in panels. Such panels may be formed during the development of the district by solid coal barrier all round the district with only essential galleries for transport and ventilation, and the panel should contain only such number of pillars as can be extracted within the incubation period which, however, is not known during the development stage unless experience is available of the adjacent collieries practising pillars extraction by caving in the same seam. Where the panels are not formed or, if formed, contain such large number of pillars as cannot be extracted within the incubation period, it is imperative to form artificial panels of convenient size. Such artificial panels are formed by isolation stoppings built of brick in lime or cement.

Isolation stoppings in de. 1 gassy mines are minimum 1 m thick of brick in lime or cement. If heavy airblast is anticipated during depilating the stopping should incorporate a weak zone of say 0.3 m × 0.3 m which is not 1 m thick but only 250 mm thick in the middle of the stopping so that it gives way during the airblast and provides a vent to the pressure behind the stopping. In deg. 2 and deg. 3 gassy mines the isolation stopping should be explosion proof. Such stopping consists of a brickwall not less than 1.8 m thick, or of two walls, each 1 m thick, 4.5 m apart having incombustible material like sand or boiler ash inbetween.

It has been observed that in thick seams, fire behind an isolation stopping sometimes bypass the stopping through the roof coal or floor coal and affects the coal outbye of the stopping. It is therefore, necessary that an isolation stopping should be constructed from floorstone to roofstone.

(2) Depillaring operations should be conducted from dip to the rise, so that goaf can be submerged in water. If however, the seam has a mild gradient or if depillaring is conducted with a diagonal line of extraction, the goaf on the rise side is not drowned under water.

Blackdamp which may be formed during depillaring in some seams tends to the dip areas in the goaf; and by blanketing the fallen coal it helps in preventing its oxidation.

(3) An attempt has to be made not to leave any coal in the goaf as far as practicable. This, however, becomes difficult unless the roof is good and permits complete extraction from stooks.

(4) In shallow seams, to prevent breathing of air to the depillared area through cracks to the surface, the cracks have to be blanketed with a layer of sand or earth nearly 1.2 m thick. Such blanketing is often done on the surface before depillaring commences. After the cracks are formed, it is dangerous to cross the cracked surface.

Precautions Against Inundation During Depillaring

These may be summarised here as follows :

1. Enough allowance should be made for inaccuracy in the plans. Inaccurate mine plans have been the cause of accidents due to inundation in 90% of the cases during depillaring. All the features required to be shown in the plan under the Regulations should be clearly shown. A common ommision is boreholes, as these are not conspicuous, and often forgotten after prospecting.
2. When depillaring zone is within 60 m of waterlogged area, advance boreholes have to be drilled. "Volsafe" machine for drilling at any angle is suitable for the purpose.
3. Depillaring below a water-logged area should be avoided. Streams, rivulets, etc. on the surface should be diverted if the expenditure is justifiable.
4. If overlying strata contain a water-bearing stratum, unless pumping capacity is adequate, caving method should not be adopted.
5. No depillaring operations should be conducted in an area which is likely to cause subsidence of the surface below the highest flood level of river, stream, or lake.
6. When depillaring in bottom seam is to proceed from rise to dip, the top seam may be dewatered in stages by advance boreholes from bottom seam with prior permission from the DGMS. As illustrated in Fig. 12.5 when depillaring takes place at A, advance boreholes may be made at C to dewater the top goaf.

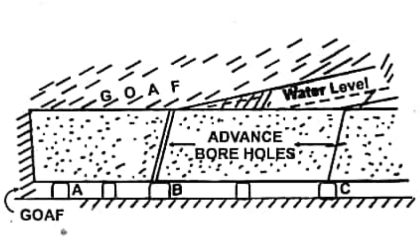


Fig. 12.5 Dewatering waterlogged goaf of top seam in stages. Depillaring proceeds from rise to dip

Depillaring Of Contiguous Seams

The method of extraction of contiguous seams depends primarily on the thickness of the parting. As has been stated earlier, development galleries are driven in the seams such that galleries and pillars are vertically coincident. In some mines, a seam thicker than 7.5 m is developed in two sections, leaving a parting of 3 m and extraction takes place as if there were two contiguous seams.

If the stone in the parting between the two continguous seams, less than 1 m thick only one seam is developed and the two seams are treated as one seam during pillar extraction. The stone parting is blasted out and thrown in the goaf. If the stone parting is between 1 and 3 metres so that it is not economic to blast out the stone, the seams or sections are depillared simultaneously, that is, the depillaring is conducted in the two seams or sections such that the line of extraction of the lower seam is vertically (or nearly so) below that of the upper seam (Fig. 12.6). If one of the seams or sections is not developed, entry is made into the virgin seam through a drift starting at a suitable point in developed seam. The undeveloped seam is partially developed for one or two pillar-lengths at a time and pillar extraction is carried out simultaneously, keeping the supports almost vertically coincident. The withdrawal of supports is done simultaneously. If the top section supports are withdrawn before those in the lower, the roof fall in the top section punctures the parting and affects the bottom section's supports; if bottom seam supports are withdrawn first, the parting may collapse and supports in the top section may be lost in the collapse. Coal of both the sections is loaded at one point wherever possible.

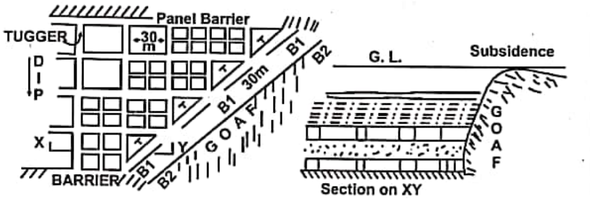


Fig. 12.6 Simultaneous depillaring in contiguous seams. TT is line of goaf in top seam. Line of goaf in bottom seam is shown by B1B1 if parting is thin and by B2 B2 if parting is thick.

When the stone parting is thicker than 3 m the two seams are developed with galleries and pillars vertically coincident either throughout the mine or a panel, and extraction proceeds in the two seams at the same time almost simultaneously with line of goaf in the top seam not more than one or two pillars ahead (i.e. outbye) of the bottom seam line of goaf.

To co-ordinate the work in the seams, it is better to have the depillaring districts of the two seams under the supervision of one overman. A good system of communication is essential. This takes the form of, (i) telephones (ii) signalling bells connected by wires laid through a borehole drilled in the parting (iii) A 18 mm diam. pipe fitted in a borehole in the parting; the pipe is recoverable.

Working Below A Goafed Area

With the parting between the seams thicker than 9 m it is not necessary to insist on simultaneous pillar extraction. The usual arrangement is that the seams are extracted in a descending order. The following points may be borne in mind :

1. The extraction in the lower seam should be conducted under a settled goaf of the top seam so that the parting is not subject to the impact of kinetic forces let loose when the strata movements in the goaf takes place. A period of 3 to 5 years is considered sufficient to allow the goaf to settle.
2. Steps should be taken to guard against inundation from water-logged top seam goaf.

3. The main fall is much quicker if top seam is goafed and extraction must be planned with this in view.
4. Heavy timbering is essential because of the dead weight of top seam goaf.

Working Over The Goaf Of A Lower Seam

Under Regulation 104, no work in a higher seam or section shall be done over an area in the lower seam or section which may collapse. The working in the upper seam is, therefore possible only when it is virgin and the lower seam goaf has settled down. As stated earlier, it takes 3 to 5 years for a goaf to settle down. An attempt can be made to work such virgin seam if the parting between the seams is 30 m or more. With a parting less than this, the top seam may be assumed to be unworkable.

If the top seam is developed before depillaring in the bottom seam, roof of top seam develops numerous cracks with depillaring in bottom seam and it is too dangerous to work in top seam.

A virgin seam over goaf of a bottom seam was worked at Bhatdih Colliery (Jharia field) and at another mine in Raniganj field working Koithi/Poniati seams. The following observations are based on that experience.

In one area where a higher seam (Koithi), was to be worked over the goaf of a lower seam (Poniati), it was found that the top seam was disturbed in patches, and in some places, titled. Parting between the two seams is 40 m. Where the lower seam was extracted in panels, upper seam portions over the panel barriers remained intact while there was noticed an abrupt dislocation over the periphery of the excavated portion and the virgin seam became somewhat trough shaped in the area corresponding to the middle of the excavated panel of lower seam. These conditions made it difficult to work the higher seam with coal-cutting machines and to plan for normal direct or endless rope haulages. Workings were, therefore, done by pick mining and main and tail haulages were installed to cope with undulating gradients. In the zone of sudden dislocation and fractured ground, roof conditions were bad and close timbering, mostly cogs, with occasional use of forepoling, had to be resorted to. Care had to be taken to dewater waterlogged areas of the lower seam when approaching a level in the higher seam which was likely to be at par or lower than the water logged workings of the lower seam. Precautions have to be taken against gases from the lower seam finding an access to the higher seam through cracks. Ventilation by focing fan was found to be advantageous for this reason. Doing to the risks of cracks and hardened coal, permitted explosives of adequate strength had to be used. Now-a-days solid blasting is the general practice since coal cutting machines cannot be used and precutting at a face is out of question.

Bumps In Coal Mines

Coal bumps generally occur in deep mines. These are sudden violent bursts of coal from pillars, usually accompanied by an air blast and shooting of coal pieces at terrific speed. The amount of coal ejected in a bump may be a few hundred tonnes. Gas may also be released. Coal bumps are the result of a sudden release of elastic strain energy stored in the pillars. They are in this respect similar to rock bursts. Coal bumps are found only in coal seams with specific geological and mining conditions. There are two types of coal bumps; shock and pressure bumps. Shock bumps occur where a strong massive stratum lying immediately above the coal ruptures as a beam or flat arch and sets up a shock wave that is transmitted downward to the coal pillars. If the underlying coal pillar is already highly stressed, the transmitted shock wave may cause the pillar to bump. Pressure bumps occur when weak pillar is stressed beyond its strength and fails suddenly and violently. It will be obvious that bumps occur when stresses larger than coal strength are induced, whatever be the cause.

Bumps occur usually under the following conditions ; (1) a strong and brittle coal that does not crush easily, (2) Depth of over 150 m (3) a strong overlying rock stratum such as sandstone that overhangs into the goaf, (4) a strong floor rock that does not heave readily, and (5) improper mining methods that create localized extensive stress concentration. The last item depends upon the planners of the mine working whereas the first four items are the creations of Nature.

Depillaring With Stowing

This has been dealt with in the chapters on sand stowing and longwall mining.

✧ QUESTIONS ✧

1. What are the preparatory arrangements to be done in a mine before commencing depillaring operations *(a) with caving, and (b) with hydraulic sand stowing ?*

2. Describe the method of depillaring a seam 3 m thick. Give a plan showing the sequence of pillars and stooks to be extracted.

3. What are the precautions to be taken when depillaring continuous seams by caving ?

4. What are the dangers during depillaring in respect of *(a) inundation (b) air blast* ? What steps should be taken to guard against them ?

5. Write a note on the behaviour of roof during depillaring with caving.

6. A virgin coal seam has to be developed over the goaf of a coal seam, 5 m thick, 30 m below. State the possible problems to be faced and how they have to be tackled.

✧ ✧ ✧

Chapter -13

Longwall And Other Methods of Working

Apart from the commonly practiced "bord and pillar" method of mining followed in this country, the other methods of mining adopted in some collieries are as follows :

1. Longwall advancing
2. Longwall retreating
3. Horizon mining
4. Level mining or semi horizon mining
5. Inclined slicing and sub level caving

Room and pillar method of mining is not practiced in any of the collieries in India though it is the standard method adopted in American mines. Horizon mining a term wrongly used by some for a method of mining known by a distinct nomenclature, level mining, but the former term only covers a method of approach to the coal seams and is not a method of extracting coal. Where horizon mining is adopted, the method of mining may be longwall advancing, longwall retreating or any other method of coal extraction.

LONGWALL MINING METHOD

Longwall method of working consists in laying out long faces (60-200 m long) from which all coal in working section of the coal seam is removed by a series of operations, maintaining a continuous line of advance in one direction and leaving behind the void (called goaf). The roof over the goaf is partially or completely supported by walls of stone (called pack walls), sand or other material like crushed stone to prevent collapse of roof and only a small strip 3 to 6 m wide

and parallel to the face is supported by timber or steel props, bars or chocks in a systematic manner. Alternatively the roof over the goaf is allowed to cave in but the roadways are secured by packwalls and chocks if they have to be used (as in longwall advancing method).

In longwall advancing extraction of coal commences from the vicinity of the shaft pillar and proceeds outward towards the boundary of the mine or panel. Approach to the face is by parallel roads, formed at a specified distance apart which is equal to length of the face. The roof over the goaf may be supported by packwalls or sand stowing or allowed to cave in. In longwall retreating pairs of headings are driven in solid coal, certain interval apart, to a predetermined boundary where they are connected by a long roadway to provide a longwall face. Extraction of coal then commences from the boundary and the coal face retreats towards the shaft. In this system goaf packing is not essential for roof support if subsidence of surface or strata above the seam is permitted.

Longwall method is the standard method of working in coal seams in Britain, Germany and other European countries. In India only a few mines have adopted it and that too in conjunction with hydraulic sand stowing, with very few exceptional cases of longwall with caving.

The roads at either end of the face are known as *gate roads.* Usually the coal transporting gate road is the intake airway and the other gate road is the return airway. The coal transporting gate road is called the *haulage gate* and the other gate, *tail gate.* If rope haulage is used in the haulage gate, it serves for material supplies and also for coal transport. But if a belt conveyor is used in the haulage gate, the other gate road (tail gate) serves for material supplies and is called *supply gate.*

Fig. 13.1 shows a layout of single-unit faces and two short length double-unit faces. Where the gate belt conveyor (or other transporting medium) serves only one longwall face, the latter is known as single unit face; where the gate belt conveyor (or other transporting medium) is centrally situated to serve two adjacent longwall faces progressing in the same direction, the combined face is known as double unit face. Fig. 13.2 shows a longwall advancing double unit face with hydraulic sand stowing. It may be noted that if hydraulic stowing is practiced the two longwall faces should be so staggered that stowing water of rise face does not flow to the dip face but is drained away by the central gate.

Longwall method is normally employed to work thin seams with thickness ranging from 0.6 to 2.4 metres lying at moderate depth from the surface, with inclinations from flat to 20^0.

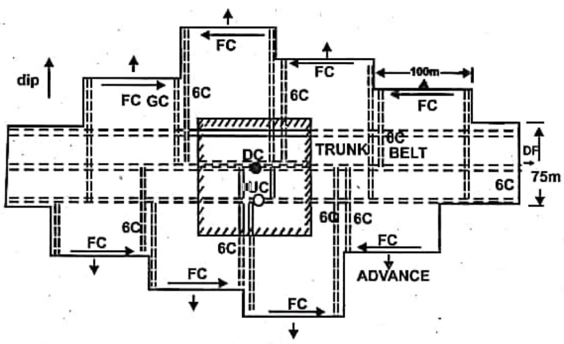


Fig. 13.1 Layout of single unit and double unit faces

FC - Face conveyor : SC - Scraper chain conveyor
GC - Gate conveyor : DF - Double unit face

It has the following advantages over pillar and stall methods ;

1. It is simple and offers concentration of work areas, being capable of giving the maximum yield per hectare of coal seam area. Concentration of work permits good supervision.
2. All the seam section is extracted in one operation, enabling the maximum extraction percentage.
3. Ventilation of working is rendered easier with simple and direct air routes.
4. The roof weight acting on the face assists in loosening the coal, yielding the greatest proportion of large coal. Cleats and slips in coal can be advantageously used to make easier winning and with proper attention to strata control techniques, friable and weak coal can also be won without much difficult.
5. Dirt bands in the seam can find useful purpose in packing of the goaf.

6. Floors liable to creep can be better controlled in this system as there are only a few roadways to be cared for.
7. Seams liable to spontaneous combustion can be satisfactorily worked by this system as conditions exist whereby all the coal which is a potential material of heating, is removed.

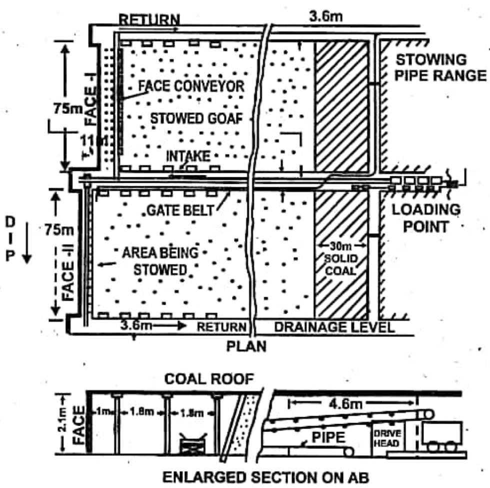


Fig. 13.2 Longwall advancing with stowing on double unit face.
Length of each longwall face 80-100 m.

8. It lends to mechanisation with the least capital cost per te of annual output. Where longwall advancing is adopted, it makes possible a quicker return on investment by enabling the mine to attain its optimum output within a short time.
9. It provides the most successful method for working beneath another worked-out seam.

The main **disadvantages** of the system are :

1. In longwall advancing method roadways are to be maintained in worked-out areas and entail substantial recurring maintenance costs. Sometimes there is convergence of roof in the roadways which are required to stand for long.

2. In the case of longwall advancing with caving a large expanse of goaf left behind constitutes a vast reservoir of firedamp, a potential source of danger.
3. In the case of longwall advancing with strip packing, if roadway packs are not well maintained against leakage, ventilation current may short-circuit through the goaf, which is both wasteful and dangerous (as it may cause incipient heating of small coal left in the goaf).

The longwall retreating system combines the advantages of pillar and stall method and longwall method. The roadways formed in the initial stages are supported by solid coal pillars and the property explored during the formation of roads - an advantage with pillar and stall working. When the retreating longwall working commences the face offers the additional advantages of longwall method. Moreover spontaneous heating, if any, that may occur in the goaf can be easily isolated. Ventilation planning is easier, leakages being very few. These combined advantages make longwall retreating a popular choice if longwall has to be adopted in a mine.

Application of the method therefore lies in working

1. thin seam (as thin as 0.7 m).
2. seams with dirt bands.
3. seams with tough roof which can be induced to bend gradually and settle on packs, or seams with weak friable roof which may cave in the goaf.
4. contiguous seams, if solid packed faces are laid out causing little disturbance,
5. gassy seams, which require meticulous planning of ventilation,
6. seams which are to be mechanised for large planned outputs.

As longwall method with caving requires a breaker row of props of high and uniform mechanical strength, the length of mechanical (friction) or hydraulic props available, limits the height of lift or section in longwall method. A high prop is unwieldy to handle, has bending tendency when under heavy pressure, and when the roof caves in, the prop is likely to be dragged in the goaf.

In longwall advancing the gate roads extend slightly beyond the face and the extended portion of the gate is called stable. Stables have to be provided where belt conveyor or rope haulage is used in the gate. Some types of machines used for cutting or cutting-and loading at the face require stables for their proper utilisation or reversal. In longwall advancing stables are usually 8 m to 10 m ahead of the face. They enable the area ahead of the face to be explored in time for taking measures to deal with any irregularity without serious disruption of production. If stables are kept much ahead of the face their ventilation poses a problem. Formation of stables is obviously not necessary on a longwall retreating face.

Length Of A Longwall Face

The optimum length of a face, whether advancing or retreating, depends upon the following consideration :

1. Output desirable : Where the operations at longwall face follow a rigid cycle of activities, the coal output is available only in 1 shift of the 3 shifts of one day. For continuous flow of coal in 24 hours, it is essential to have 3 or more longwall faces. This decides the output to be achieved per face which depends upon the working section of the seam, length of the face and its advance per day.

2. Capacity of coal cutting and loading machines : Length of face conveyors and capacity of gate conveyors to transport the coal.

3. Ventilation at the face : The methane gas emission has to be within such limits that CH_4 and CO_2 in the return air should not exceed 1.25% and 0.5% respectively according to Coal Mines Regulations. The quantity of air that has to pass the coal face is influenced by this factor. Large quantity means high velocity of air current and this causes discomfort to the workers. The quantity of air is therefore related to gas emission which, in turn, depends upon coal production in a given period.

4. Capital available for face mechanisation : Long faces require large number of props, with accompanying link bars, chocks, pushing and retracting equipment, water infusion equipment (for hard coal), more coal drills, long lengths of face conveyors with higher H.P. (sometimes 2 motors, one at either end), etc.

5. *Geological disturbances* : If the area is not well proved for geological disturbances small length faces are desirable.

6. Rate of stowing : The unpacked goaf edge should not lag much behind the face. The coal output and face advance is therefore linked with the rate of stowing.

7. *Development work* : In thin seams, floor or roof stone has to be cut in gate roads to make sufficient height for workers to pass. This is an unproductive work which can be kept to the minimum by providing long faces for a given output.

The length of a longwall face varies from 60 m to 200 m. The maximum face length is restricted in some countries by mining Regulations; e.g. in Germany the maximum length is 250 m.

Direction Of Face

On a hydraulically stowed longwall face employed power loaders like coal ploughs, a dip-rise face advancing towards the strike is preferable but for manual shovelling, side dump loaders and scrapers, a face laid along the strike and advancing towards the rise is desirable. If hydraulic sand stowing of the goaf is adopted and the face is along the strike it should be slightly oriented at 1 in 100 to provide for drainage of stowing water. A face along the strike is preferable for stowing in seams dipping at more than 1 in 5 for the following reasons :

(i) The pressure of sand on barricades is much less. Hence, their design could be simplified and breakdowns minimised.

(ii) On a double-unit longwall face, the face conveyors are generally identical, mainly for interchangeability of components. If the faces are along the strike the conveying capacity of both face conveyors is equal and hence both faces advance systematically are the same rate. On the other hand on a dip-rise face, the conveying capacity at dip face where coal has to be conveyed uphill is low and this restricts the general progress of the face.

Cyclic Longwall

On a conventional longwall face equipped with coal cutting machines, face conveyors (belt or scraper chain type), the operations at the face follow a definite sequence and a cycle of operations covers a period of 24 hrs. A straight line of face is essential for installation of

belt or scraper chain conveyors. Where sand stowing is adopted for stowing the goaf, the maximum distance between the packed goaf edge and the face is restricted generally to 6 m for proper roof control. The cycle of operations is usually as follows on a stowed face if coal cutting machine and manual loading of coal on the face conveyor are adopted.

A shift-Coal cutting and drilling

B Shift-Shotfiring, dressing down roof and sides, and roof supporting by props and chocks.

C shift-Coal loading and erecting extra supports at local places of bad roof after coal removal.

D shift-Same as A shift.

E shift-Same as B shift.

Where stowing of goaf is not practiced and the roof is allowed to cave in, it should break regularly along a line parallel to the face but without burying the props, conveyor or machine near the face. Such line along which the roof should break must provide strong support to the roof by chocks and strong props. Systematic timbering rules approved by the D.G.M.S. are applicable to all longwall faces.

Non-Cyclic Longwall

It may be noted that in a cyclic longwall system, each phase of operations like cutting, blasting, loading, etc. should be completed in the shift allotted for the work. This is unavoidable for smooth operation of the working cycle. Coal is available however only in one shift in a day of three shifts.

The supports and conveyor are installed in straight lines parallel to the face. Such straight lines are marked on the roof by stretching a string covered with chalk powder. Development of flexible chain conveyors which ar capable of snaking i.e. operating even when all its pans are not in a straight line, has enabled operations at longwall faces to be conducted in a noncyclic manner. Such conveyors are also known as phython conveyors. A flexible chain conveyor needs no dismantling and it can be always kept close to the face, which need not be in a straight line, by hydraulically or pneumatically operated pusher rams placed every 3-4 pans apart along the entire length of the conveyor. By installing such conveyors mechanical power loaders can be employed at the face for loading the coal direct into the conveyor. Since a zigzag or stepped face can be worked when flexible chain conveyors are installed, the coal cutting and loading operations can be carried out in all the three shifts without the need to follow a rigid cycle of face operations. The robustly built conveyors in this group are known as armoured chain conveyors and on them can be mounted coal cutting machines, coal ploughs and shearer loaders. When such combinations are used at the longwall face, props are not erected between the face and the conveyor, thereby providing what is called a prop-free face. Roof support at the face is provided by bars extending from props erected behind the conveyor, i.e. goaf side, or by hydraulic shields or hydraulic chocks.

Longwall Working With Coal Ploughs

The coal pluough is employed on a non-cyclic longwall face with a prop-free front. A coal plough is a machine which is mounted on an armoured chain conveyor and cuts a slice of coal, 100 mm to 200 mm from the entire working height of the seam during its travel along the face. The cut coal is loaded on the conveyor by a ramp which is a built-in part of the plough and which follows the cutting teeth. Separate arrangement for loading is therefore not necessary. Coal ploughs are used on a large scale in Germany, Holland and other European countries. The seam thickness suitable for its operation is from 0.6 m to 2 m. The plough consists of 4 or more teeth in two nearly vertical planes fixed to a base plate which is mounted on the armoured chain conveyor and driven by the motor of the latter through a chain. The motor is usually chain pulls the plough up or down the coal face and it is threaded through a 115 mm dia. tube attached to the conveyor all along the face. The two ends of the chain pass over the two sprockets, one at each end of the conveyor, and are finally attached to the plough. The plough cuts the thin web of coal during travel in either direction. The conveyor and the plough are held upto the face by hydraulic jacks placed at intervals along the goaf side of the face conveyor Fig. 13.3.

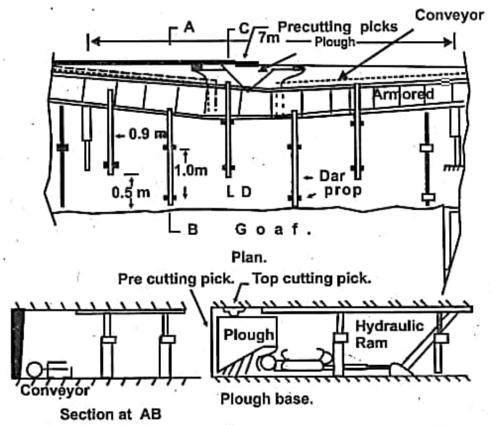


Fig. 13.3 A coal plough at a longwall face

Where the coal is hard, water infusion of the coal face precedes the ploughing operation. Water infusion is a process of drilling 1.2 to 1.5 m deep holes spaced 3 to 6 m apart along the face at 45° to it and forcing water at high pressure (6 to 10 kg/cm^2) through them. A water infused face not only renders ploughing easier but it also helps effective coal dust suppression.

The plough ability of a coal seam is determined in Britain by the impact strength index (I.S.I.) which is a modification of a test devised in Russia by Protodyakonov. The penetrometer developed in Britain, however, offers a better assessment of the ploughability of coal. The penetrometer consists essentially of solid rod 13 mm dia. (usually called an indenter) and a hydraulic ram which forces the indenter horizontally into the coal, the whole being supported by a special portable staking system. The resistance to penetration of the coal is recorded on a gauge and readings are taken at every 6 mm of penetration up to a depth, if possible, of 150 mm.

In addition to such information the ease with which the worked section of coal parts from the roof and floor has to be ascertained before taking a decision to installed a plough.

Most of the European coal mines adopting non-cyclic longwall have switched over to shearer loaders for cutting and loading coal on a prop-free front face. In India too, the new mines planning for large outputs by longwall mining utilise shearers at the face for cutting/ loading and the DERD has proved by far the most popular machine.

The shearer loader, popularly, called shearer in short, is basically a normal coalcutter with the chain and jib replaced by a horizontal drum laced with picks. It is mounted on a skid plate provided with bearing pads which rest on an armoured face conveyor (AFC). The machine consists of three units : (i) the cutting unit or gearhead (ii) the haulage and control unit, and (iii) the motor unit.

The cutting unit : It housed a special gearbox which drives a horizontal shaft projecting towards the face. Shearing drums or shearing discs, laced with cutter picks are mounted on the extension of the shaft. The shearing drum can be raised or lowered in a vertical plane with the help of a boom or ranging arm for cutting at various heights and is therefore known as ranging drum. There may be just one ranging drum at one end on the machine which is then known as single ended ranging drum shearer, SERD shearer, or there may be two ranging drums, one at each end of the machine, which is then called double ended ranging drum shearer, DERD shearer. The arrangement of one drum at each end in a DERD shearer permits coal cutting as well as loading on the conveyor when the machine is travelling in either direction. Such bi-directional machine is specially advantageous where the nature of roof is such that speedy erection of roof supports is essential. The DERD shearer has its gearhead of the same type as the SERD shearer, permitting interchangeability.

In the case of the SERD for cutting seams thicker than 1.5 m, the machine with the drum raised cuts the top section first from main gate to tail gate. At the tail gate, the drum is lowered to cut the coal near the floor and the lower section is cut from the tail gate to main gate. The cut coal is loaded by the cowl/dozer plate and whatever coal still remains on the floor is lifted by the ram plates.

In the case of the DERD the machine normally operates with the leading drum raised to cut the top portion of the seam while the trailing drum takes the bottom section of the seam down to the floor level in the same travel. The diameter of the drum is usually some 70% of the total extracted height. As the DERD shearer extracts the seam in single-pass operation, in either direction, it allows the early erection of roof supports. A DERD shearer can extract seam of upto 3308 mm thickness. The width of the drum varies from 500 mm to 700 mm and the shearer cuts from the coal face a slice of thickness 450 to 650 mm. A plough is attached to the front end of the shearer through an articulated joint and the cut coal is deflected by the plough on to the AFC. The shearer cuts the coal, loads it on the AFC and simultaneously travels on the AFC. The operation of shearing starts from one end of the longwall face. The dust suppression is by water sprays on the cutter picks.

Haulage & control unit : The travel of the shearer is effected with the help of a chain haulage through mechanical controls or with hydraulic system. In the hydraulic system operation by hydraulic controls provides infinitely variable speed of travel from 0 to 0.12 m/s and the travel speed is governed throughout the range by the load on the electric motor. In the haulage system which is mechanically controlled only multi-fixed haulage speeds are possible.

Some machines are equipped with radio control. This comprises a transmitter carried by the shearer operator, an aerial and receiver housed within the machine in a separate chamber actuating electro hydraulic valves on all machine functions. An electric push button control system is fitted to supplement the radio control. With radio control the operator can stand at a distance of upto 10 m from the machine and operate it.

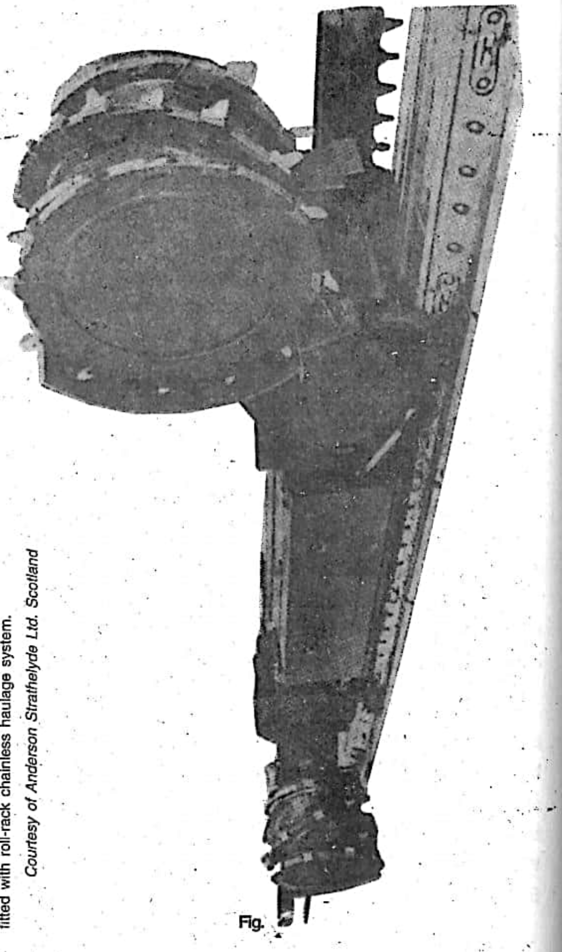


Fig.

Double ended ranging drum shearer, model AM500, mounted on a heavy duty armoured face conveyor and fitted with roll-rack chainless haulage system.

Courtesy of Anderson Strathelyde Ltd. Scotland

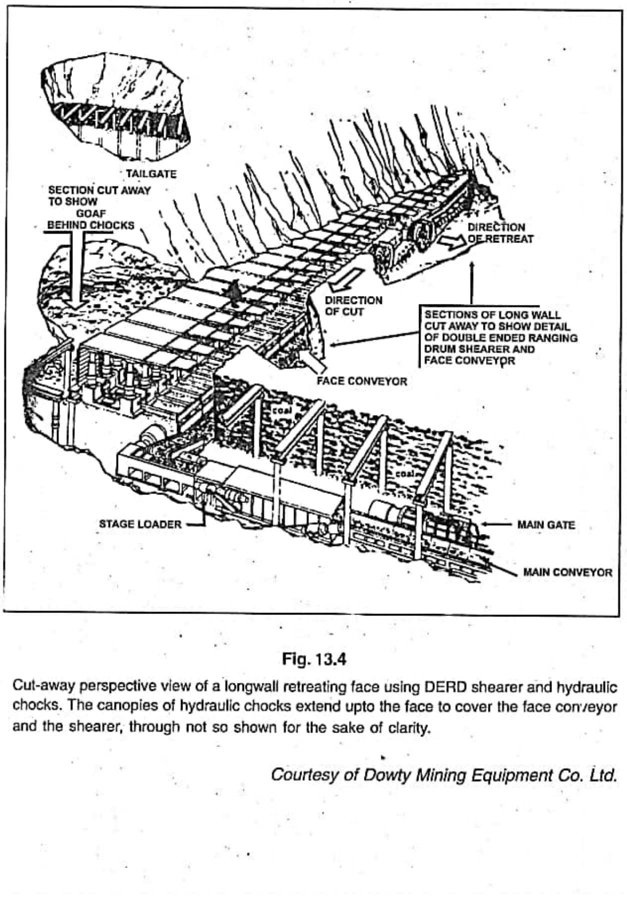


Fig. 13.4

Cut-away perspective view of a longwall retreating face using DERD shearer and hydraulic chocks. The canopies of hydraulic chocks extend upto the face to cover the face conveyor and the shearer, through not so shown for the sake of clarity.

Courtesy of Dowty Mining Equipment Co. Ltd.

The motor unit : It provides power from 225 kW upto 1000 kW with the help of a single electric motor or 2 motors at a maximum voltage of 4.2 kV at 50 cylces, 3-ph. A.C.

M/s. Anderson Strathclyde Ltd. Scotland, manufactures shearers of AB 16 series for seams of over 915 mm thickness and the AM 500 series for seams of over 1270 mm thickness. Shearers of both series are available with haulage arrangement of chain type or roll rack type.

Face advance by the shearer may be 2.4 to 3 m per shift and the production may be 1,000 te per day, depending mainly upon the type of the machine and the seam thickness.

Fig. 13.5 shows the layout, once adopted at DERD longwall face at Pathekhera colliery (WCL). Seam thickness was 1.3 - 1.7 m and the depth nearly 100 m. The method was longwall retreating with caving. Face length was nearly 110 m. The machinery deployed was as follows (Main items) :

Support system 6 leg. 240 te chocks.

DERD of 200 kW power

Armoured flexible conveyor of 2 × 90 kW.

Stage loader of 90 kW motor

Gate belt conveyor, 1000 mm width, 90 kW motor.

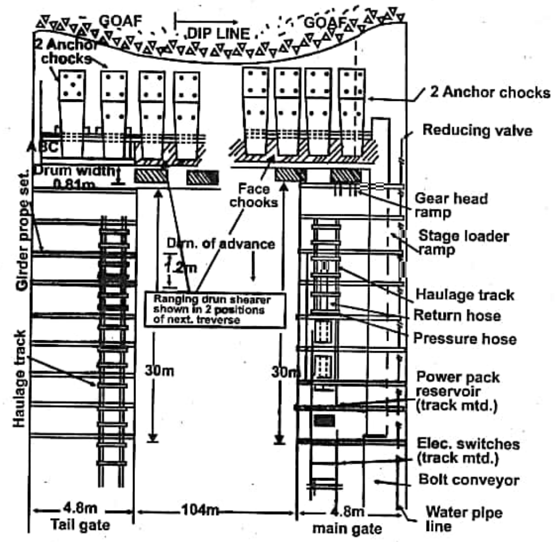


Fig. 13.5 Layout of a longwall rëtreating face with caving at Pathekera

This layout may be considered typical of the layout on any longwall face retrating with caving and employing DERD/SERD shearers, AFC's, powered supports, gate belt conveyors, stage loaders and other matching ancillary equipments.

Longwall Retreating With Stand Stowing

Fig. 13.6 depicts a typical layout of longwall retreating face with caving and a straight (non-snaking) conveyor. The longwall retreating with hydraulic stowing on a double unit face, as was practiced at one colliery, is illustrated in Fig. 13.7 with slight modifications.

The rise face AB is so advanced during retreat as to be somewhat outbye the face CD. This prevents stowing water of face AB from flowing to the face CD. Pairs of roadways, P_1, P_2 P_3 are driven to the panel barrier and the cross connecting dip galleries in each pair are on the apparent dip to have obtuse angle bends on stowing pipe range at the junctions. Each face is served by two intake airways and one roadway of the pair P_3 will serve as return when a retreating face is opened to the dip of P_3. The cycle of operations adopted at the face is the same as the cycle described for longwall advancing using coal cutting machine and chain conveyor at the face in conjunction with manual loading of the drilled and blasted coal at sand stowed face.

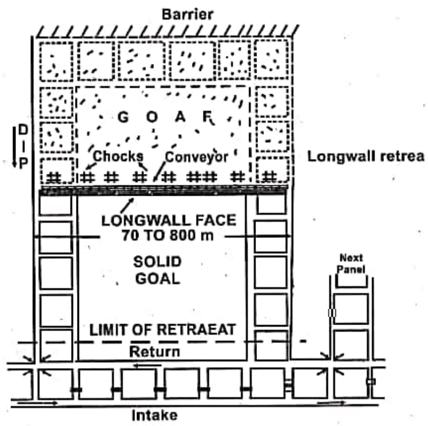


Fig. 13.6 Longwall retreating face with caving

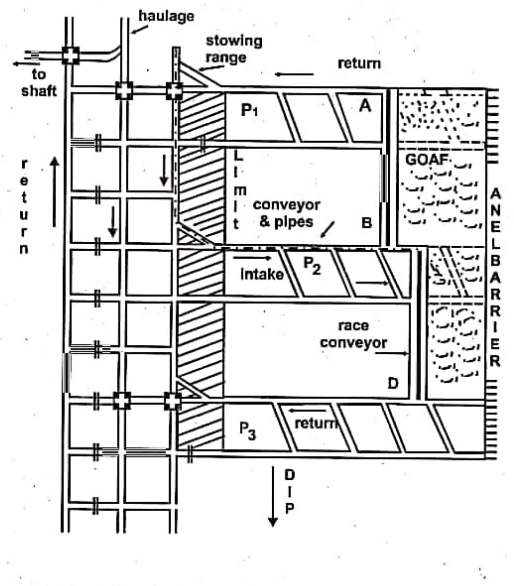


Fig. 13.7 Longwall retreating with hydraulic stowing

Barry face

A modified method of longwall retreating with sand stowing in a seam already developed on bord and pillar method of mining was tried successfully at Sodepur, Sitalpur, Parbelia and other collieries. The pillars were 35-45 m centre to centre. Three pillars along dip rise in a straight line used to provide a longwall face for coal extraction. Between the sand-stowed goaf and the pillars a single haulage track was laid for coal tubs on the floor of the coal seam. The haulage track was kept close to the face practically within shovellable distance for the manual loaders. (Fig. 13.8) As this necessitated frequent shifting the tract consisted of 2 m long rails welded to channels. No timber sleepers were used. The tubs were supplied at the face by a main and tail rope haulage and the set of tubs was held by the rope as their loading continued. A long cut at all the pillars in the manner of a longwall face was given, followed by drilling and blasting. The O.M.S. of the manual loaders was 4-5 tubs of 1.1 m^3 capacity.

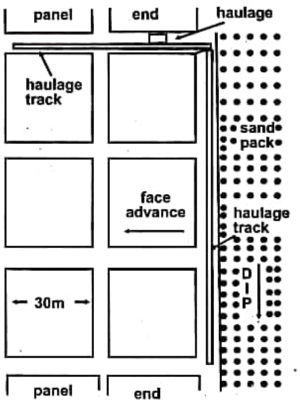


Fig. 13.8 Barry face

Longwall working in India

Longwall with sand stowing on retreating faces was practiced in Perbelia and Ranipur collieries of Raniganj coalfied in the 1940's. Coal cutting machines suitable for longwall and electric coal drills were in use. Wooden props at the face and wooden chocks along the gate roads was the method of support. The blasted coal was loaded by manual loaders into shaker conveyors (now obsolete) which discharged into coal tubs in the gate road. In 1950's the state of longwall mining was nearly the same. The decade 1960-1970 witnessed extension of

longwall working to some mines in the Jharia and Raniganj coalfields. IN 1963 Anderton shearer with TCR props were introduced on newly opened longwall faces in Chinakuri colliery, where the system worked in conjunction with sandstowing. In the same decade longwall faces were opened at Methani, Bhanora, Jamura 6 & 7 pits, South Balihari, Rana and Jitpur collieries. With the collaboration of French mining engineers, NCDC conducted trials of sublevel caving with longwall faces under artificial roof of wire netting at Gidi A colliery. Multi-slice longwall faces with sandstowing in thick seams at Sudamdih colliery (NCDC) and longwall with standstowing at Jitpur (IISCO) were other noteworthy developments during the decade. The longwall system at Bhanora with TCR props and at Jamuria 6 & 7 pits with Dowty 40-te hydraulic props was on retreating faces with caving in the midsixties and these were the first bold experiments of longwall with caving in the country.

On longwall faces the notable strata control failures were at Rana seam in Rana colliery, Rana seam in Bhanora colliery, Taltor seam in Jamuria colliery, GDK-9 incline (at 3-T panel of seam 3) and Churcha colliery. Some of these strata control failures resulted in abandonment of the face along with all the face equipment. At Borachuk seam of Dhemonain colliery partial strata control failure which was observed at 360-te powered support longwall face was mainly contributed by inadequate support resistance and the problem vanished when higher support resistance provided by higher support capacity was used at longwall face. The support resistance was raised to 45 te/m^2. Use of 550 te support props gave satisfactory roof conditions at the face.

Moonidih colliery has the distinction of being the first Indian coal mine to introduce shearers and powered hydraulic roof supports on caved longwall retreating faces in 1978. It started a training institute for longwall face workers about the same time. Pathekhera (1982), GDK-7 mine of SCCL (1983), VK-7 mine of SCCL (1985) Churcha (SECL) were some of the few mines that adopted longwall with caving and powered supports after 1982.

Table 1

Details of Face Equipment & Powered Supports at Longwall Faces of CIL/SCCL Between 1982 and 1985 (Alongwith Geomining Parameters)

	Particulars	Moonidih Kohex	Dhemomain Dowty	Seetalpur GDI	Pathakhera GDI	DOWTY	GDK-7 SSCL DDI
1.	Name of the seam	XVII Top	XVII Top	Borachak	Hatnal	Upper Workable	No. III seam
2.	Av, gradient of the seam	1 in 7	1 in 7	1 in 7	1 in 4.8	1 in 8	
3.	Face length	132	125	150	120	111	106
4.	Av. seam thicknes/height of extraction (m)	1.8/1.8	1.8/1.8	3.1/3.0	1.8/1.8	1.5/1.5	10.0/3.0
5.	Av. Depth from surface m.	350-400	350-450	300	350	105	80-110
6.	Type of powered supports	Chock OK-TR	Chock	Chock shield	Chock	Chock	Chock shield
		6 × 240 te	4 × 280 te	4360 te	6 × 240 te	6 × 240 te	4 × 325 te
7.	Type of AFC	2 × 55 kW SAMSON,	2 × 90 kW MP NCB 190	2 × 90 kW NCB 190	2 × 90 kW NCB 190	2 × 90 kW NCB 190	2 × 112 kW NCB 222
8.	Type of stage loader	55 KW	90 KW	90 KW	90 KW	90 KW	112 KW
9.	Gate Belt	800 mm	GWARBK 1000 mm (POLISH)	ASL 1000 m 90 KW	VISWA Eng. 1000 mm	1000 mm,ASL	1000 mm, ASL

To open a longwall face with caving, employing DERD or SERD shearers, AFC's, powered supports (hydraulic), conveyors,\stage loaders and other ancillary equipment upto the trunk belt conveyor, the capital investment is nearly Rs. 80 crores for a face of about 150 m length. With the devaluation of the rupee in 1991, the foreign exchange component of this heavy investment comes to nearly 50%. The large amount of interest on such capital investment and the fast rate of depreciation of the machinery working under very ardugous conditions underground, form a sizeable chunk of the production cost per tonne of coal. It therefore becomes imperative to open longwall faces with caving only in mines of good quality coal, planned for high production faces. At Pathekhera colliery of WCL, to quote an example, the coal is non-cooking, grade F, which is to be sold to the nearby Sarani thermal power station under a long term agreement made about 15 years ago at a price much below the existing statutory selling price fixed by the Government of India. Moreover, the production per travel of the DERD was limited from the seam, which is 1.3 to 1.5 m thick and may be considered thin by Indian standard. Because of these factors, the coal produced from the longwall face proved to be highly uneconomic.

Equipment for the complete mechanisation at the face with DERD shearers, powered supports and transport by gate belt conveyors upto the trunk conveyors constitute "full package" as one of the suppliers, MAMC, puts it.

As the mechanised longwall faces with DERD shearers, powered hydraulic supports and other matching equipment for a full package are highly capital intensive, the coal companies set their thoughts to medium level mechanisation on longwall faces. Two types of mechanised faces are now receiving attention.

1. ***Combination one :*** Arcwall coal cutter, side discharge loader and light chain conveyors at the face; friction props or 40-te hydraulic props; no powered supports.
2. ***Combination two :*** SERD or DERD shearers, AFC and 40-te hydraulic props; no powered supports.

Some of the mines in ECL are being planned for medium level mechanisation of longwall faces in 1992 and future years.

In the light of experience gained during mechanisation of longwall faces and the production available during the last 10 years, some recommendations were made at the **National Seminar on Mechanised Longwall Working** held in Calcutta in January 1991. These recommendations are reproduced below.

Recommendations :

Realising that the past trials of longwall mining in Indian Coal Industry has established the applicability of the system from technical and organisational point of view, the participants of the National Seminar on Mechanised Longwall Working made the following recommendations :

1. Increased efforts should be made to obtain adequate geo-mining informations of the panels where longwall workings are proposed in the future; in particular, seam seismic study should be introduced wherever possible so that the geological disturbances are known beforehand and the density of the boreholes should be suitably increased.
2. Powered support longwall faces being capital intensive, should be restricted only to areas with proven good geo-mining conditions and having high quality coal so that adequate return may be obtained.
3. Infrastructural facilities such as adequate power supply, maintenance, overhauling facilities, etc. are essential for the successes of powered support longwall faces. While planning mechanised longwall faces in different coalfields, the possibility of establishing regional workshops with all facilities for complete overhauling should be kept in mind.
4. Realising that in some of the present longwall applications the development work has not kept pace with the planned requirements, more emphasis should be paid on development work.
5. As the operations of the gate road are important for smooth functioning of mechanised faces, provision for adequate supervision should be made.

6. Transfer of equipment after completion of the panel should be planned in detail well in advance. The time requirement for shifting of the equipment to the next panel should be reduced and proper monitoring of face transfer operations should be carried out. For this purpose the replacement panel as well as the equipment needed for shifting and other infrastructures should be pre-arranged.
7. The process of selection of supports should be linked with the geo-mining conditions. Along with mine planners the equipment manufacturers should also be supplied with the geomining data so that the process of support design is improved.
8. Proper specifications of the various equipments for longwall package should be taken care of in the planning stage itself by the planners as well as by the suppliers.
9. Standardisation of the equipment should be tried right now to reduce different varieties at different places. An expert committee should be formed with representatives of users, manufacturers, planners, Directorate of Mines Safety, research and educational institutions to look into this aspect.
10. Man-riding system should be introduced more and more to reduce the travelling time and fatigue.
11. Free steered vehicles and other material handling equipment should be introduced wherever possible to reduce the transporting time of heavy equipment in the underground. Proper tools and tackles should be also included.
12. To prevent idling of the machines due to non-availability of spares manufacturers, both in public and private sectors, should be encouraged for the manufacturing of new equipment and spares indigenously. Regular orders to indigenous manufactures should be placed so that they can plan in time. Foreign suppliers should provide necessary details about spares and their indigenisation.

13. Import duty on spares for mining equipment should be reduced till these are developed indigenously.

14. The practice of going for high setting load of the supports should be examined in the Indian context by a committee of experts.

15. The possibility of introducing advancing, caved longwall faces and short single entry longwall faces should be examined by a committee and may be reviewed with some field trials in suitable locations.

16. As the applicability of a few longwall faces would be mostly for working the deep seated deposits, the climatic problems would become severe. R & D work related to estimation of heat, load and the development of cooling techniques should be taken up.

17. Some of the high capacity longwall faces in deep mines may need large amount of air flow for which special types of ventilation system such as Y and W type systems may be necessary. R & D work related to introduction of such new ventilation systems should also be taken up.

18. The development and design of shearer drums suitable for Indian coal seams should be taken up as early as possible as the drum design on the foreign norms has not proved to be very successful.

19. The success of longwall mining would depend on the availability of adequately trained teams. For this purpose, the industry, the equipment manufacturers as well as educational institutions should pay better attention.

20. Quality circle concept (including inviting suggestion from the workers) should be tried out at longwall faces.

21. Condition monitoring of face equipment should be done on a regular basis.

Table

Main parameters and specifications of equipments for longwall face package supplid/being supplied by MAMC to some mines in 1987 and later

Description	Muulidih	Moonidih XVIII	N. Amlabad	JK Nagar	GDK - 11A
A. Installation data					
1. Longwall face length, (m)	150	150	124	150	150
2. Seam thickness (m)	1.8 - 2.7	1.4 - 1.9	2.2	2 - 3.5	3 - 6.5
3. Seam gradient	8 - 12°	7 - 9°	1 in 6.5	1 in 20	1 in 6 - 8
4. Roof starta	Shale	Sandstone and shale	Sandstone and shale	Shally sandstone	Pyrite sandstone
5. Detph of seam (m)	150 - 300	280 - 500	350	150 - 200	150 - 200
6. Face advance/cycle, (m)	0.63	0.75	0.63	0.70	0.85
7. Support centre (m)	1.5	1.5	1.0	1.5	1.5
8. Suport resistance at yield (te/m²) before cut at	60.57	58.66	18.16	77	60.22
working height, (mm)	2500	2400	2300	2800	2500
9. Average bearing pressure at yield (te/m²) at roof	72	68.9	38.4	88.64	69.1
10. Supply voltage	1100	1100	1100	1100	1100 & 3300
11. Rated output from face (te/day)	N.A.	1820	670	3135	3375
B. Roof supports					
1. Type	Chock shield	Shield	O.C. hydro prop.	Chock shield	Chock shield
2. Capacity (te)	325	400	40	550	415
3. No. of legs	4	4	1	4	4
4. Hydraulic travel, (mm)	1300	1270	750	1800	1500
5. Closed ht. (mm)	1900	1130	1250	1800	1500

6. Leg	*DAST	DADT	SAST	DADT	DADT
i) Type					
ii) Setting load (te)	65.56	63.6 (R.L)	12.5	110.4	86.4
iii) Yield load, (te)	81.25	80 (R.L.)	40	137.5	103.8
7. Advancing ram, double acting					
i) Stroke, (mm)		800	900	760	800 910
ii) AFC push force (te)	8.7	12.6	5.15	15.2	20.96
iii) Support advance force (te)	18.7	27.1	1.75	28.4	44.9
8. Hydraulic distribution system	Unidir.	Ringmain	Unidir.	Ringmain	Ringmain.
9. Max. working pressure in leg circuit (bar)		397	392	–	354 400
10. Power pack					
i) Motor, (kW)		75	75	37	75 112
ii) Supply pressure from power pack, (bars)	138	200	130	210	333
11. Unit support wt. (te)	9	9.71	0.08	15.5	12
C. Shearer					
1. Model	AB 16 DERDS	AM 500 DERDS	AB 16 SERDS	AM 500 DERDS	AM 500 DERDS
2. Power rating (kW) Motor rpm - 1470	200	375	200	375	750

* DAST - Double acting single telescopic ; DADT - Double acting double telescopic;
DERDS - Double ended ranging drum shearer ; SERD - single ended ranging drum shearer.
N. Amlabad will have longwall retreating with sandstowing. At other mines the method of working is longwall retreating with caving.

Courtesy of MAMC, Durgapur.

Room And Pillar Mining

The method consists of splitting the mining area into districts by driving at intervals of 800 to 1500 m a set of 3 to 10 main entries (headings) driven parallel to one another to a predetermined boundary from the shaft or inclines. The entries are 3.8 to 5.4 m wide and 15-25 m centres. This large number of entries together with their cross-connections provide the necessary number of working places for the machines to be fully utilised right from the start. The areas so formed are then developed on panel or block system with the use of either the conventional mining machinery (*i.e.* for undercutting blasting and loading) or the continuous mining machinery. Fig. 13.9.

A typical layout of panel system is given below. From the main entries two or generally three "panel butt" or "room entries" are driven at right angles, 4 to 5 m wide and 12 to 27 m centres. These room entreis are taken off from main entries at intervals of approximately 90 m to work the room. The rooms or the working places are opened to the rise of room entries (and on both sides if the seam is flat). Rooms are 6 to 9 m wide and 90 m long driven at short distance from one another leaving only a small rib (say 3 m or so) between adjacent rooms. When partial extraction is the aim, width of pillars (or ribs) of coal left between the rooms is the least, consistent with the safe exist of the machines from the rooms. When full extraction is aimed at the width more to enable slices to be taken off the pillar as soon as it is formed by the machine on the retreat.

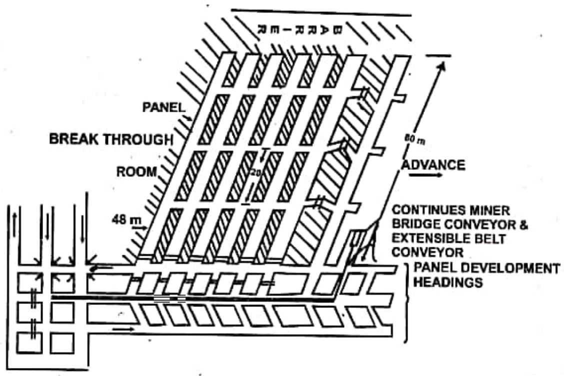


Fig. 13.9 Room and pillar mining layout

The room and pillar method of mining has all the advantages of pillar and stall method. Some of its disadvantages are :

1. The number of machines used is comparatively larger than that required in other systems. Machines are required to be moved very frequently. This requires better standard of maintenance and operation crew.
2. Amount of capital required per te of annual output is usually larger.
3. Percentage of recovery is usually much lower than in longwall.

The scope of application of this method may be confined to almost flat seams of 1.2 to 4 m thickness, comparatively free from geological disturbances and lying at depths below 300 m. As loss of coal during working is not insignificant, this method may be considered where reserves of coal are plentiful and in our country the method may be considered unsuitable for our cooking coal seams.

HORIZON MINING

Horizon mining is a system of mining, applicable to inclined or undulating seams and also to relatively flat seams where these occurs in groups whereby all the coal seams are extracted between predetermined horizons, levels, or planes. It involves driving main roadways horizontally (or almost so) through the measures or strata from the shaft at pre-arranged intervals of depth, and these road-ways form, as it were, the main arteries of the mine, through which coal is transported throughout the life of the mine, or of the horizon concerned. At least two levels are driven at different horizons; lower level, called the haulage level, is used for haulage and serves as intake airway and the upper level called the ventilation or return level, is used as return

airway and supply road. Connections are made to each of the seams lying between these two levels and the portion of each seam intersected by the levels is divided into sections of suitable size either by staple or blind shafts or, in rare cases, by inclined roads. (Fig. 13.10).

Laterial drifts or roads, or simply laterials, are those roads driven parallel to the strike from the shaft and they may be sited in one of the coal seams or, more usually, in the strata below the lowest coal seam in the horizon concerned (see Fig. 6.3, p. 171). The term cross measure drifts, or simply cross-cuts, is used for all the approximately level main roads driven in rock at right angles to the line of strike, i.e. in the direction of the full dip or rise of the strata. In general, the cross-cuts in the various horizons should be driven directly above one another. A network of these roadways, laterials and cross cuts driven at the same depth of horizon, constitutes a horizon or level. Vertical distance between horizons is 60-200 m.

Horizon mining is actually not a method of mining in the sense longwall or bord-and-pillar is, but is a method of lying out the workings and roadways in a coal seam and cross measure strata for speedy transport. The actual method of mining may be longwall, bord and pillar or room and pillar though the method that has been normally adopted has been longwall advancing or longwall retreating as the countries that have first tried horizon mining and later developed it, were accustomed to longwall methods of mining. The usual methods of transport are the conveyors on the faces and gates, spiral chutes in the staple shafts leading to haulage level, and locomotives in the haulage level. Seams are worked in descending order.

Layout For Horizon Mining

Fig. 13.10 illustrates the system in its simple form, having regard to single, uniformly inclined seam lying to the dip of the shafts.

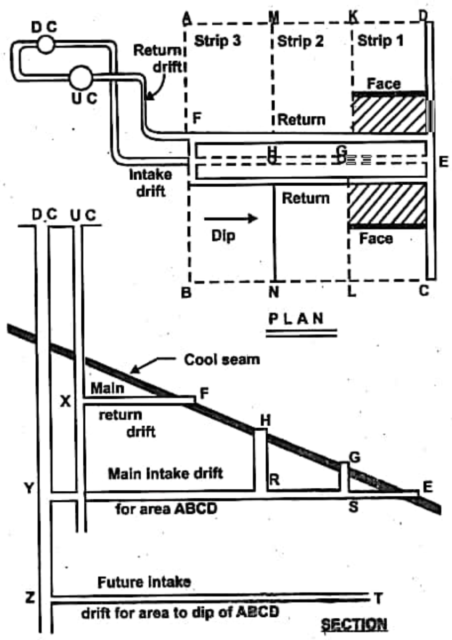


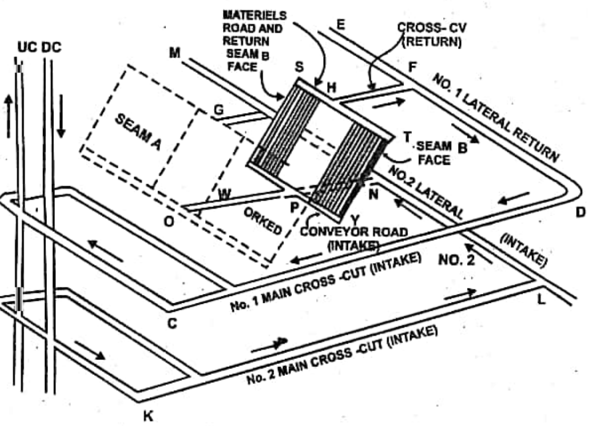
Fig. 13.10

The first step is to drive main cross-measure drifts. XF and YE respectively, from the shafts in the direction of the full dip of the strata to intercept the seam at points F and E. The first (upper) drift XF forms the main return airway, and the second (lower) drift YE forms the main intake airway for the first area of coal to be worked, namely ABCD on the plan. Ultimately, air will pass down the DC shaft, along drift YE,

then ascensionally up the seam from E to F, and so to the UC shaft via drift FX. The vertical interval between the two drifts varies in different cases according to the dip of the bends, the number of seams to be worked, and other factors, but may range between 60 and 200 metres. The next stem (in the particular case under consideration) is to drive lateral level roadways EC and ED in the seam (see plan) from point E, at right angles to the main drift YE. The roadway CED represents the main intake and conveyor level for the first pair of working faces, the loading point into tubs or mine cars being at E.

Thereafter, the seam may be opened up by one of the two methods.

1. The first method is to drive a main centre gate in the seam direct to the full rise from E, with two companion return airways which connect up with the main return drift at F. From these rise roadways other intermediate level gate roads GK, GL, HM, HN, FA, and FB (in succession) are turned away at right angles, so dividing the area ABCD into three steps, each 200 to 250 metres wide.



Horizon mining method for multi-seam working

Fig. 13.11

2. The second method is to drive the rising return airway and the intermediate level roads as before, but to dispense with the centres gate and drive vertical staple pits, RH and SG, upwards from the main intake drift YE to pierce the seam at, or near, points H and G.

In both methods, each strip of coal is worked in the usual way by longwall advancing along the strike of the seam, with the faces on the line of full dip and rise, the coal being conveyed down the faces to the loading gate at the lower end, namely to ED and EC for strip 1, GK and GL for strip 2, and HM and HN for strip 3. For strip 1, the coal is then conveyed along the bottom level to the loading point at E from which it is taken to the shaft by locomotive haulage.

In the first method, so far as strips 2 and 3 are concerned, the coal, after reaching G and H, is conveyed down the seam-confined centre gate to point E. In the second method, however, the coal, after reaching G and H, is delivered to a spiral conveyor situated in each of the staple pits and is loaded into tubs on the main intake drift at S and R.

It is generally considered better to adopt the staple pit arrangement which cuts through the strata and follows the shortest route from the seam to the main haulage road, thereby cutting out the centre rise gate which is subject to ground movement and is more costly to maintain, but local conditions will determine which is the better method in a particular case.

If stowing material is required at the faces, it is sent down the UC shaft and taken via the main return drift to point F whence it is transported by conveyors to the rise end of the faces.

When the area ABCD is exhausted, the lateral roadways ED and EC may be extended in both directions to open up further areas of coal, similar to ABCD, on either side between the same two horizons. The number of such panels of workings will obviously depend on the laterial distance to which the coal seam extends, or is to be worked.

When all the coal between the first two horizons has been extracted (or rather whilst extraction is still taking place) another main intake cross-measure drift ZT is set out from the DC shaft at a lower level, to intercept the seam at a point lying to the dip of E. The lower are of coal between the second and third horizons is then worked in a similar way to ABCD, drift ZT becoming the new main intake and drift YE the new main return for that area.

Later, a fourth horizon may be opened up at greater depths if the reserves of coal justify the cost of development.

In practice, of course, each main cross-measure drift normally pierces a number of workable coal seams which can all be extracted in succession and in descending order between each pair of horizons. This is illustrated in the Fig. 13.11 (for two seams A and B). The figure gives a more or less perspective view of two horizons (No. 1 being the upper horizon and No. 2 lower horizon). In this case CD and KL are the main cross-measure drifts from the shafts; DE and LM are the two corresponding laterals below the lowest seam; FG and NO are the cross-measure drifts driven backwards to reach the higher seam A and to intersect the lower seam B at H and P. Seam A has been extracted and seam B is being worked. There are two advancing longwall faces in seam B, namely XS and YT, each being on the line of full dip and rise. The level intake and loading gate in the seam is XY which is turned away in both directions from point P, whilst the return airway and materials road in the seam is ST at the top end of the faces and in No. 1 horizon.

The course of the ventilation from the DC shaft is along Nl. 2 main cross-cut and laterial, to point N; thence along the cross-cut NP where the air splits both ways and ascends along the faces to points S and T; and so to the UC shaft via the return cross-cuts and lateral in No. 1 horizon.

In case where the faces XS and YT are too long, intermediate level conveyor roads may be driven in the seam and the faces split up into shorter lengths.

The foregoing layout represents merely one method of development and extraction but there are other possible variations depending on the gradient of the seams, the incidence of faults, and other local factors. The general scheme of things to work a mine by horizon mining, may be understood from the description and the figures.

General Considerations Before Adopting Horizon Mining

1. Large capital expenditure on shaft sinking and drifting is required before production starts. Interest on capital and depreciation of machinery as well as civil works is therefore heavy.

To justify it the mine must have large reserves and production should be high over a long life. Production of at least 50,000 te per month from one pair of shafts may be considered the lower limit under Indian conditions and the life of the mine, not less than 30 years.

2. The property should be preferably virgin.
3. The reserves should be established by well organised prospecting and drilling programme.
4. The seam density i.e. the number and thickness of seams within the given vertical distance should be high.
5. The strata should be strong as each horizon requires shaft insert and long drifts that last for nearly 1/2 to 3/4th of the life of the mine.

Other factors which need to be considered before opening a mine apply in the case of horizon mining as well.

Advantages claimed for this system are :

(i) It provides the main road for efficient and adaptable haulage systems. A locomotive haulage, in its most efficient form capable of dealing with high outputs of the orders of 3,000 tonnes/day or more, is possible. Outputs are concentrated at a few loading and hoisting points, permitting mechanisation in transport and hoisting and achievement of large production rates.

(ii) It makes possible a highly efficient ventilation system, there being two separate independent roadways intake and return, without the possibility of any air losses and short circuit. Pressure difference between the intake and the return is easily maintained. There is a further advantage in deep workings, in that, fresh air is not heated before it reaches the faces to the same extent as in in-the-seam mining development. Also emission of gas through the strata is usually much less and the fresh air does not get vitiated through its passage along the airway, an aspect which is of importance when high outputs of the order of 10,000 tonnes per day are to be obtained.

(iii) Maintenance cost of roadways is the least throughout the life of the horizons, as the roads are in stone.

(iv) It is easy to work several seams at a time.

(v) It is eminently suited for inclined (beyond 10^0) and disturbed seams and for areas of high seam density.

(vi) These are stands explored geologically as the laterals and cross-cuts are driven in the initial days of mine life.

General considerations before adopting horizon mining, stated earlier, indicate the disadvantages of the the system. High capital expenditure for development work and long gestation period are the main disadvantages. Any seam which is not worked before abandoning an upper level and staple shafts is virtually lost, as against the in-the-seam mining, where any unworked weak can still be worked at any time in the future.

Level Mining Or Semi-Horizon Mining

This is a system of development of thick, steeply inclined seams on the bord and pillar pattern in such a manner that excepting the "main dip" all the roads in the seam are level. The galleries are so laid out and the parting between adjacent horizons is so planned as to ensure galleries of one horizon to be either above or below vertically those of adjacent horizon to be either above or below vertically those of adjacent horizon. The parting between the two horizons must be not be less than 3 m and the floor area of the pillar must not be less than that prescribed under the Coal Mines Regulations at the depth corresponding to the width of galleries. This method was adopted in the thick, steeply inclined Sirka and Argada seams in Karanpura field during development stage. The seams are nearly 22 m thick and the gradient approximately 1 in 2.

It is incorrect to call it horizon mining. The main advantage of this method is that all the galleries in a horizon are leveled. This adds to the efficiency of machinery and workers and permits of high capacity transport form the face. Each horizon constitutes a panel which can be readily isolated from the other horizon in case of emergency. A property with extensive area on the strike without stone bands and without faults, special dip faults, is ideal for level mining.

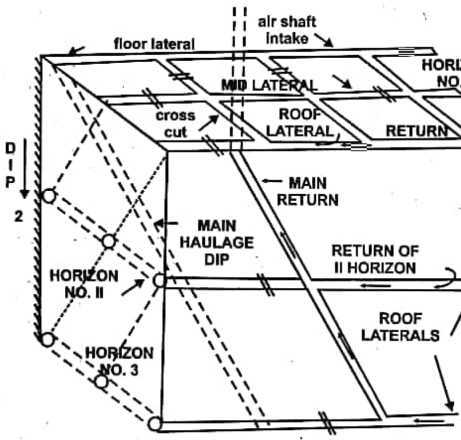


Fig. 13.12 Level mining

Fig. 13.12 illustrates the manner of developing a seam by level mining method of working. The terms used for roadways are some what different as compared to the terms used in bord and pillar or longwall method. The approach to the seam which dips at 1 in 2 is by a haulage incline along the apparent dip. A level roadway along the floor of the seam and extending upto the boundary of the mine is called floor lateral. A level roadway along the roof of the seam is called a roof lateral. A level roadway connecting these two laterials is called a crosscut. The crosscut may be at right angles to the laterals or at an angle forming rhombus pillars for transport by shuttle cars. If the seam is thick it may permit drivage of a mild lateral.

To ascertain whether the floor lateral and roof laterial are going along the floor and roof respectively, holes are drilled on the sides of the roadways by ordinary coal drills.

The ventilation system is simple. Vertically above the main haulage dip a gallery for return air is driven parallel to it in coal near the roof. The roof lateral of each horizon is joined to this main return airway which is connected to an upcase shaft.

The depillaring of seams developed by level mining in the absence of stowing posses problems and so far no suitable method has been devised for it. Sirka seam at Sirka colliery which has been fully developed by level mining is going to be extracted by heavy earth moving machinery upto the quarriable limit of 1:3 (and perhaps up to 1:5 in future) by quarrying but the pillars formed beyond the quarriable limit may be split after permission from the D.G.M.S. and then abandoned.

Sublevel Caving And Slicing Methods

These have been described in the chapter on "working of thick seams".

✧ QUESTIONS ✧

1. What are the circumstances when longwall method of mining can be advantageously adopted ?
2. Write short notes on :
   (a) *cyclic longwall* (b) *prop-free front*
   (c) *non cyclic longwall* (d) *barry face*
   (e) *double unit face.*
3. How is the ploughability of a coal face determined ? Under what conditions a coal plough can be used at a longwall face ?
4. Give a layout of a longwall face with a DERD shearer. What output can be reasonably achieved by a DERD shearer in a day ?
5. What is horizon mining ? State its advantages and disadvantages.
6. Give the cycle of operations at a double unit longwall face (retreating) using coal cutting machines and chain conveyor at the face in conjunction with hydraulic stowing. Stage the distances from the face of the conveyor, props and sand boxing.
7. What are the general considerations that govern the layout of a coal mine on horizon mining pattern ?

✧ ✧ ✧

Chapter -14

Thick Seam Working

In India coal seams over 4.5 m thick are considered as thick seams. This norm varies abroad from country to country; e.g. in Russia and China a seam over 3.5 m thick; in Germany 1.5 m thick; in France 4 m; in Japan 2.25 m thick; In order West European countries the figure is 2.5 m.

Seams thinner than 1.5 m present problems of manual tub loading, walking, installation and use of machines, etc. but thick seams present technical problems in complete coal extraction, roof control, dealing with spontaneous heating, etc. At Kunstoria colliery, to quote an example, the seam thickness is 8.4 m. during development by continuous miner 3 m thickness of coal near the floor was developed by bord and pillar. In the depillaring stage due to lack of stowing material only the 4.8 m seam thickness is extracted in one lift and the remaining coal is allowed to fall in the goaf is lost.

In underground coal mines with thick seams, the ventilation is sluggish, mechanisation is difficult as heavy unwidely supports are required. Bumps are common in thick seams. Air blasts pose a problem and the percentage of extraction is poor resulting in loss of large quantity of coal which remains underground and is a potential for risk.

A seam upto 3 m thickness can be extracted in one lift, slice or section by longwall advancing or longwall retreating. It can also be developed by bord and pillar followed by depillaring with caving or with stowing. The percentage of coal extraction varies from 80% to 90%, high percentage being possible with the adoption of stowing.

A seam over 3 m thickness and upto 6 m thickness is generally extracted in 2 lifts (or slices or sections, as they are sometimes called) through seams upto 6 m thickness have been extracted by developing

on bord and pillar pattern upto 3 m height and depillared to full height by caving in one lift. In Giridih colliery lower Karharbaree seam, 6 m thick with sand stone roof, dipping at nearly 1 in 10 and lying at a depth of nearly 200 m was developed by bord and pillar in the floor section taking a height of 2.4 m. During depillaring the general method of extraction followed in that seam and which is normally followed in seams of similar thickness is as follows. The pillars are split and the development galleries as well as split galleries are heightened from floor to roof by taking roof coal. Each stook is extracted in two benches as shown in fig. 14.1. No coal cutting machine is used though holes are drilled by electric drills. The top bench is kept 2 m in advance of the bottom bench. Miners are engaged on the same stook at the bottom bench and top bench attacking the coal at X^1 and X. The coal of top bench is dropped on the floor of bottom bench, as indicated by the arrow, where it is loaded manually into coal tubs.

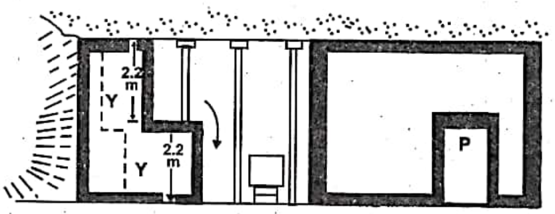


Fig. 14.1 Depillaring in two lifts; split gallary at P being heightended and being extracted

In Fig. 14.1 the split at P is being heightened. Y Y^1 represents the final stage. When this stage is reached the bottom lift Y^1 is scooped as much as safety permits so that Y Y^1 is nearly a vertical face, thus creating a rib of coal, 1.5 m to 1.8 m thick against the goaf. That completes the extraction of the stook. The roof supports are then withdrawn and the place fenced off. Ladders are provided for access between bottom lift and top lift.

If a coal seam of similar thickness is initially developed by bord and pillar in top section, increasing the height of development galleries and split galleries by taking our floor coal creates problems of drainage, ventilation, coal loading etc. though they can be generally overcome.

The percentage of coal extraction is some what low, 70% to 80%, as thicker chowkidars and ribs at the goaf sides have to be left insitute for safety. In seams of thickness 4.5 m and above, roof testing is not satisfactory, the roof support by props not reliable, and ventilation is poor. Normally the DGMS does not give permission for depillaring with caving in one life in seams thicker than 5.5 m.

Thick seams which cannot be extracted in one life or section are extracted in multisections by the following methods.

(A) Ascending order

Starting from the floor section and proceeding towards the roof. In this case the goaf is stowed pneumatically or hydraulically. This method is adopted in Indian mines with hydraulic sandstowing. The roof of the slice under extraction is coal except in the last slice which has the real of sandstone or shale.

(B) Descending order

Starting from the top section of coal and proceeding downwards to the floor. This manner of proceeding downwards provides three methods of extraction, all using artificial roofing of wire nets.

1. Inclined slices by caving with artificial roofing.
2. Horizontal slice by caving with artificial roofing.
3. Stowed slices extracted with artificial roofing.

In all the above methods no coal or parting is to be left between adjacent slices or sections. If the method of working or other considerations require some parting to be kept between adjacent sections or slices, such parting is thin, between 0.3 m and 1 m.

(C) Multi section working with thick coal partings and caving.

(A) Multi section working with stowing (ascending order)

For a reasonable percentage of extraction in a seam over 3 m thickness stowing is essential if subsidence of upper coal seam and the surface is not permissible. It is possible to extract a seam in 3 to 4 sections, each of 2.4 to 3 m thickness with stowing. Extraction of coal in more than 4 sections is however not safe; convegence and bed separation of roof rocks start with the extraction and stowing of first section itself and the bed separation reaches a such magnitude after extraction of 3rd or 4th section that the roof control becomes a problem. The coal extraction is 80% to 90% in 2 sections.

In Parbelia Colliery of Raniganj coalfield, Disergarh seam 4.5 m thick, dipping at 1 in 5 at a depth of 500 m was extracted in conjunction with stowing in the bottom section 2.4 m thick. The bottom section was extracted by longwall advancing, longwall retreating and also by carry faces in those areas where development was by bord and pillar (pillars 45 m centres to centre). The longwall faces and retreating barry faces were along dip rise. After the bottom section was extracted and stowed in a particular area, that area was allowed to settle for $1^{1/2}$-2 years. The upper section of 2.4 m was then developed by bord and pillar over the sand. Coal cutting machine was not used as free face for blasting was obtained by scooping out the sand. Timber props in upper section were erected over sand by providing wide lid at the foot of the prop. Pillars in the top section were extracted with stowing.

In Chinakuri Colliery (depth 600 m) Disergrh seam 3.6 m thick, dipping at 1 in 5, bottom 1.7 m coal was extracted by longwall advancing with fixed drum shearer machine in conjunction with hydraulic sand stowing for a few years. The longwall faces were 200 m long along dip rise. The top section coal, 2 m thick, is extracted by longwall in conjunction with stowing but the longwall faces of top section are parallel to strike as they are found more suitable than dip-rise faces when stowing over sand floor.

(B) Multi section working in descending order

Methods which are adopted for multisection working in descending order employ longwall (either advancing or retreating) method of coal extraction with artificial roofing. working under artificial roofing is a concept completely new in India and therefore, a detailed description of the method employed at gidi 'A' Colliery (Karanpura coalfield) will be of interest. The method that was adopted is called "inclined slicing with artificial roof". Collaboration of French Engineers was sought for experimenting the method in the early sixties and therefore the method was sometimes referred to as French method and is described later.

In the method known as "Horizontal slicing with caving" the general mine development is on horizon mining pattern. The method is suitable for steep thick seams of irregular thickness. Drivage of cross measure drifts and staple pits is essential and this increases the unproductive work in stone. The slices are horizontal, each slice extending from the floor of the seam to the roof and not parallel to roof and floor. While taking each slice fresh roof has to be brought down for some distance. The method is not yet tried in India.

Inclined slicing with stowing, with the slices extracted in descending order, is a method adopted in some mines in France and other European countries. For adoption of such method pneumatic stowing with crushed stone is a suitable arrangement as the size of crushed stone is convenient for retention over the wire netting of artificial roof. Hydraulic sand stowing has not been adopted in France for this method. In Hungary, however, hydraulic sand stowing has been successfully used for stowing slices extracted in a descending order. The artificial roofing in this case consists of wooden planks kept on artificial roof of wire netting (which is laid on the floor of the section under extraction).

French Method Of Working Thick Seams

At gidi 'A' colliery (Karanpura coalfield) were carried out with the help of French mining engineers for extraction of a thick coal seam without stowing. An important feature of the French method is that the entire thickness is extracted by caving in suitable slices starting from the roof and proceeding to the floor in a descending order. The natural roof of the top slice (first slice) is allowed to collapse and rest on iron wire netting which serves as artificial roof for extraction of subsequent slices till the bottom-most slice coal is extracted.

The broad feature of working a thick seam by French method are as follows :

The broad features of working a thick seam by French method are as follows :

(i) The thick seam can be extracted if it is virgin with nearly regular thickness and has a roof which caves regularly when the supports are withdrawn. The method is therefore well suited to roof of bedded sand stone and for a seam at 60 m depth or more. Seam gradient should not exceed 35^0 (1 in 1.4).

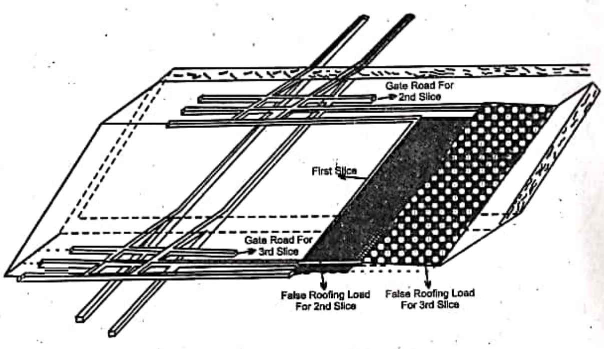


Fig. 14.2 Developing for inclined slicing. False roofing may be laid for 2nd and 3rd slice separately

(ii) The method employs longwall advancing or retreating manner of coal extraction.

(iii) In a thick seam, say 12 m thick, each slice may be 2.4 m thick so that in the inclined slicing method 5 slices are available.

The first is the top most slice near the roof. Operations of coal cutting, blasting, roof support, conveyor shifting, etc. are carried on in each slice. It is however possible to avoid such operations in one of the slices and skip over the slice by going over to the slice below. the slice which is thus skipped over is called "sub level". Coal of the sublevel is taken when it collapses after cabin of the slice below it. Sublevel caving is possible in "Inclined slicing method" as well as "horizontal slicing method".

(iv) Gate roads have to be formed in the virgin seam for access to longwall face in each slice and they serve the purpose of transport, ventilation, material suppliers, etc. (Fig. 14.2).

(v) Iron wire netting in the form of rolls and flat M.S. strips for strengthening the wire netting are used. Before installing any machinery, roof supports and commencing coal extraction of top slice, the wire netting is spread on the floor of top slice and suitably anchored in the gate roads. Chain conveyor and friction props are installed in their places over the wire netting.

(vi) Friction props or hydraulic props with cantilever steel bars have to be used for support as timber props and bars may not have sufficient strength for foot control in this method.

(vii) With the normal coal cutting, drilling, blasting, conveyor shifting, etc. the longwall face is advanced and the roof supports withdrawn to induce the roof to cave in. Extension pieces are added to the wire netting to keep it within 1 m of the longwall face all the time.

(viii) When the roof caves in, its broken rocks rest on the wire netting. This wire netting which is anchored in gate roads of top (i.e. first) slice serves as the roof of the 2nd slice when the latter is being extracted and mining operations during extraction of 2nd slice are carried on under this artificial roof of wire netting. Roof of 2nd slice is safe compared to that of first slice as there is no danger of roof collapse. In this way 3rd and lower slices are extracted.

The principle behind the French method of extraction in slices with artificial roofing is that when the roof caves after extraction of first slice and rests on top of the flexible wire netting the dynamic and upredictable nature of the roof is lost. In second and subsequent slices the roof on the wire netting consists of a dead weight and is thus much more reliable and easier to handle.

With this background it may be possible to understand the French technique as it was tried at Gidi 'A' colliery for the first time in this country. the seam is near the outcrop, 12 m thick, with a dip of 1 to 3 to 1 in 5 under 60 m cover. Immediate roof consists of 1.5 m shale overlain by bedded sand stone, and the immediate floor of the seam consists of shale. Access to the seam is by three inclines, two along the roof, 16 m apart, and one along the floor, the floor incline place centrally between the two roof inclines. One of the roof inclines is equipped with track and haulage and is used for supply of materials while the

other for the upcast ventilating fan. The main intake of air is through the floor incline which also serves for coal transport by belt conveyor. The area of virgin coal seam is divided into 4 panels and each panel coal is to be extracted by inclined slicing method commencing from the top slice. So far only two panels have been extracted and the remaining panels are wating for essential equipment like friction props, etc. but the DGMS may not permit working in the remaining panels without use of shields.

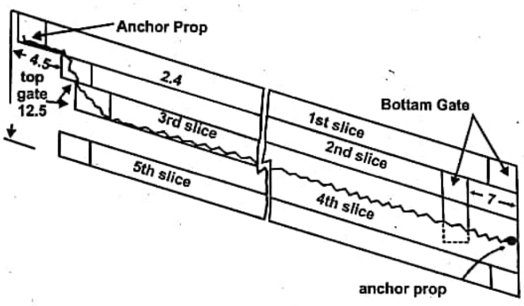


Fig. 14.3 Relative positions of inclined slices. The figure are in metres

Before starting underground work of coal extraction, the surface was covered with 20 m of quary overburden to stop breathing of air from cracks which may be caused by caving. The overburden also helped to increase the weight of roof and induced early roof break after the first slice was extracted.

Method Of Extraction Of 1st Slice

In the first panel having an area of 90 m x 150 m, the 12 m thick seam is to be extracted in five inclined slices, each 2.4 m thick and parallel to the roof and floor, in a descending order. The longwall face is 90 long dip-rise and a gate road (top gate) from the roof inclines is driven to serve that face. Armoured chain conveyor with a conveyor mounted coal cutting machine is used along with the usual coal drills. Friction props and steel cantilever bars are used for roof support. The face is worked by longwall retreating, completely caving the worked out area including the top and bottom gate roads. (Fig. 14.3).

Coal Getting

The coal is undercut to a depth of slightly more than 1.25 m which is the length of the cantilever roof bar by a conveyor mounted longwall coal cutter. Whatever floor coal is left after face blasting, is removed by drilling and blasting.

The conveyor has 50 H.P. motor with the drive head located in top gate and it transports the coal downhill. It is kept close to the face but within 450 to 600 mm so that manual loaders can shovel the coal on to the conveyor. Shifting the conveyor towards the face is done by pull-lift attached to iron wedges dug into inclined holes near the floor of the face. Two or 3 pull-lifts are worked simultaneously at 6 m distance apart. After the conveyor is shifted the space is ready for extending the wire netting and erecting a fresh row of props.

Roof Support In 1st Slice

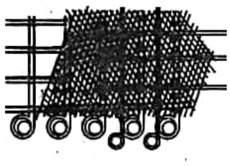
40-te steel friction props and 1.25 m long articulated cantilver bars are used. Interval between rows of props is 1.25 m and the distance between props in the same row is kept at 65 cms. The minimum span between the face and goaf edge is 3 bars *i.e.* 3.75 m and the maximum span, 4 bars *i.e.* 5 m. Inclined props are also erected near the goaf edge to strengthen the vertical props.

In the first slice the width of the opening of the face is of 3 cantilever bars before blasting and of 4 cantilever bars between blasting and caving. After blasting, one cantilever bar is extended on a bar shoe which takes the roof weight of the freshly exposed roof till a prop withdrawn from the rear (goaf edge) row is set below the bar. One bar has only 1 prop to support it and two bars are connected by the shoe. Withdrawal of props is being done with a small portable winch operated by comp air. Two to three props can be withdrawn by the which (7 te pull) at a time but normally only 1 prop is withdrawn.

Wire Netting

M.S. strips 60 mm wide and 3 mm thick are laid on the floor of the top slice parallel to the face and right angles to it so that they form a mesh with 40 cm x 40 cm apertures. Galvanised wire netting is spread over them tied to the strips by binding wires. The wire netting used is made of 2 mm dia. wires, 20 mm mesh and 1.5 m wide in the form of rolls (Fig. 14.4).

The wire netting is anchored to spikes of iron, 25 mm dia. driven at intervals into the floor of the bottom and top gate roads and also near the barrier of the panel from where the retreating face started. The wire netting is thus anchored on the periphery of worked out area on 3 sides and the 4th side which is not anchored is kept within 1 m of the face, and for extension rolls of wire netting are joined to the free side by galvanised iron stitching every 15-20 cm. Addition of roofs of wire netting and M.S. strips becomes necessary after each cycle of the face, immediately after the complete conveyor is shifted (by 1.25 m).



Left Fig. 14.4 Stronger false roofing to serve several slices

The first main roof fall took place after 40 m progress of the face. The indication of impending roof fall was the rumbling sound in the roof after supports were withdrawn from the worked out area. Sometimes it was necessary to induce the roof to cave by drilling 6 m deep inclined holes at the goaf edge by special drills and blast them. Roof pressure used to built up during an advance of five to six cuts and then pressure used to get released by breaking of higher roof strata accompanied with overall weighting on the face.

Extraction Of Second Slice

Entry to the second slice is made by sinking the existing entries for the first slice to the level of the second slice. The gate roads for the longwall face of second slice are not vertically below the gate roads of first slice but slightly staggered inside the panel. The arrangement helps to lock the wire netting with the solid coal left intact during extraction of first slice.

The longwall face of second slice is freshly driven underneath the wire netting. The face is started from the end of the panel where the first slice extraction ended leaving 6 m solid coal from the edge. The sold coal helps in locking of the artificial roofing. The first slice is taken by longwall retreating but the second slice is taken by longwall advancing. The bottom gate road of second slice serves for air intake and coal transport and the top gate road, for return airway and material supply. Due to the nature of the support system no coal cutting machine is used in 2nd slice and the necessary free face for blasting is obtained by serrating the face at five to six places (Fig. 14.5). Progress of the face is made by blasting at the serrations. The snaking conveyor touches the corners of these serrations (stables) which advance independently of one another. The serrations are at 12 to 15 m intervals.

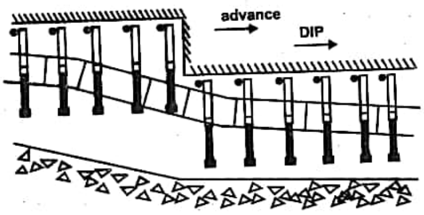


Fig. 14.5 Serration (or stable) at the face of 2nd slice. Total stables 5 to 6 along the face.

Support System

Main support in this slice is given by friction props and flexible bars in the form of 'V' i.e. one vertical prop and one inclined prop at its base. Flexible bar is made out of 1.5 m piece of steel rope 110 mm width with two attachments welded at its two ends to fit on top of the props. The vertical member of the 'B' is placed next to the coal edge and the inclined member helps the artificial roofing maintain profile. The wire netting should maintain profile in the are a where persons have to work and walk. As the respective stables advance, 'V's from the goaf edge row are withdrawn and placed ahead on the front row. Armoured conveyor of the face is placed at the goaf edge close to the artificial roofing as it comes to rest on the floor. (Fig. 14.6).

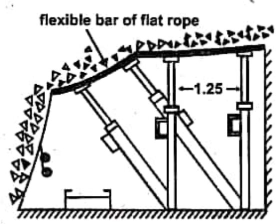


Fig. 14.6 Flexible roof bars of flat wire ropes.

Extraction Of Third Slice

The entry to the third slice is obtained in the same manner as in the case of 2nd slice by driving top and bottom gate roads to the level of 3rd slice. The top gate roads is staggered inside the face and the bottom gate road is formed by sinking the gate road of 2nd slice. The 3rd slice, 2.4 m thick, is extracted like the 2nd slice in longwall advancing manner and using the same wirenetting as the artificial roofing. Where the patches of wire netting are deteriorated due to long use only those patches are replaced by stitching fresh wire netting. Normally all extraction under artificial roofing should be completed within one year of its laying by working two or more slices simultaneously.

Sub Level Caving Of 4th Slice And Extraction Of 5th Slice

After extraction of 2 slices with the artificial roof of wire netting (*i.e.* a total of 3 slices) 4.8 m thickness of coal on the floor of the seam is still left to be extracted. Out of this 4.8 m of coal, the top 2.4 m which constitutes 4th slice is not extracted by usual process of coal cutting, drilling, blasting, conveyor, shifting, etc. Entry is made to the 5th slice (bottom-most slice of 2.4 m thickness) by suitable gate roads and the slice is extracted in longwalll retreating manner. The roof for the 5th slice is solid coal and not the artificial roof of wire netting. The progress in the main face (5th slice face) is made by blasting only. The minimum span of the face is two bars i.e., 2.5 m while the maximum span 3 bars or 3.75 m and each face advance by blasting is 1.25m.

When the goaf edge bars and the props are withdrawn, the coal of the 4th slice (called sub level) of 1.25 m width and 2.4 m height caves down usually by itself in the goaf, though sometimes it is necessary to bring it down by blasting.

At this face second wire netting is added along the roof of the main face (5th slice face) as the face progresses, mainly for the purpose of containing the sub level coal between the 2nd wire netting and the original netting lying below subsided goaf. The roof netting also prevents the immediate splash of broken coal which can inflict injury to the workmen and dislodge the supports inside the face. For extracting the sub level coal, it is necessary to puncture the netting at places or roll it up in sections. Big chunks of the sub level coal need secondary blasting and all the coal cannot be recovered resulting in loss of 10 to 20 percent (of the sub level coal). The coal remaining in the goaf is a source of spontaneous heating. The second wire netting (along the roof) has to be extended as the face advances.

In European mines wherever sub level caving is adopted coal breaks into small pieces and it can easily loaded on to the conveyor. The thickness of sub level is also somewhat high and as much as 22 to 30 m of sub level is being extracted with very little variation in the methods. Sometimes to remove the sub level coal the lower slice, *i.e.* immediately below the sub level, is equipped with 2 conveyors, one for the loading of the face coal while the other for the loading of the sub level coal, the latter installed on the goaf side where the sub level coal caves down.

Fig. 14.7 Sub level (4th slice)

(See Fig. 9.24. It shows an arrangement where a chock shield is used during sub-level caving).

Roof Support And Face Cycle

At the minimum span the face is supported effectively by two rows of vertical props and bars and the face conveyor is at the face end of the props. After the main face is advanced by 1.25 m by blasting, one bar is extended, temporarily on a bar shoe. The roof netting is also extended on the top of this bar. After cleaning of coal a temporary prop is set at the face end of the bar. Other ends of the freshly extended bars are supported by wooden runners 3 to 4 m long which connect 3 to 4 bars. The wooden runners themselves are supported on 3 to 4 pairs of inclined props, each pair being placed at the interval between the roof bars. The inclined props are placed to form an inverted 'V', the apex supporting the runner while one of the legs is placed at the new face edge and the other in line with the first line of vertical props. The bar is supported by inclined props in order to facilitate the shifting of the conveyor at a later stage.

The conveyor cannot be shifted now as the sub-level coal is yet to be drawn. The next operation consists of withdrawal of goaf edge props and bars and drawing of the sublevel coal. After the caved coal is loaded, the conveyor is shifted towards the face and another line of the props is erected below the bars which were so long supported on the temporary vertical props at one end and the inclined props at the other end.

The French method of inclined slicing and sub level caving was experimented in the first panel mainly the object of studying its application under Indian conditions. The production and productivity, therefore, were the secondary importance. The face workers were of secondary importance. The face workers were trained for all-men-for-all-job system. The steady progress that was obtained in the first slice was one cycle in two days with 78 face workers per day giving face O.M.S. of 2.4 to 2.6 tonnes. The present support system does not provide the prop free front. With the use of suitable types of cantilever bars and with a straight face a drum shearer can be introduced which can give better productivity at the face. With the present system a progress of one cycle per day was achieved with a maximum production of 7000 tonnes per month which gave a district O.M.S. of 3 tonnes.

Encouraged by the results of sub level caving in the first panel, the 3rd and 4th slices in the second panel were allowed to cave in as sub level when the 5th slice was taken, but the results were not satisfactory and a large percentage of coal was lost in the goaf, primarily because the chunks of coal of sub level were very large and difficult to handle.

Deterioration Of The Wire Netting Artificial Roof :

The deterioration of the wire netting is due to the following causes :

1. Puncturing by props erected on the artificial roof while working the first slice under the real roof of the seam.
2. The separation of adjacent pieces of wire netting, particularly during the working of the second slice, i.e. first under the artificial roof.

(C) Multi section working with thick coal partings and caving

Thick seams like Jambad and Poidih (Raniganh field) have been developed in two sections and sometimes in three sections, were the seam thickness is 13 m and more, by bord and pillar keeping the partings between adjacent sections of 3 m as required under the Coal Mines Regulations. Depillaring of such sections by caving is conducted in the same manner as applicable to contiguous seams described in the chapter on pillar extraction in bord and pillar. The percentage of coal lost in the goaf is excessive and the total percentage of coal extracted from the seam during development and depillaring is less than even 25.

Blasting Gallery Technique, et. For Thick Seam Extraction At SCCL.

In the coal mines of Singareni Collieries Co. Ltd. a few thick seams like King seam (6-25 m at Kothagudem), Queen seam (6-25m at Kothagudem and Yellandu), No. III and IV combined seam (7-16 m at GDK) and one thick seam (9-19 m at Manuguru) exist. A seam of 6m thickness is considered thick at SCCL and one less than 6 m thick is treated as normal seam. The methods adopted for extraction of thick seams at SCCL are described in a paper by S.A. Vyas and M.K. Bangamin contributed at a Symposium on "Thick seam mining" held by CMRS in 1992. The relevant extracts of that paper are given below.

In thick seams of 7-8 m thickness development by bord and pillar method was in two lifts/sections in panels with superimposition of pillars and galleries of the two sections. The size of panels was such as to complete the depillaring within the incubation period which varies from 9-18 months. The depillaring is carried on in the manner described in the chapter "Pillar extraction in Bord and Pillar" for simultaneous depillaring of contiguous seams. It was observed that extraction percentage is not more than 35 as substantial amount of coal is lost in the panel barriers and about 25% of coal is lost in the parting between the top and bottom sections. About 20% coal is lost in the actual extraction in the panel itself.

Another method adopted is to extract the pillars of thick seam in two lifts, with stowing in bottom section and caving of top section during depillaring. By this method about 58% of coal can be extracted in two lifts.

The methods which have been introduced in recent years are :

(a) Multiple extraction by total stowing.

(b) Top section caving by hand section and bottom section caving by LHDs.

(c) Blasting gallery technique by caving.

Methods Which Are Recently Introduced (Early 1990s)

(a) Multi lift extraction by total stowing

At Gauthamkhani of Kothagudem Area, the Kind Seam thickness is in the range of about 20-25 m with a stone band of about 2.5m - 4.2 m in the middle. The seam is liable to spontaneous heating and incubation periods is varying from 9-12 months. The seam is developed in bottom and middle sections initially. Superimposition of galleries and pillars are strictly being followed. The relevant portion of the method of work and salient permission from DGMS are enumerated below :

1. Depillaring operations shall not be commenced unless it is fully ensured that the panel is isolated effectively to prevent danger from other panels/goaved areas from gases, inundation, etc.,
2. During actual extraction, each pillar shall be split into four parts by driving central dip and level split gallery not exceeding 4.2 m in width. The original galleries shall not be further widened.

3. The resultant stook being left shall not be less than 5 m x 5 m from corner to corner.
4. Splitting of pillars shall from the dip most area of the panel in retreating manner. No pillar shall be split unless the adjacent inbye voids are stowed with hydraulic sand stowing.
5. Not more than two pillars shall be under splitting at any time and not more than two such pillars shall be kept void.
6. As far as possible, after completing bottom – most lift, the second lift and subsequent other lifts shall be taken from splitting and stowing in retreating manner. Similar operations will continue in ascending order from 1 lift to the next higher lift.
7. While developing pillars over sand, the gap formed due to shrinkage of sand shall be filled up as far as practicable while taking the upper lifts.
8. The depillaring district shall be inspected by an Under Manger atleast once in a day.
9. Regular gas samplings as laid down in Mines Regulations shall be done so as to detect the presence of CO in the return air.
10. During the extraction of coal, the ground movement on surface shall be observed and a proper subsidence record shall be maintained.

It has become possible to extract upto about 60-60% of thick seam in each panel by following the above system. With regular supply of sand by proper barricading no serious problems have been encountered in this system and the self-heating of coal and premature collapse of other lifts has been minimised.

(b) Top section caving by hand section and bottom section caving by LHDs

There are two workable seams at GDK-9 Incline in Ramagundam Area. The thick seam is called No. III seam which is about 8-9 m while the Bottom seam is called No. IV seam which is about 304 m in thickness. The parting between the two seams is 6-7 m. The seams are dipping at a gradient of 1 in 8 to 1 in 9. As part of conservation of coal the III seams is developed in two sections with a minimum parting of 3 m. All the dips and levels in each panel have been superimposed. Top section is mostly developed to a height of about 2.4; so also the Bottom section upto 2.4-2.6 m.

Simultaneous extraction of two sections is permitted with slight advance operation in Top section to a maximum of about half a pillar Regular test holes are put to confirm the workings of Bottom section in relation to the Top section. As far as possible even during the depillaring the height of extraction in top section is maintained around 2.4-2.6 m only. Both in top section and bottom section the pillars are split into two with a level split of about 4.2 m; thereafter half pillars are taken in batches in top section, while in bottom section the slices are taken. Conventional supports are used in top section while in bottom section the support system is slightly different. By employing 3 LHDs it has become possible to produce upto 850 te per day and monthly output was achieved upto 22,350 tonnes. In Bottom section roof coal was taken to a maximum of about 4.5 m in the retreat operation. The ribs also have been reduced judiciously from a thickness of 2 m to less than 1 m wherever it was possible. At GDK-9 this method proved to be very successful with good safety record as well as production. It is expected in future also wherever the seam gradient is less than 1 in 6, the thick seams can easily be extracted in two lifts with maximum recovery of coal upto about 70% in each panel.

(c) Blasting gallery method by caving

GDK-10 Incline is situated in Ramagundam Area where III & IV seams are being extracted by normal conventional method. III seam is about 10-12 m in thickness and it has been successfully tried to be extracted in one lift by Blasting Gallery Method with collaboration of M/s. Charbonnage de France International. This mainly involves drilling holes right through the entire thickness of the seam by jumbo drills in a systematic fan cut fashion, bringing the coal down by blasting with special explosives and loading the coal by remote control LHDs. Main advantages of Blasting Gallery Method are :

1. The full thickness of the seam can be extracted in one lift with percentage recovery tupo 70-80.
2. Capital investment is almost 1/3rd of Longwall equipment with an expected daily output of about 850-1000 tonnes.

3. Easy to train the required manpower and easy maintenance of the equipment.

4. A panel of about 150 m × 1000 m and thickness of the seam ranging from 10-12 m can extracted in about 4 years with systematic planning and supervision.

5. Most of the machinery deployed in Blasting Gallery method can be manufactured in India; hence, spare parts management becomes easy which reduces the importance of foreign spares and equipment.

The Actual Method Of Blasting Gallery :

Blasting Gallery method envisages drilling and blasting the entire thickness of the seam by successive blasts while retreating along the level gallery which is driven along the floor of the seam. A panel of about 1000 m × 120-150 m is divided into sub-panels of 150 m × 120-150 m by driving main rises. A barrier of about 15 m is left in between the panels. It is necessary to adopt this sub-panell in to restrict the size of panel so that the extraction is completed within the incubation period (in III seam incubation period is between 12-18 months). The subpanel, after extraction of the coal, is to be sealed off effectively. The sub-panel is further divided into two parts by driving a central main to reach the boundary of the panel. Such rooms are driven at 13-15 m × 60-65 m is formed in between the rooms. The main rise and central main drives are 4.7 m in width to facilitate the housing of chain conveyor and also the movement of LHDs and jumbo drills.

All the development is done along the floor of the seam, to a height of about 3 m. It is essential to stick to the correct sizes of room and rises as otherwise it is likely to land into roof control problems at a later stage. The development of rooms and rises is done by road headers as this would ensure the correct size of galleries as well as the required progress.

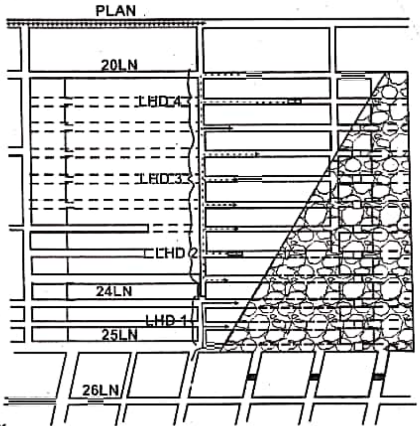
System of supporting : The galleries are supported by 100 × 200 mm built up steel roof bars held in position by means of

two nos. of 40 te capacity open circuit hydraulic props. Such bars are placed at 1.5 m apart by steel braces. Upto 40 m from the face in each blasting gallery, the above supports are installed and maintained. Junctions are supported with cluster of the above supports as per requirement.

Drilling : Holes are drilled by means of jumbo drills. The jumbo drill with a speed of 1 m/min. is capable of drilling upto 30 m long holes. The drill rods are hollow and have 32 mm dia of double female type. The diameter of drill hole is 43 mm, water is injected to the central hole of the rod at 20 bars to eliminate the dust produced while drilling. The holes are drilled in such a way that they cover half the pillar towards the rise side and half the pillar towards the dip side from the gallery where the jumbo is situated 30-33 holes are drilled in a ring which takes about 5 hours. The length of holes varies from 2.3-10.8 m.

Blasting : Originally the quantity of explosive per shot was restricted to 2 kg only which has not given proper fragmentation of coal. As a result of the above large coal chunks were produced needing secondary blasting. On further examination by Mines Directorate and project proponens, explosive quantity is increased to 3 kg per shot with proper plastic spacers. The diameter of the explosive cartridge is normal 32 mm only. Delay Action Detonators are being used along with detonating fuse for initiation and blasting purpose. The entire operation of drilling and blasting takes about 1.5 shifts.

Loading and transportation of coal : The blasted coal is loaded by LHDs having remote facility. About 800-900 te of coal is produced from blasting gallery district per day. The LHD will unload the coal on to a chain conveyor installed in the rise. A lump Breaker is installed near the belt conveyor to reduce the coal to (-) 200 mm size. Finally the coal is transported to surface from underground through a series of belt conveyors. With the successful operation of first blasting gallery district, SCCL is envisaging to introduce the system in some more underground projects at Ramagundgam in order to improve the production and productivity in near future. The OMS achieved in blasting gallery is more than 4 tonnes.



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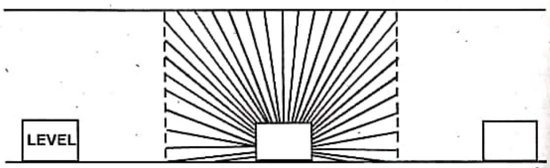
Blasting gallery district plan.



Rill drilling principle

Fig.

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