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Denett

ELEMENTS OF MINING TECHNOLOGY

VOL. 2

Part A & B

D. J. Deshmukh

A.I.S.M., M.M.G.I., C.C.M.
Co-author of Wining and Working Coal in India
and
The first Indian Author of Text/reference books on
Indian Mining Technology
Examiner for B. E. (Mining), Nagpur University

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Preface to 9th Edition

This revised edition includes matter on profilometer, massblasts at Zawar group of mines (HZL), Radon, Nitrogen flushing to quench underground fires, and a few other topics. Spiralarm finds no place in this edition as it has gone out of use with the introduction of electronic devices to measure methane percentage and warn the workers. The chapters on mining methods at Kolar gold Field have been deleted as they are no longer relevent with the stoppage of workings at greater depths. A noteworthy addition is the inclusion of a few photographs.

A few examples on mine ventilation have been included in S.I. Units but the majority of the numbericals are in metric units for an easy grasp by the students.

Some information of Indian Mining in Part B has been reproduced from the Mining Annual Reviews of Mining Magazine, London by kind permission of the management. Updating of information in Part B has been based on the published technical literature.

My thanks are due to the following companies and organisations which rendered valuable assistance for revising and updating this book.

Central Mining Research Station,	*Dhanbad*
J.K. Day & Sons,	*Calcutta*
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Volts Ltd.	*Calcutta*

• *Publisher* •

Conversion of common FPS and metric units into S.I. units

Force

1 kgf = 9.81 Newtons
1 lbf = 4.448 Newtons
1 tonf (long ton) = 9.964 kilo Newtons

Pressure

1 lbf/in^2 = 6.895 kN/m^2
1 $tonf/in^2$ (Long ton) = 15.44 MN/m^2
1 atoms (standard) = 101.3 kN/m^2
= 100 kN/m^2 approx
1 bar or b = 10^5 N/m^2
1 kgf/cm^2 = 98.1 kN/m^2
1 kgf/mm^2 = 9.81 MN/m^2
1 mm w.g. = 1 kgf/m^2
= 10 Pascals

Speed

1 mile/h = 1.609 km/h
1 mile/h = 0.4469 m/s

Acceleration

1 ft/s^2 = 0.3048 m/s^2

Energy

1 ft lbf = 1.356 J

Moment of inertia

1 lb ft^2 = 0.04214 kg m^2

Power

1 kW = 1.341 H.P.
1 H.P. = 746 Watts

Torque

1 lbf ft = 1.356 Nm

Temperature

K = 273 + °C

Density

1 lb/ft^3 = 16.02 kg/m^3

GRADIENT

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

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PART - A

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In this edition chapters on mining methods at Kolar Gold Field have been deleted as they are no longer practised, with the closure of working at lower depths.

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Content of Three Volumes of the Series

ELEMENT OF MINING TECHNOLOGY

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CH. 1. Mining Geology : Minerals, Rocks & Rocks Structures. **2.** Coal and Coalfields of India. **3.** Boring. **4.** Shaft sinking. **5.** Opencast Mining. **6.** Access of Minerals Deposits & Pit-bottom, Pit-top layouts. **7.** Drivage of Roads in Coal & Stone. **8.** Explosives, Accessories & Blasting Practice. **9.** Rock Mechanics and Roof Supports. **10.** Stowing Practice **11.** Bord & Pillar Method of working Coal; Development **12.** Pillar Extraction in Bord & Pillar **13.** Longwall & Other Methods of Working **14.** Thick Seam Working.

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PART A

ELEMENT OF MINING TECHNOLOGY

Volume-II

CHAPTER 1

MINE GASES AND THEIR DETECTION

The air of the atmosphere that we breathe is a mixture of several gases and its composition is practically constant over the whole surface of the earth from the sea level upto an altitude of at least 25 km. Because air is a mixture and not a chemical substance, the components can be separated. This is normally done by cooling it to about –196°C after which the various components are separated by fractional distillation. Typical analysis of atmospheric air which also represents the intake air of any mine is given below.

Constituents	by weight %	by volume %
Oxygen	23.15	20.93
Nitrogen (Including Argon and other rare inert gases)	76.81	79.04
Carbon dioxide	0.04	0.03

Argon is an inert gas, 0.94% by volume in the atmospheric air. It behaves like nitrogen and therefore its percentage is normally included in the percentage of nitrogen.

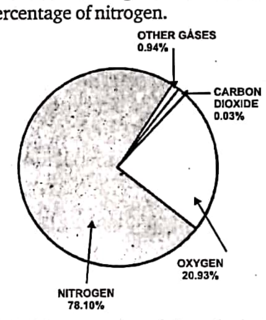

Fig. 1.1 Composition of atmospheric air

Analysis of air in the return airway of a gassy coal mine (deg. 2 and 3) is generally as follows :

Constituents	% by Vol
Oxygen	20.28
Nitrogen	78.90
CO_2	0.36
Methane	0.46

It will be observed that return air is depleted in oxygen content and contaminated by mine gases. Impurities come from exhalation by men, blasting, underground fires, burning of lights, bacterial action and gases given off from strata. It also contains moisture and dust of coal and rock.

Dry air is subject to the general laws of gases, given later.

When referring to noxious and poisonous gases, met with in a mine the commonly used names are as follows :

(a) **Blackdamp :** It is a mechanical mixture of the extinctive gases, carbon dioxide and excess nitrogen; sometimes it is referred to as chokedamp or stythe.

(b) **Firedamp :** It is used either as (i) synonymous with methane or (ii) referring to the mechanical mixture of gases, chiefly inflammable, given off naturally from coal and consisting for the most part of methane, CH_4.

(c) **Whitedamp :** It is synonymous with carbon monoxide, CO.

(d) **Stinkdamp :** It is synonymous with sulphuretted hydrogen, H_2S (Hydrogen sulphide)

(e) **Afterdamp :** This is a mechanical mixture of gases existing in a mine after an explosion of firedamp or coal dust. Its composition is extremely variable but usually includes carbon monoxide, carbon dioxide, nitrogen and sometimes H_2S and SO_2 with very small percentage of oxygen. The percentage of CO and CO_2 is much in excess of what is normally found in a mine.

THE GAS LAWS :

There are certain laws governing perfect gases. They are applicable to mine gases and atmospheric air (strictly speaking, applicable to dry air only) to a great extent and are as follows. They figure often in ventilation calculations :

1. Boyle's Law : The volume of given mass of gas varies inversely as the absolute pressure if the temperature remains constant.

$$\text{Thus} \quad V \alpha \frac{1}{P} \; ; \text{ or } P_1 \times V_1 = P_2 \times V_2 = \text{Constant}$$

where P_1 and V_1 are initial absolute pressure, and volume respectively and P_2 and V_2 are final absolute pressure and volume respectively.

A corollary that follows from Boyle's law is :

The density, or weight per m³, of a gas is directly proportional to the absolute pressure, if temperature is constant.

2. Charle's Law : The volume of a given mass of gas is directly proportional to its absolute temperature when the pressure is constant.

$$V \alpha \text{ abs. temp. T, or } \frac{V_2}{V_1} = \frac{T_2}{T_1} = \frac{273 + C_2}{273 + C_1} = \text{constant}$$

where T_1 and T_2 are absolute temperatures and C_1 and C_2 are the temperatures observed on Centigrade scale.

The absolute temperature is also known as thermodynamic temperature and it is equal to the temperature in °C + 273.15 For practical purposes it is customary to use the figure 273 instead of 273.15.

A corollary that follows from Charles' law is : The density, or weight per m³, of a gas is inversely proportional to the absolute temperature when the pressure is constant.

3. A combination of Boyle's law and Charle's law provides that if a given mass of gas is subjected to a change of pressure and temperature at one and the same time, then

$$\frac{P_1 V_1}{T_1} = \frac{P_2 V_2}{T_2} = \text{a constant, R}$$

The value of R is constant for a particular gas and also for air. R is known as the **characteristic gas constant.**

4. Graham's law of diffusion :

The relative rates of diffusion of a number of gases vary inversely as the square root of their relative densities.

$$R \alpha \frac{1}{\sqrt{D}}$$

where R is the relative rate of diffusion and D is the relative density or

$$\frac{RA}{RB} = \sqrt{\frac{DB}{DA}}$$

where RA and RB are the relative rates of diffusion of gases A and B respectively and DA and DB are their relative densities.

The density or air, D is given by the equation

$$D = \frac{0.4645\ B}{273 + t}\ kg/m^3$$

where B = barometric pressure in mm of Hg

t = temperature in °C

Density of air at 15°C and 760 mm Hg is 1.218 kg/m³.

COMMON GASES IN A MINE :

Oxygen, O_2 :

This is a very vital element in the atmospheric air which human beings and animals have to breathe for their existence. The gas is colourless, odourless, tasteless, slightly soluble in water and slightly heavier than air, (sp. gr. 1.1*) ; 100 vol. of water at 20°C dissolves about 3 vol. of O_2. Critical temp. –119°C and critical pressure, 50 atoms. It is a most chemically active element which reacts with almost all the elements and supports combustion.

Oxygen of the air in a mine is consumed for :

1. Breathing by persons,
2. Slow oxidation of coal and other carbonaceous materials,
3. Burning of flame saftey lamps, and in a non-coal mine, acetylene lamps or naked lights,
4. Decay of timber by fungus growth.

This accounts for the diminished percentage of O_2 in the return air of a mine. Another reason is the mine gases given off from coal and adjacent strata. They reduce the overall oxygen percentage in air.

It has been found that a person at rest in bed consumes 0.3 liters/min. of oxygen, a person walking at 3 km/hr consumes nearly 0.78 litres/min. and a person walking at 8 km/hr or doing ordinary exercise requires nearly 2.0 litres/min. of oxygen.

Each 1% reduction in the oxygen percentage results in about 30% less light from oil lamps and the latter is extinguished when the oxygen percentage falls to about 17.5. It should be noted that if a candle or a flame of safety lamp continues to burn in any place, the atmosphere is safe for human beings if other poisonous gas like carbon monoxide or H_2S is not present, since

* *The term specific gravity as applied to gases refers to the weight of an given volume of gas compared with the weight of an equal volume of air at the same pressure and temperature.*

human beings can survive in an atmosphere containing as low as 15% oxygen. Canaries or small birds can survive if O_2 percentage is as low as 8 and they are not good guides for sufficiency of O_2 for human beings. An acetylene lamp, however, burns when the oxygen percentage is as low as 12 to 13 and it is not a good indicator for detecting oxygen deficiency which may prove fatal for a human being.

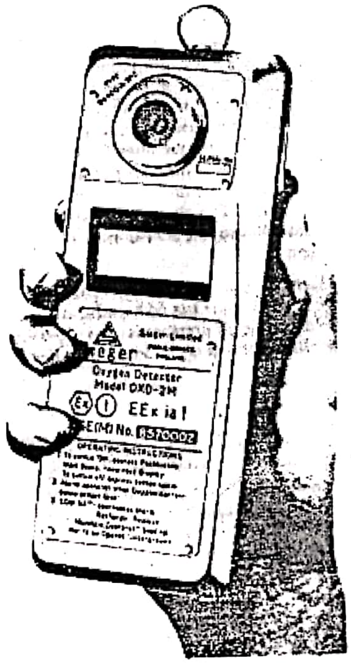

Fig. 1.2 Oxygen deficiency monitor, Mpodel OXD-2M

Mining laws in India require that mine air should contain minimum 19% of O_2. This minimum limit is 20% in USSR and 19.5% in USA.

Uptron of Lucknow is manufacturing oxygen deficiency monitor. Model **OXD-2M** is a compact hand held instrument designed to indicate oxygen deficiency and incorporates an audible alarm which operates automatically if oxygen level falls below a preset point. It has a fast response, long life electro chemical cell sensor. A three digit LCD indicates oxygen concentration within the range 0 - 30% v/v and the same display is also used to indicate low battery or over range condition. The instrument contains a rechargeable Ni-Cd battery, 110 mAh which will give more than 27 hours' continuous operation when fully charged. The instrument has a warm up time of only 5 seconds. It is small enough to carry in the pocket and there are also facilities for connecting a line or a probe and aspirator to allow measurements in inaccessible areas such as sumps, etc. The instrument is intrinsically safe for use in underground coal mines. The alarm is adjustable from 14% to 23% of oxygen content. Dimensions 192 mm × 80 mm × 56 mm; weight 800 gms.

The inhalation of pure oxygen under normal pressure for a limited period of upto 6 hours has no harmful effect. In deep mines, 1000 to 1500 m deep, the air pressure is about 15% more than at the surface. But human breathing is not noticeably affected by the extra pressure.

Nitrogen, N_2 : This gas is quite abundant in atmospheric air comprising about 4/5th of the atmosphere. It is colourless, adourless, tasteless, nearly as heavy as air with a sp. gr. of 0.967. It is practically insoluble in water, 100 vol. of water dissolving at 15°C only 1.8 vol. of nitrogen. It is an inert gas which neither burns nor supports combustion but it is important for the growth of plants and animal tissues. It undergoes no alteration in the process of breathing. Though the nitrogen we breathe in has no metabolic function it serves as an inert diluent and maintains the inflation of certain gas filled body cavities such as pulmonary alveoli, the middle ear and the sinus cavities. Critical temperature –146°C critical pressure 35 atmos.

* *To liquefy a gas it must be first cooled below a point known as its critical temperature. Thereafter condensation to a liquid is caused either by compressing the gas to its critical pressure, or by further cooling the gas to its boiling point at atmospheric pressure, or by a combination of both cooling and compression. Above the critical temperature a gas will remain gaseous no matter what pressure is applied. At the critical temperature, the critical pressure must be applied to cause condensation. Below the critical temperature the gas will liquefy at pressure less than the critical pressure.*

When men work at pressures higher than atmospheric, the blood and tissues of the body begin to absorb nitrogen. But if the high pressure is abruptly reduced the nitrogen is also given up by the body quickly and this results in painful and dangerous conditions. This phenomenon is guarded against during caisson method of shaft sinking in heavily water bearing strata when men have to work in caisson and the air is forced into the caissons in the shaft at such a pressure that the water is pushed back into the ground to keep the working place dry. The worker is subjected to the high air pressure in 2 – 3 stages in different compartments when going to the work spot and the reverse process takes place when he returns to the surface having atmospheric air.

Nitrogen is sometimes used for quenching underground fires.

Carbon Dioxide, CO_2 :

This gas is colourless, odourless, bitter in taste, with a sp. gr. of 1.52. It is very soluble in water, 100 vols. of water dissolving at 15°C 100 vols. of CO_2 forming a weak acid. It is not combustible and does not support combustion. It does not sustain life. Critical temp. is 32°C.

The gas is a product of the process like respiration by human beings and animals, oxidation and combustion. It is present in the return air of all mines in very small percentages and is found in the dip areas of depillaring districts in coal mines due to its heavier than air nature. It is produced in a mine by breathing by men, burining of flame lamps, decay of timber, slow oxidation of coal in mines, blasting, and working of internal combustion engines such as diesel locomotive. In some coal mines it may be given off by outburts of gas. Mine fires and explosions result in the formation of carbon dioxide along with other gases.

With the aid of sunlight during day plants absorb CO_2 from air and by photosynthesis split up the gas into constituent atoms, absorbing the carbon necessary for their growth and setting free the oxygen which is used by human beings and animals. In this way nature continually renews the oxygen content of the air, so keeping it almost at a constant percentage in the atmosphere.

The following are the effects of breathing air containing large percentage of CO_2.

If CO_2 replaces air by

3% – breathing doubled (at rest)
6% – violent panting, headache, exhaustion
10% – severe distress endurable for a few minutes; after half to one hour of work, suffocation and unconsciousness.
15% – consciousness loss
25% – death after hours

For a person affected by CO_2 recovery with artificial respiration is usually rapid because of rapid breathing when CO_2 is present in lungs.

Carbon dioxide has an extinguishing effect on the flame of a flame safety lamp; the flame becomes dim at low concentration and will extinguish if held in it for long. In still air flame of a flame safety lamp is extinguished at a CO_2 percentage of 3-4.

BLACKDAMP :

This is a mixture of CO_2 and nitrogen in percentages higher than the normal percentage in the mine air. The CO_2 percentage ranges from almost negligible to 20% and that of nitrogen ranges from about 80% to 100%. The composition mostly depends upon the manner of formation of CO_2 in the mine; if it is from the oxidation of coal, then CO_2 - 5% and N_2 - 95% if from rotting timber, CO_2 - 20% and N_2 - 80%. The gas is colourless, odourless and may have an acid taste due to the presence of CO_2. It does not support combustion and is not poisonous but men cannot live in it due to lack of oxygen. Its sp. gr. varies according to the composition and blackdamp with 5% CO_2 has sp. gr. equal to that of air. The effects of breathing blackdamp depend on its composition. If it contains a large percentage of N_2 the effects are similar to that as in the case of deficiency of oxygen.

The physiological effects of blackdamp (12% CO_2 and 88% N_2) are as follows :

25% blackdamp (75% air, *i.e.* 15% O_2 and 3% CO_2) rate of breathing doubled.

40% blackdamp (60% air *i.e.* 12% O_2 and 5% CO_2) more frequent and deeper breathing.

50% blackdamp (50% air, *i.e.* 10% O_2 and 6% CO_2) panting, headache and face turns blue.

The heavier blackdamp gives a warning of presence of CO_2 but the lighter blackdamps do not give such warning and are therefore more dangerous. A victim of blackdamp recovers rapidly on breathing fresh air or by artificial respiration.

The presence of blackdamp affects the flame of an oil flame safety lamp. For every 5% of blackdamp (corresponding to 1% reduction in the percentage of O_2) the light diminishes by 30% and extinguishes when the oxygen percentage falls to below 17.5% which corresponds to about 17% of blackdamp. The gas, if present, is near the floor of the roadway and a flame safety lamp or naked light dims when held near the floor but shows its normal brightness when held up unless the whole height of the roadway is full with gas.

Blackdamp can be partially cleared away by sprinkling of lime but large concentrations can be removed only by improving ventilation.

Carbon Monoxide or Whitedamp, CO :

2012

It is produced whenever carbon or carbonaceous matter is burnt with insufficient supply of oxygen. The gas is colourless, odourless, tasteless and non irritating. It is only slightly lighter than air (sp. gr. 0.967) and combustible but does not support combustion. It is hardly soluble in water, 100 vols. of water dissovling 2.3 vols. of gas at 20°C. In air it burns with a light blue flame to CO_2. It forms an explosive mixture with air when present within the range of nearly 12% and 75% by vol. Critical temp. –140°C, critical pressure 35 atmos.

The production of CO in a mine may be due to any one or more of the following causes.

1. **Oxidation of coal and other carbonaceous matter :** Incomplete oxidation may result in its formation and under normal mining conditions, the percentage formed is negligible and harmless in the return air of a coal mine.
2. **Explosives :** Explosives contain the amount of oxygen required for complete chemical reaction but the chemical reaction when the explosive is blasted is seldom perfect and this results in the formation of CO. Workers should not be allowed to return to the place of blasting untill the fumes have been cleared by ventilation.
3. **Spontaneous combustion :** This is the main source of production of dangerous percentages of CO in a coal mine. Active fire in an underground mine also forms CO in dangerous percentage.
4. **Methane or coaldust explosions :** Gases produced by the explosions of methane and coaldust invariably contain large percentage of CO, which are responsible for deaths of victims who might have escaped the violence of explosion proper.
5. **Underground machinery :** Air compressors, run faultily, and exhaust gas of internal combustion engines like diesel locomotives, are common sources of production of CO. In fact every machine produces some CO if proper lubricants are not used. The general air on a road must not normally contain more than 0.005% of CO. (This is ensured by adequate air current in a mine).

PHYSIOLOGICAL EFFECTS OF CARBON MONOXIDE :

Carbon monoxide is a very poisonous gas and its affinity for the haemoglobin of the blood is nearly 300 times that of oxygen. If CO is present even in small quantities in the inhaled air, it is difficult for blood to absorb proper quantity of oxygen to support life because of the formation of a stable carboxyheamonglobin when carbon monoxide reacts with haemoglobin. A person exposed to atmosphere containing CO may not know that he is inhaling the poisonous gas unless he is equipped with CO detecting devices. Carbon monoxide imparts a pink tinge to the blood and a man thus poisoned presents, when alive and even in death for some time, a most life like appearance.

The effects of breathing air containing CO are :

% of CO in air

0.02 – headache, discomfort and possibility of collapse after 45 minutes at work or two hours at rest.

0.12 – palpitations after 10 minutes at work or 30 minutes at rest.

0.2 – Unconsciousness after 10 minutes at work or 30 mts. at rest.

0.5 to 1.0 – Death after 10 - 15 mts. of work

The after effects of CO poisoning are headache, loss of strength, and in some cases, even paralysis.

DETECTION OF CARBON MONOXIDE :

An efficient method should be adopted to detect CO in the initial stages itself. For this purpose warm blooded birds like munia or mouse are commonly used as they are affected much earlier than man by CO. Such birds form the essential equipment of a rescue party entering a mine after explosion or fire. Only fresh birds are used as some may get accustomed to small percentages of the gas. With 0.15% of CO present in the air a bird shows distress (ruffing of feathers, pronounced chirping and loss of liveliness) in 3 mts and falls off its perch in 18 mts. With 0.3% CO the bird shows almost immediate distress and falls off its perch in 2 - 3 mts. Immediate signs of distress are not likely to be observed on birds when exposed to only 0.1% of CO; they are visible only when the concentration is more than 0.3%. Ordinary sparrows are not suitable for CO detection and in an atmosphere containing CO to the extent of 1 to 2% the sparrows did not exhibit any signs of distress untill they were dead. For atmospheres containing more than 0.15% carbon monoxide, as is met with in mine resuce work, a cage with munia birds is a good indicator.

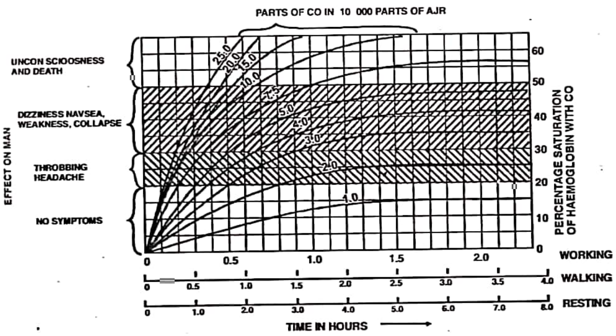

Fig. 1.3 Physiological effects of carbon monoxide (after T. D. Spencer)

Carbon monoxide detectors are based on the chemical reaction of CO with certain chemicals resulting in a change of colour. The chemicals are kept in detector tubes which are sealed at both ends and can be stored for 1-2 years. The seals are broken at site just before detection of gas percentage and the tube inserted into an aspirator which draws in mine air. The air containing CO causes change in the colour of the chemicals in the tube. The resulting colour is compared to a standard chart by which the percentage of CO can be known. Some of the detectors are :

The P. S. detector :

This detector consists of a glass tu… containing silica gel impregnated with light yellow potassium palladium sulphite with silica gel at both ends for absorbing other gases. A fixed volume of air (1?0 cm^3) is drawn through the tube at a constant rate over a period of tw… minutes through a calibrated orifice by operating a rubber aspirator. Carbo… monoxide in th… air turns the light yellow colour of potassium palladium sulphite to brown an… e length of colour change from one end of the tube indicates its oncentration. Range of the instrument is 0.005% to 0.12% of CO. The detector is manufactured by Siebe Gorman and Co. Ltd., Britain.

The Hopcalite detector :

The generation of heat on the oxidation of carbon monoxide to carbon dioxide has been utilised by a detector manufactured by the Mine Safety Appliance Co., USA. It consists of an analysing cell containing hopcalite, which is a specially prepared mixture of manganese dioxide and copper oxide. Air is drawn into the cell by a hand operated pump. Carbon monoxide in the air is catalytically oxidised by hopcalite and the rise in temperature so produced is recorded by a thermocouple connected in series to a meter which is calibrated in the percentage of carbon monoxide. Range of the instrument is from 0.005 to 0.015% carbon monoxide.

The Hoolamite Tube :

In another detector hoolamite which is a mixture of iodine pentoxide and sulphuric acid is used. The greyish white colour of hoolamite will be converted to a shade of green, brown or black by the reaction of CO which liberates iodine. The percentage of CO may be estimated from a colour chart.

Drager Multigas Detector :

A firm of Drager has produced a very convenient and handy instrument for detection of not only CO but also other gases. The instrument is used for on the spot determination of the gas%. If CO is to be detected a tube containing chemicals sealed at both ends and marked 'CO', is fitted at the spot of detection into a hand operated bellows type pump after breaking the sealed tips at both the ends. The arrow of the tubes should point towards the pump. The bellow is pumped a prescribed number of times, varying from 1 to 10 and the extent to which the chemical changes its colour gives the percentage of gas which is read directly on the tube. Minimum percentage of CO detected is 5 parts per million (ppm). The tubes are supplied in packs of 10 and a tube can be used only once. The whole operation takes about one to two minutes. The shelf life of these tubes is 2 years. There are Drager tubes for measurement of the most varied gases such as CO, CO_2, H_2S, ethylene, methane, etc. The presence of other gases and moisture in air does not affect the reading of the gas to be detected. The manufacturers prescribe the following number of pump strokes for the following range of measurement (at 20°C, 760 mm Hg. pressure).

Gas	Anticipated range of measurement ppm or percentage by volume	No. of pump strokes
CO_2	0.1 to 1.2%	5
	5 to 60%	1
CO	10 to 300 ppm	10
	100 to 3000 ppm	1
	0.1 to 1.2%	1
H_2S	10 to 200 ppm	1
	0.2 to 7%	1

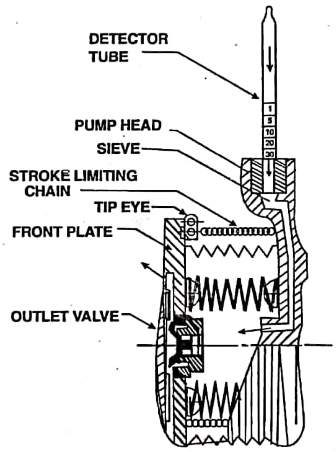

Fig. 1.4 Bellow type pump used with CO detector tube

Carbon Monoxide detecting of M.S.A. Ltd. : (2008)

The firm of M.S.A. markets CO detecting instrument which consists of an aspirator, a colour comparing scale and CO detecting tubes. The CO Detector (Aspirator) allows the user to use a variable number of squeezes (1, 2 or 5), thereby enabling detection of CO concentrations from as low as 10 ppm to as high as 1000 ppm by the colorimetric process i.e. comparison of colours. The tube contains a chemical (yellow silica gel impregnated with palladium sulphate and ammonium molybdate) and is sealed at both ends.

It is inserted at the spot of detection into the CO detecting aspirator after breaking the sealed tips. The aspirator bulb (60 c.c.) is squeezed once, twice of 5 times and the CO sucked into the instrument changes the colour of the chemical in the tube to green. The extent of colour change depends on the percentage of CO. Each squeeze of the bulb (60 c.c. capacity) sucks in a fixed volume of the air to be sampled which passes through a detector tube fixed at the intake end. The change in colour of the indicating gel of the detector tube is matched with the colour chart available on the barrel of the detector and the percentage of CO read off corresponding to the number of squeezes. Shelf life of tubes, 18 months.

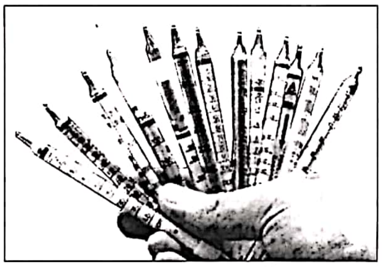

Fig. 1.5 CO Detector (Aspirator) and the CO detector tubes

If the percentage of CO is small, upto 10 ppm 5 squeezes of the aspirator bulb are necessary. The yellow reacting gel is moisture sensitive and the guard gel is not sufficient to remove the moisture in large volumes. The tubes are available in packs of 10 and a tube can be used only once.

C. M. R. S. has developed a suitable device consisting of various graphs which can be fitted to the existing M. S. A. CO detector tubes for determination of carbon monoxide in samples containing upto .0002% of CO. This device would be useful in detection of minor traces of CO in mine atmosphere. Without the device of C. M. R. S. the CO is detected in the range of 0.001 to 0.10% by volume (i.e. 10 to 1000 ppm.)

EMCOR carbon monoxide detection and monitoring system :

The EMCOR carbon monoxide detecting and monitoring system has been developed by the British National Coal Board as a multipurpose analyser for underground use. The instrument is approved in intinsically safe to BS-1259, 1958.

Specifications :

Range :

1. 0 - 50 ppm
2. 0 - 500 ppm

Outputs :

1. 0.4 - 2.0 V Analogue
2. Control 'Mine Flash'

Dimensions :

1. Height 260 mm
2. Width 100 mm
3. Depth 75 mm
4. Weight 2.27 kg.

Fig. 1.6 EMCOR CO detection & monitoring system
(1. Sensor, 2. 0-500 ppm rnage, 3. 0-50 ppm range, 4. Liquid crystal display, 5. Multiflash output, 6. Analogue output)

Its single design concept makes it usable in any of the three ways.

1. As a handheld instrument to replace stain type detector tubes.
2. As a local alarm, to warn of concentrations of carbon monoxide above a preset level.
3. As a continuous monitor linked to an underground data transmission system to give an immediate indication at the surface of fire or spontaneous combustion.

The sensor is the well tried electrochemical cell and by use of chemical filters interference by hydrocarbons which may be found in mine air has been minimised. For example, the response to 100 ppm ethylene is less than 1 ppm (as carbon monoxide). The instrument is powered by 2 duracell Mn 1800 dry cells which give more than three months of continous operation.

There is a highly visible liquid crystal display which is unaffected by an internal range switch giving a 0.4-2 V analogue output for either 0-50 ppm or 0-500 ppm range via external socket. The selected analogue output is indicated by one of the two LED (Light Emitting Diode) lamps. A second external socket can be connected to a 'mine flash' visual alarm which flashes synchronously with the appropriate LED on the front of the instrument. Normal operation is indicated by the LED flashing once every 15 seconds and once every second if a preset alarm level is exceeded.

Though CO burns with a bluish flame and forms a cap over the flame of an oil safety lamp, the percentage of CO to produce a noticeable cap has to be high and entry of a person in an atmosphere having such high percentage is dangerous. Therefore detection of CO by a flame safety lamp is not recommended.

UPTRON India Ltd. is marketing Carbon monoxide monitor manufactured by Sieger Ltd. of England. There are 2 models : BCO-1 and BCO-1R. These models incorporate a sensor with an electrochemical cell. The units have the facility for mains or nickel cadmium rechargeable batteries, a 0.4 to 2 volt analogue output for transmission of data, an alarm relay output for 'Mineflash' operation or an output for driving an approved recorder. The BCO-1 has a built in sensor whilst the alternative BCO1-R unit offers the advantages of a remotely sited sensor and higher range. They are intended for fixed point monitoring and can be powered from an intrinsically safe flameproof mains operated power supply which trickle charges the internal batteries. In the event of an external mains failure the instrument will continue to operate from its own integral standby battery. The BCO-1 will continue to operate for 42 days ; however, the BCO1-R will operate for upto 5 days. A liquid crystal display gives a large, clear, digital readout in ppm and the same display is also used to inidcate 'over range' and 'fault' conditions. The instruments are both dust and water resistant.

Both the models have low cross sensitivity to other gases. The BCO-1 can be used as a transportable unit and operated from its internal batteries which can be recharged underground using a suitably approved power supply. A special feature of the BCO-1 is a built in system checking circuit which allows the operator to carry out rapidly a fault check of the unit using controls located on the front panel.

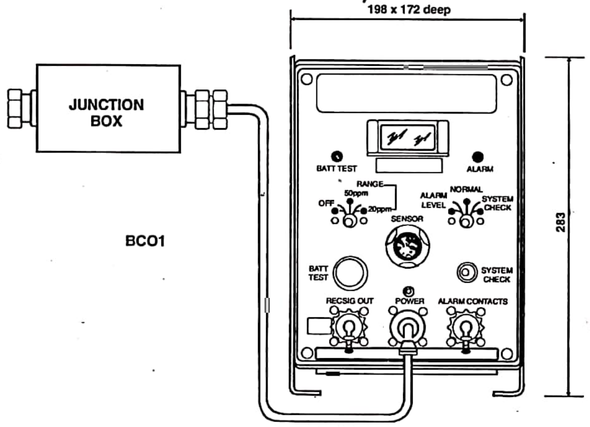

Fig. 1.7

Specifications of BCO-1 :

Range 0-200 ppm CO; Temperature range 0 to + 40°C; Humidity range 20-90% RH, continuous exposure 0-99% RH, non-condensing intermittent exposure; Input voltage 12 V DC, 18 V DC; Alarm range 10 to 200 ppm CO; Alarm indication - Flashing red LED and STATE relay contacts; once every 15 seconds indicates normal operation. Once every 2 seconds for first minute of alarm, increasing to twice per second after the first minute; Alarm relay single pole contacts rated 0.5A, 28V DC 10W non inductive.

Hydrogen sulphide or stinkdamp, H_2S :

This is produced when coal containing sulphur is heated out of contact with air to a temp. of about 444°C. Coal may contain sulphur in very small quantities in the form of pyrites or finely disseminated organic compounds of sulphur.

The gas is colourless and has the smell of rotten eggs. It has a sp. gr. of 1.75 and it is readily soluble in water, one vol. of water dissolving about three vol. of the gas. It is combustible but does not support combustion. When mixed with air it forms an explosive mixture, the limits of inflammability being 4.3 and 45% of H_2S. The gas burns in air with pale blue flame.

H_2S gas occurs ordinarily only in traces in relatively few coal and metal mines. It may be found in stagnant water in old workings, in areas of gob fires or spontaneous heating. It may issue from blowers or 'feeders' of gas along with CH_4 in a coal mine. In a metal mine it may be produced by the action of acidic waters on sulphide ores.

The gas is as poisonous as CO and it may cause death in a short time if inhaled in large quantities. The maximum permissible concentration in rooms for 8-hour exposure is given as 0.02%.

Presence of the gas can be detected by its disagreeable smell of rotten eggs. A blotting paper soaked in lead acetate changes its colour to black in the presence of the gas. A moist silver coin would also change to black in its presence due to formation of black sulphide on the surface. The gas can be detected by Drager multigas detector and M.S.A. hydrogen sulphide detector which resembles the M.S.A. CO detector. The H_2S detector tube contains white granules of activated aluminium oxide coated with silver cyanide which turns greyish black if exposed to H_2S. When making a test for detecting the gas the aspirator bulb is squeezed 10 times. The air sample is drawn in the detector tube and the extent of discolouration gives the percentage which is read off directly on the movable graduated scale.

C.M.R.S. has patented a H_2S detector tube which detects 1 to 50 ppm of H_2S.

Sulphur dioxde SO_2 :

It is a colourless gas with a strong sulphurous smell, neither combustible nor a supporter of combustion. It is 2.21 times heavier than air. The gas is very poisonous and extremely irritating to the eyes and respiratory passages.

It may be produced in small quantities during blasting in mines, and after a fire or coal dust explosion.

Nitrous fumes :

The fumes are a mixture of different oxides of nitrogen such as NO, NO_2, N_2O_3 and have a choking smell. They are yellow to reddish brown in colour and are easily dissovled by moisture in the mine air.

Nitrous fumes are formed during blasting of explosives containing nitroglycerine as one of the constituents if the explosive is not denotated completely. Exposure for a few minutes causes headache.

Hydrogen H_2 :

Hydrogen is the lightest gas known. It is odourless, colourless and tasteless, slightly soluble in water. It does not support combustion but it is itself combustible. When mixed with air it forms an explosive mixture, the limits of explosibility being 4 and 75% by volume.

The gas occurs rarely in mines but may sometimes be found in fire areas as a result of distillation of coal. Storage batteries of underground battery locos, when charged at the battery charging stations, evolve the gas in very small quantities.

The gas is not poisonous but suffocates a person due to lack of oxygen.

Methane, CH_4 :

This is commonly known as firedamp though, technically speaking the term firedamp refers to the mixture of gases emanating from the strata of a coal mine. Such mixture consists of practically methane with small traces of other gases like ethane, ethylene, etc. and therefore firedamp is often considered synonymous with methane. The gas is responsible for a number of gas explosions in coal mines. It is tasteless, colourless, odourless, lighter than air (sp. gr. 0.553); it is combustible and burns with a pale blue flame but does not support combustion. It is hardly soluble in water, 100 vols. of water at 20°C dissovling only 3.3 vol. of the gas. When mixed with air it forms an explosive mixture, the limits of explosibility being 5 and 15% by volume. The gas is not poisonous but suffocates a person due to lack of oxygen if present in large quantities. Critical temp. - 83°C.

Methane is a product of decaying of cellulose $(C_6H_{10}O_5)^n$ and is formed whenever vegetable matter decomposes under water and out of contact with air, as in marshes. Hence it is also called marsh gas. It forms part of coal seams and associated strata as coal has been formed millions of years ago in that manner. It is generally regarded that the gas remains in coal (a) partly in a state of mechanical imprisonment in small cavities, breaks and fissures, and (b) partly in a state of occlusion. As a rule firedamp is more prevalent in deep mines that in shallow ones and in some mines it may not be present at all. Methane is emitted not only in coal mines but is also found in rock salt, potash and clay mines. It has been found in a mica mine as stated in D.G.M.S. Circular No. 18 of 1961. The circular states 'Attention is drawn to an accident in a mica mine in which three workmen were seriously injured due to ignition of inflammable gas in a drive. Firedamp or marsh gas can be formed by the decomposition of timber or vegetable matter. Accumulation of such gas can thus be encountered in blind headings in mines which are being dewatered after being abandoned for sometime.

In a coal mine the gas may find its way in the workings in the following ways :

i. Gradual exudation or bleeding from the coal and adjacent strata in the roof and floor.

ii. In the form of a blower. This can be felt on the hand and may in some cases be heard.

iii. In the form of a gas outburst. Such outburst may sometimes be associated with violence.

iv. Release by roof fall or sudden fall of barometric pressure which may force the gases of the goaf into the workings.

The gas being lighter than air is usually found near the roof and in rise workings of a coal mine. In a badly ventilated roadway it may be found throughout the roadway due to diffusion of the gas in air. Fault planes and dykes are other places where the gas may be expected in a non gassy coal mine.

Factors affecting gas emission :

The important factors affecting gas emission are :

1. The nature of the coal seam and adjacent strata.
2. The method of mining and the type of coal cutting machines used.
3. Speed of advance of a longwall face or of the general production faces in board and pillar method.
4. The ventilation arrangement and their efficiency.

The rate of emission is more during a coal cutting shift on a long wall face.

Methane Layering :

Some years ago, first of all in Britain, attention was turned to the fairly stable layers of methane occuring at the roof level for some tens of metres. Examination of the evidence of a number of gas explosions showed that they were caused by the ignition of these layers. The circulars of D. G. M. S. have also drawn the attention of the mining industry to the phenomenon of 'roof layering' in the workings and the dangers arising therefrom. In one case

it was revealed that nearly 38% gas had accumulated at the roof level at a working face but its concentraion was only 1% at 0.3 m below the roof. A layer of gas is also known to travel along the roof against an air current for an appreciable distance away from its source. (D. G. M. S. Cir 48 of 1959). In another instance the following accumulations of inflammable gas were obtained in the roadways :

Place	At roof level	75 mm below the roof	150 mm below the roof	225 mm below the roof
1. The face of a dip gallery	10%	–	–	1.5%
2. The face of a dip gallery	10%	–	–	1.5%
3. The face of a dip gallery	10%	8%	0.5%	–
4. A level gallery	3-10%	–	–	0.5%
5. A level gallery	10%	8%	3%	2.5%

The readings above are typical of the conditions occuring in a gassy mine where the ventilating current is not turbulent. (D. G. M. S. Cir. No. 7 of 1964)

One practical method to deal with layering is to increase the velocity of ventilation air current. Bafflers across the roadway may have to be temporarily erected to diffuse the layer expeditiously.

Degree of gassiness of a coal mine :

The degree of gassiness of a coal seam is laid down by a Circular of the D. G. M. S. in 1967 and the coal mines are classifed into the following three categories.

Gassiness degree	% of inflammable gas in general body of air	Rate of emission of gas m^3/te of coal raised
I	< 0.1 and	< 1
II	> 0.1 and / or	1 – 10
III		> 10

In Russia, the coal mines are classified according to their gas emission in one of the following categories.

Category 1 – less than 5 m^3/tonne of daily output

Category 2 – from 5 to 10 m^3/tonne ... do ...

Category 3 – from 10 to 15 m^3/tonne ... do ...

Mines with more than 15 m^3/tonne of daily output are considered to be outside their categories.

GAS BLOWERS :

A gas blower is a powerful irruption of gas in the form of jets from cracks or coal faces. It is accompained by hissing sound which can be heard over a long distnace in the calm of an underground mine. The gas from a blower is generally pure methane. A blower may last only a few days or months and in some cases even for years but the gas emission is usually heavy in the initial stages and gradually goes down till it is finally exhausted. Gas from permanent blowers can be coursed to the surface and utilised for scientific or industrial purposes, for power generation, lighting or heating.

A well known example of a blower was in a British coal mine of Garwood. The blower of very long duration was met in a sinking pit and the gas was led to the surface and burnt for nine years in a flare so high that the flame could be seen some 15 km away. Another example of a gas blower of very long duration is that of Cymmer colliery in South Wales (U. K.) which has been feeding gas, 97.5% methane, at the rate of 25 m^3/hr for upwards of 70 years. The blower was met with during the sinking of the shafts and was still blowing long after mining operations at the colliery had ceased.

Blowers are usually met in areas which have been faulted or folded. In coal seams which are prone to gas blowers underground roadways should be advanced with advance exploratory boreholes using special drilling machines. If a blower is met in a mine it can be dealt with by

i. Closing or blocking the crack from which the blower emanates.

ii. Allowing it to exhaust naturally in course of time if it is not very large. Ventilation of the district has, however, to be improved to keep the percentage of gas in the general body of the return air within permissible limits.

iii. Coursing the gas of blower to the surface by special pipes.

Detection and testing of firedamp :

Mine air is tested for methane by the following three methods

(a) Chemical analysis of mine air samples in laboratories at the surface.

(b) Testing underground by means of flame safety lamps.

(c) By the use of special instruments to directly indicate methane percentage content in the air by volume.

For the chemical analysis of mine air samples in laboratories, the air sample is transferred to an analysing apparatus where the methane is burnt and the products of combustion are selectively absorbed by absorption reagents. The technique is time consuming and requires much effort on the part of the chemist.

On the spot detection of methane and determining its percentage in the mine air is carried out by methane detectors. As firedamp forms an explosive mixture with air when present in percentages from 5 to 15, it is essential to detect the gas well before it reaches the lower limit of inflammability. The easiest and simplest method to detect its presence and percentage is by means of a flame safety lamp. But there are other gas detectors too. If methane layering has taken place it can be measured by methanometers only and the flame safety lamp serves no useful purpose in a thick seam.

The methane detectors work on one of the following principles.

1. Formation of a gas cap of a variable height over the flame of a flame safety lamp. This is the principle behind the use of flame safety lamps and spiralarm as gas detectors.
2. Wheatstone bridge principle of measuring the unbalanced electric current when resistnace of one circuit of the bridge increases due to reaction with fire damp. This is the principle governing D-6 (M.S.A.) and many other methanometers. A BMI methane monitor developed by MRDE also works on the same principle.
3. Diffusion – combustion – contraction. When methane burns in air, it produces carbon di oxide and water vapour.

$$CH_4 + 2O_2 = CO_2 + 2H_2O \text{ (water vapour)}$$

1 vol. + 2 vol. = 1 vol. + negligible volume as water vapour condenses

The products of combustion result in contraction of the space equal to 2/3rd of the space originally occupied by methane and oxygen of the air. Such contraction is utilised in the Ringrose firedamp detector.

4. Difference in refractive index of pure air and methane air mixture and its effect on the interference pattern of light. This is the principle behind Riken Interferometer and similar instruments.

5. Miscellaneous principle as in infra-red recorder. This is based on infra-red spectrometry which utilises the principle that methane absorbs radiation of wave length 7.5μ.

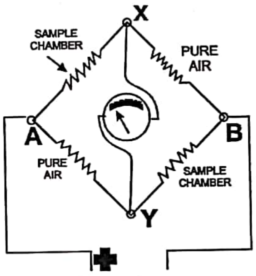

Fig. 1.8 Principle of wheatstone bridge

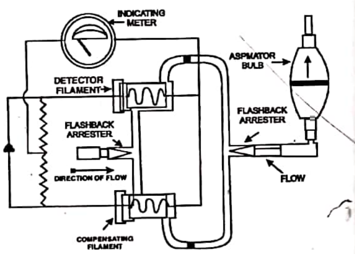

Fig. 1.9 Simplified electrical circuit diagram illustrating principle of the M. S. A. methanometer

6. Reaction of methane with some chemicals. This is the principle behind detection of methane by detector tubes manufactured by the firm of Drager.

The constructional details of flame safety lamps and their use as firedamp detectors are described in the chapter on mine lighting.

The portable methanometers like Reyrolle Burn methanometer, M. S. A. methanometer, Oldham methanometer, and others work on the principle of wheatstone bridge, illustrated in Fig. 1.8. The network of Wheatstone bridge consists essentially of four identical platinum - iridium spiral wires or filaments arranged as shown (fig. 1.8). When the points A and B are connected to a source of direct electrical current, like a battery, the voltage drop across AX and AY is equal so that if X and Y are connected by a conductor, no current can flow through the latter. If, however, resistance of one of the spiral wires is changed, the voltage at points X and Y are differnet and current flows through the wire connecting X and Y. This current is measurable by a galvanometer.

In a methanometer one pair of spirals diametrically opposite to each other in the network, is contained in a small cylindrical aluminium chamber which contains pure dry air and is hermetically sealed. The other pair of spirals is contained in a second chamber which can be filled with mine air sample to tested. A milliammeter is connected diagonally in circuit between points X and Y. If air sample containing methane is admitted in the sample chamber and made to react on the chemical coating on the filament after passing current through the circuit rendering the wires white hot, the heat generated changes the resistance of wires in the chambers. The voltage at X differs from that at Y resulting in passage of electric current through the milliammeter which is graduated in percentages of methane as the temperature rises and consequently the resistance of the spiral wires in the sample chamber is dependent on the percentage of methane in the air sample.

M. S. A. D-6 Methanometer : (2008) (2008)

One of the methanometers commonly used in our coal mines is the D-6 methanometer of M.S.A. It is portable, handheld and suitable for spot checking of gas accumulation with measurement range from 0 to 5% by volume. A telescopic tube (like the telescopic aerial of a small T.V. set) is provided on the meter for introducing into it gas sample near the roof or other in accessible points. The 2-section telescopic probe extends from 760 mm to 1.42m. Rechargeable batteries are used in the methanometer. The power pack consists of 2 Nos. nickel cadmium batteries in series and provide 2.2 to 2.6 V for operation. Capacity of the power pack is 1.2 A. H. Voltage of the battery should be checked before methane check. The voltage must be between 2.2 and 2.8. If the voltage is below 2.2, do not check for methane as the reading is likely to be faulty.

A fully charged battery lasts for nearly 1000 ten-sec. readings ries can be charged from a battery charger connected to an A.C. electric supply of 250 volt, 50 cycles and the complete charging takes about 14 hours. The condition of the batteries in the meter can be checked by pressing a button on the right hand side and the voltage is recorded on the scale of the meter. (Fig. 1.10)

The methanometer works on the principle of Wheatstone Bridge and diffusion of gases. At the spot of detection when the plastic dust cover at the top is opened methane air mixture enters the sampling chamber of the meter.

The sampling chamber contains a pair of treated pelletised filaments which form half of the Wheatstone bridge circuit. One filament is active and the other deactivated but its presence in the gas chamber assists stability. After the methane air mixture finds its way in the meter a push button on the left hand side is pressed. This results in heating of the gas on the active filament, thereby creating an electrical imbalance of the Wheatstone bridge and this is indicated on the meter as methane percentage. The whole operation of detecting gas percentage is over in 1.5 minutes.

The instructions of operation for D-6 methanometer are :

(a) Before taking the instrument underground for methane checking, check the voltage of the battery. It must be between 2.2 and 2.8. For this, hold the instrument in the left palm with the thumb resting on the larger methane check button and the middle finger on the smaller voltage check button. Press the voltage check button. Red mark on the dial of the meter shows correct voltage range.

(b) Underground, at the spot of methane checking; lift the diffusion head cover so that the diffusion head is fully exposed to the atmosphere. For checking methane percentage in the atmosphere (and not of inaccessible place near the roof), hold the instrument in the left palm and press the methane check button on the left side. The methane percentage can be directly read on the meter. Do not keep the methane check button pressed longer than it takes to read the methane percentage. The normal duration is not more than 20 seconds and for an experienced operator, only 10 seconds. Keeping the button pressed beyond this period unnecessarily drains the battery.

(c) For checking the methane percentage of inaccessible place with 1.5 m of the instrument near the roof, use remote sampling telescopic probe. For this purpose flip open the dust cover over the diffusion head. Unscrew the probe cap and screw the threaded socket of the probe (lower end) firmly to the diffusion head with the probe end in place, squeeze the aspirator bulb 2-3 squeezes are enough for a constant reading when the methane check button is pressed.

The methanometer needs no maintenance except periodical battery charging. Its weight, including the battery is 470 grams.

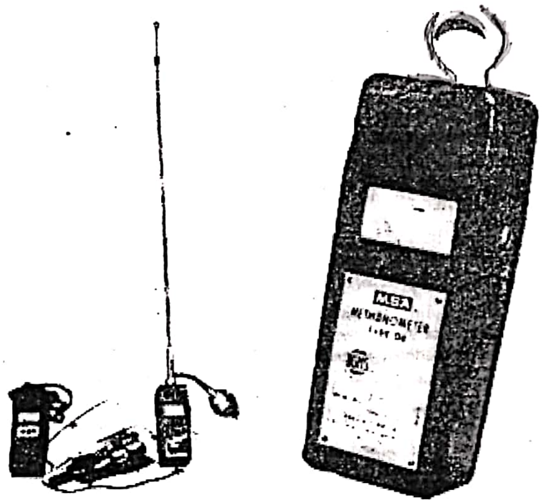

Fig. 1.10 MSA D-6 methanometer (Right) The same with telescopic probe normally used for higher sports

The methanometer and in fact, any other methanometer should not be used for detection of methane at a place where its percentage is expected to be high, e.g. in the afterdamp after an explosion. To know the percentage of methane in afterdamp the gas samples should be collected and analysed on the surface in a laboratory.

Caution :

i. Do not open the methanometer in mines even for replacing the batteries.

ii. If the methanometer is not to be used for some time, store it in dust free, and cool place after taking out batteries.

Automatic firedamp detectors :

The automatic firedamp detector (AFD) which is manufactured by the firm of M.S.A. is a portable instrument designed for continuous monitoring of CH_4. It is used with a readily chargeable battery giving a ten hour operating period. No external switch is required and the instrument functions immediately upon insertion of the battery. The detector is scaled to read 0-3% of methane in a mine and under normal conditions the lamp flahses at 15 sec. Intervals. When the methane concentration exceeds the preset level, the flash rating will increase to once per second. This alarm setting can be easily adjusted for any percentages over the full scale of the instrument.

The instrument gives a continuous indication of methane concentration by meter reading and flash lamp. It can be either carried on the person or suspended at the work place.

The A.F.D. operates on the principle of Wheat-stone bridge in the same manner as the D-6 Methanometer. It is put into operation using the following procedure.

i. Install battery and seal the instrument

ii. Allow 15 mts. for adjustment.

After this period lamp flashes at 15 seconds intervals.

iii. Check the alarm level using MSA calibration unit.

iv. The meter reading should rise at the preset level and the lamp should flash at the preset level at one second interval (MSA calibrator unti should be used for preshift instrument checking)

The instrument is now ready for use.

The weight of the instrument is 1070 grams without battery and 1500 grams complete with battery.

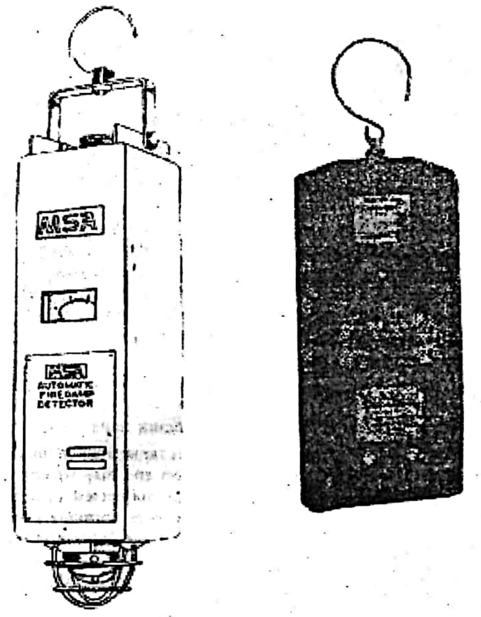

Fig. 1.11 Automatic firedamp detector

Uptron India Ltd. is marketing an alarm methanometer Model AFD-2, manufactured by Sieger Ltd. of England. It is a continuously operating alarm methanometer designed to protect underground mines against dangerous levels of methane gas in the working environment. It has both a flashing light alarm and digital display of methane level within the range 0-5% v/v methane. The instrument is intrinsically safe. A fully charged battery operates it for 15 hours.

The sensor is a diffusion type, low powered electro catalytic pellistor. The sensor housing, including sintered flash back arrestor, carbon cloth filter, etc. is one complete replacement unit. Under normal no-fault operating conditions the visual alarm flashes at a slow rate of 1 flash/15 secs. But at the pre set alarm level (usually 1.25% v/v methane) this changes to an immediately eye-catching signal rate of 1 flash/sec.

Specifications :

Range - 0 to 5% v/v CH_4 ; Alarm - adjustable 0.5 to 5% v/v CH_4 ; Alarm Signals - Normal LED's on. Flash once every 15 secs; Fault - Lamps off. No. LCD display but decimal point of display is on; Alarm - LED's flash once every second ; Low Battery - Letter 'L' appears on LCD display ; Over Range ; Letter '0' appears on LCD display and alarm is latched 'on'. Operating Temperature Range 0 to + 40°C ; Operating Humidity Range - 0 - 100% RH ; Warmup Time - 5 min. Operating time - 15 hrs. with fully charged battery ; DC supply - 2.3 V to 2.8 V ; Weight - 1.4 kg Dimensions - AFD2 (less carrying hook) 310mm × 117mm × 65 mm.

Methane Monitor, MEMACS I :

M. S. A. is marketing a Methane Monitor called MEMACS I.

MEMACS I will detect methane in the range 0-5% in the general body of air by means of catalytic pellistores and directly control an associated flame proof relay which may be connected to a control circuit for electrical power supply. The relay is normally energised and when methane exceeds a pre-set level the relay is deenergised by the MEMACS I. The present level is internally adjustable and a light emitting diode (Hi-alarm) on the front of the MEMACS I is lit when the relay is deenergised thus indicating the cause of the change of state of the control circuits connected to the relay contact. In addition a warning alarm is provided which would normally be set to operate at a lower methane concentration than the control relay. This is indicated by an LED (low alarm) on the front panel which under normal conditions flashes once every 15 seconds and in alarm conditions flashes once every second. A continuous reading of the measured level of methane is provided on a liquid crystal display and an analogue output signal of 0.4 to 2.0 V DC complying with B. S. 5754 and representing 0-5% CH_4 is available for connection to a data transmission system.

FAIL-SAFE features are provided such that if the cable to the relay is either short circutied or open circuited, or if the analogue signal is less than 0.2 V, then the relay is de-energised. Additional fault condition is provided as follows :

1. **Overrange latch :** This holds reading, shows 'FS' on LCD and the yellow LED front panel is lit. Latch cancelled by magnet placed against 'RESET' label on front panel.

2. **Low battery :** When battery voltage is below 215 V 'Lo Bat' appears in the display, analog output is off, multiflash output is off, pilot relay is dropped out and Hi - alarm LED is on.

3. **Filament failure :** Filament short circuit or open circuit indication-analog output is switched off. In addition on open circuit 'FS' is on display and on short circuit is not, multiflash is off. Pilot circuit output is de-energised and pilot circuit LED is on (Hi-alarm).

4. **Low signal :** If analogue output drops below 0.2 V multiflash is de-activated and Lo-alarm LED ceases to flash, pilot circuit drops ont, and Hi-alarm LED is on.

Dimensions : - Height : 220 mm; Width : 120 mm; Depth : 90 mm;

Fig. 1.12 Methane monitor MEMACS I

1. Lo alarm LED (Multiflash Level). 2 Hi-Alarm LED (Pilot Switch). 3. Power supply input (12 VDC/DC2). 4. Liquid crystal display. 5. Integral standby battery. 6. DC3 + BAT/can be used as power supply to provide extended standby capacity. 7. Over/range LED. 8. Analogue/Multiflash output. 9. O/R reset switch. 10. Pilot switch out t.

Interference Methanometer :

Interference Methanometers (also called interferometers) are used in some of out mines for finding out percentage of methane on the spot in underground mines. The common example is Riken Interferometer manufactured in Japan. An Interferometer utilises the optical properties of methane air mixture and works on the principle that refractive index of methane being higher than that of air, the presence of any quantity of methane in air increases its refractive index prportionately. Light forms interference pattern which shifts laterally if it is allowed to pass through two cells, one containing pure air, and the other, a mixture of air and methane. This shift is proportional to the quantity of methane present in the mixture and the percentage can be measured on a scale calibrated in percentage of methane.

Fig. 1.13 shows the schematic diagram of the instrument which consists of a pocket torch type bulb which gets power from a dry cell. A lens is used to give a parallel beam of light which can be separated into two beams with the help of a glass slab. One beam passes through the measuring cell and the other, through the comparison cell. The two light paths are geometrically equal and also optically equal if the gases in the two cells have the same refractive index. An interference band is obtained by slightly tilting the prism and it can be projected on a scale (calibrated in percentage of methane) which is situated in the focal plane of telescopic objective lens.

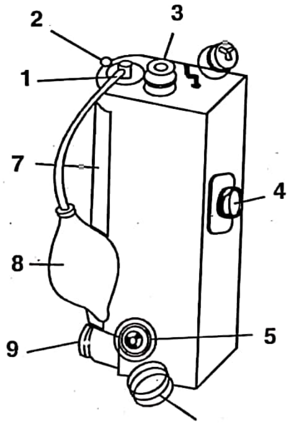

Fig. 1.13 Riken interferometer

1. Outlet port to which aspirator bulb is connected.
2. Inlet port to which sampling tube is connected.
3. Eyepiece lens.
4. Push button to illuminate methane scale.
5. Zero adjusting knob.
6. Screw on cover to protect zero setting.
7. Snap on lid covers replaceable moisture, CO_2 absorbent cartridge and dry cell.
8. Aspirator bulb.
9. Push on cap to electric bulb-compartment and illumination forcussing device.

Portable Interferometer Type Methane Gas Indicator :

A portable optical methanometer has been designed and developed jointly by Central Mining Research Station, Dhanbad (CMRS) and Central Scientfic Instruments Organisation, Chandigarh (CSIO) for use in coal mines and other industry. This is a modified design of the Japanese make. It is highly sensitive and consistently precision instrument which is indigenous in nature and has been developed for the first time in the country.

To measure the methane content in methane air mixture in coal mines and other industries, its scale offers a range of 0-10% methane by volume. The main scale is graduated in 0.25% intervals from 0 to 2%, 0.5% in the range of 2 to 10%, and the vernier drum is graduated in 0.02% intervals from 0.00 to 1.00%.

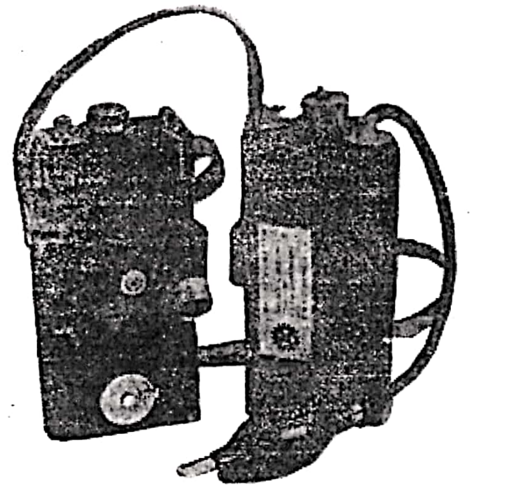

Fig.

The instrument is based on the principle of difference of refractive indices of methane air mixture and dry air, employing the phenomena of interference of light. The system of interference frings containing two black lines (fringes) is produced as in Jamin's parallel plate interferometer and these can be viewed through an eyepiece on a well illuminated scale.

Only one dry cell of 1.5 volts is enclosed in the instrument to operate the two bulbs (lamps) thorough micro switches during use.

The *optical system* comprises a lamp housing, optically plane parallel beam splitting glass plate, lenses and slit, prism, zero adjusting prism, objective lens, microscale and the eye pieces. The gas flow system consists of reagent tube, gas chambers (gas cells) and anti diffusion coiled tube and an aspirator bulb at the outlet. The eye-piece, inlet and outlet with the aspirator bulb project outside the leather case of the instrument. The overall weight is about 1 kg.

When the methane air mixture is introduced in the gas chamber by operating the rubber aspirator, fringe patterns are shifted. The shift in black lines is due to the path difference introduced between the two interfering beams produced, and is directly proportional to the methane percentage present in the air in the gassy mines. This shift can be read on the calibrated scale.

The readings are independent of the voltage of the battery used and can be retained for some time in the instrument for rechecking and records. It is powered by one 1.5 V dry cell, and it can be used safely in any explosive atmosphere. It is simple to operate, and percentage of methane gas is quickly determined in steps of 0.02% for the whole range.

As there is an increasing tendency in the coal mining industry to use more sophisticated electronic methanometers which give digital display of gas percentage practically instantaneously, the interference methanometers will not be described here in more details.

Ringrose Firedamp Detector :

An apparatus known as the Ringrose firedamp detector works on the principle of diffusion combustion contraction which has been explained earlier. The apparatus, once used in our coal mines, has practically gone out of use as the filament in it has to be replaced every shift and the filaments have to be imported from the manufactures abroad. The apparatus will therefore not be described here.

Methanometers based on Spectrometry for gas analysis :

Optical spectrum can be divided into ultra violet ultra violet (0.25 - 0.40 μ), visible band (0.40 - 0.70 μ), near infra-red (0.70 - 2.50 μ), middle infra-red (2.5 - 25 μ), far infra - red (25 - 500 μ). Among these, the middle infra-red region has a special significance for detection of methane and other gases. The infra red spectrometer is widely used for gas analysis and is capable of distinguishing individual components from complex mixtures. The principle of operation is that each gas or vapour absorbs infra radiation at particular wave length. The absorption of IR radiation by the methane molecules increases the temperature of the gas, which causes pressure rise of the enclosed gas. A pneumatic detector then works by sensing this variation in pressure.

Spectrometer is used these days on an increasing scale for gas analysis. The mass spectrometer, a very useful and accurate instrument for gas analysis, can handle as many as 20 components. However, it is very expensive and it cannot be justified economically where only a few analyses per day are required.

Methane Telemonitoring and Sensors :

An electronic methane telemonitoring system for the continuous, automatic measurement and record of methane at a preselected point in an underground mine had been developed by Bharat Bijlee Ltd., Bombay after getting the technical know how on the methane sensor from C.M.R.S., Dhanbad. The system provides for continuous, automatic monitoring of methane without any human agency, from an underground selected point and the methane percentage was recorded continuously on a chart in the room at the surface. When the methane percentage exceeded a preset level, an alarm signal used to be given by a bell at the underground substation or other desired place and arrangements were made for simultaneous alarm at a suitable point on the surface as well. There was also a provision for shutting down the electrically operated equipment when the methane percentage used to be more than the preset level.

The methane telemonitoring system consists of two sets of equipment.

(a) Underground, i.e. remote station equipment and

(b) Surface station equipment.

The electronic methae telemonitoring system was installed in Sudmdih, Monidih and Chinakuri collieries. The installations were of single point system, i.e. the methane percentage from only one point could be recorded but multipoint systems which record the methane percentage from a number of selected points (upto five) had been produced by Bharat Bijlee Ltd.

The sensor system should be installed away from dust producing centres. Otherwise they give erratic results as observed from the installation in one of the mines.

In some advanced forcign countries sensors have been developed which have multiprobes, which, through the multichannel electrical system of the sensor unit, indicate to the Manager at any time the complete picture of ventilation efficiency of the underground mine by atomatically recording (1) methane percentage (2) ventilation pressure (3) temperature (4) humidity (5) quantity of air flowing (6) CO percentage.

One temperature humidity monitor manufactured by Sieger Ltd., England and marketed by Uptron Ltd., (Model TH-1) is described in the next chapter.

Methane Drainage :

If a mine is very gassy it may not be possible to keep the percentage of gas in the return air below the limits permitted by mining regulations even by a high standard of ventilation employing high powered fans. Drainage of the gas out of the mine to the surface is a great advantage in such cases. It results in (i) less firedamp in the ventilating system of the mine rendering the latter safe (ii) the main ventilating fan need not be very large (iii) rate of advance of coalface can be increased or alternatively, longercoalfaces can be adopted (iv) the gas can be utilised at the surface for power generation by burning it in specially designed boilers. Methane is a rich source of fuel. One kg of methane in burning evolves 13,600 kcal of heat, whereas 1 kg of gunpowder releases only 580 kcal and 1 kg of nitroglycerine gives out 1500 kcal.

Methane drainage was first tried in a colliery in the Ruhr coalfied of Germany in 1943 and since then it has been successfully adopted in some foreign countries. In India it is being experimented at Amlabad colliery and the reason for not trying it on a large scale is that out mines are not very deep and not very gassy so that they can be well ventilated by powerful fans.

At Amlabad colliery the make up of gas over a period of seven years was on the average as follows in the sixties.

XVII seam (depth 225 m) – $35 m^3$/te of coal raised

XVI seam (depth 252 m) – $70 m^3$/te of coal raised

XIV seam (depth 425 m) – 200 to 600 m^3/te of coal raised

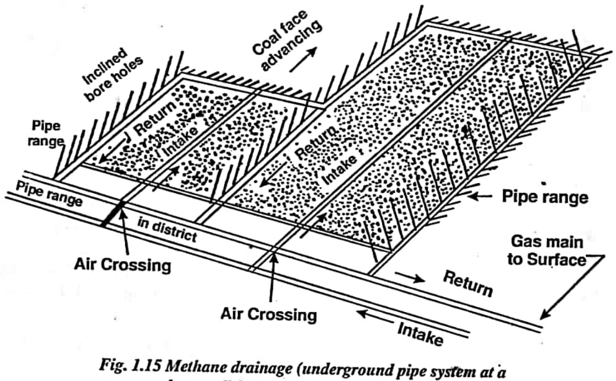

Fig. 1.15 Methane drainage (underground pipe system at a long wall face and layout of bore holes)

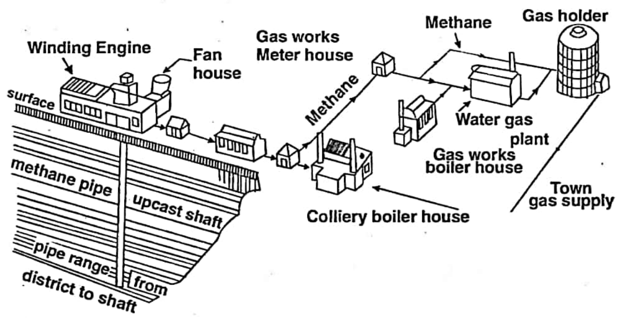

Fig. 1.16 Methane drainage; schematic layout at the surface

The make up of gas in XVI and XVII seams in terms of m^3/min has been of the order of 12 to 20, but in XIV seam opened in 1970, the make up of gas has varied between 15 and $30 m^3$/min in the small workings around the pits.

- Desgassification during methane drainage can be conducted from

(a) the seam being worked, or

(b) the seam above or below the seam being worked, depending upon accessibility.

Methane drainage of neighbouring seams is the technique commonly adopted. The aim is to place one or more boreholes into the strata which are presummed to be a natural reservoir of gas by reason of their porosity or contact with carbonaceous matter like coal or carbonaceous shales. The boreholes are 45° to 60° off vertical leaning toward the inbye, 37 to 75 mm in dia. and 20 to 60 m apart. They are drilled in the roof from the intake or return airway until the upper seams are intersected. Hole lengths of 80 m are not uncommon. All the holes are equipped with water separators and devices for measuring the pressure and quantity of gas flowing. The gas is removed from the boreholes by suction using rotary vacuum pumps installed on the surface.

Fig. 1.15 shows the layout of boreholes for methae drainage in a mine with longwall working. Gas is conveyed first through 75 mm dia. pipes and the diameter is progressively increased to 150 mm and then to 300 mm. Long boreholes yield generally gas with high methane percentage. Air should not be succked inside the pipe. The flow from a borehole varies widely from 0.5 to $13 m^3$/per minute and the period during which gas is available may be 5-10 months. In Russia a gas emission of 20-$25 m^3$ per tonne of coal mined is considered to be the level at which methane drainage should be considered desirable.

Where methane drainage is practised the safety precautions in the mine and on the surface have to be elaborated and there should be no leakage of gas into the atmosphere or of air inside the pipes.

In British mines methane drainage is gradually on the increase. The proportion of coal mines draining methane has changed from about 0.17% in 1949 to about 60% (105 mines) in 1986 and the trend is continuing.

Determination of gas percentage (CMR 145) :

1. If electricity is used in a mine, in deg. 2 and deg. 3 gassy mine, gas percentage in general body of air shall be decided once in a week by taking air samples.
2. If any sample shows gas more than 0.8% sample should be taken daily.
3. If any sample shows gas less than 0.6% gas sample may be collected only once in a month.

Cutting of electric supply (CMR 145) when gas in any ventilating district is more than 1.25% cut off electric supply to all apparatuses, lights and cables in that district.

Radon gas and its daughter products are commonly found in mines of radioactive minerals such as Jaduguda Uranium mine. Their concentration should not be allowed to exceed 3700 s^{-1} m^{-3} for prolonged exposure.

QUESTIONS

1. Give the physiological effects of the following gases on human beings.
   (a) Nitrogen (b) CO (c) CO_2 (d) CH_4
2. Describe the methods of detecting carbon monoxide in an underground mine.
3. What are the various principles behind the detection of firedamp and its percentage in a mine.
4. Describe a methanometer working on the principle of wheatstone bridge.
5. Write notes on :
   (a) Gas blowers (b) Gas outbursts
   (c) Methane drainage (d) Methane layering

O O O

CHAPTER 2

UNDERGROUND ENVIRONMENT & VENTILATION

A worker in a mine should be able to work under conditions which are safe and healthy for his body. At the same time the environmental conditions should be such as will not impair his working efficiency. This is possible if the mine air is nearly the same as on the surface and without toxic and inflammable gases. The humidity, temperatures and dust content of the air in the mine should also be within certain limits. This is achieved by proper ventilation of underground mine working by continuous supply of fresh surface air. As explained in the earlier chapter, the oxygen content of air is reduced for various reasons and it has to be replenished. The ventilation of a mine has therefore the following objects.

1. To restore the proper composition of mine atmosphere which should not contain less than 19% of oxygen or more than 0.5% of carbon-di-oxide.
2. To dilute other noxious and inflammable gases like CO, CH_4 etc. so that they are harmless. A place is not considered to be free from firedamp if gas percentage is above 1.25.
3. To provide good environmental conditions and to prevent excessive rise of temperature and humidity so that the workmen can work with maximum efficiency. The wet bulb temperature in development faces should not exceed 33.5°C.
4. To remove or dissipate the coal or rock dust produced in the mine.

Quantity of Air :

An idea of the approximate quantity of air required for ventilation of a mine can be had from the following empirical rules.

In gassy coal mines of Cat. II & III the major consideration to decide the quantity of air going underground is the rate of emission of gas (firedamp) which should be so diluted by the ventilating air that its percentage is not more than 0.5 in the main return airway of the mine. With this object some

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(A) Quantity of air in a ventilating district should be

i. Minimum 6 m^3/min. per person employed in the district on the largest shift.

ii. More than 2.5 m^3/min per daily tonne output.

(B) Underground air should not have less than 19% O_2.

(C) Underground air shold not have more than 0.5% CO_2 or other noxious gases.

(D) Inflammable gas should be below 0.75% in the general body of return air of any ventilation district and below 1.25% at any place in the mine.

(E) Wet bulb temperature should not exceed 33.5°C; if it exceeds 30.5°C at any place, air current should be fasterthan 1 m/s.

Air samples and temperature readings should be taken once in 30 days.

In gassy coal mines of Deg. II & III the quantity of air should be more than the minimum stated above and a reasonable figure should be 8 m^3/min per person employed.

The quantities of air stated above must reach the working faces. As there is leakage from intake to return at the ventilation stoppings, ventilation doors, air crossings, ventilation air locks on the surface and other places, more air should pass the downcast shaft/incline. Therefore, the quantity of air going down a mine should be as follows (per minute).

Gassiness degree (coal mines)	Per person in U/G mine	Per tonne of daily output
I	7m^3	3m^3
II & III	8 – 10m^3	4 – 5m^3

In metal mines which are not deep, say upto 300 m, the quantity of air that should go down the intake shaft/incline shoule be 4 – 5m^3 per min. for every worker in the underground mine. In deep mines like those in the Kolar Gold Field where the strata temperature is high, this quantity is considered adequate to keep the mine air reasonably cool.

The instruments which are helpful for proper control of mine ventilation, to know its adequacy and assess the environmental conditions for human work are :

1. *Thermometers*
2. *Barometers*
3. *Hygrometers*
4. *Kata thermometers*
5. *Air velocity meters*
6. *Water gauge for mine fans*
7. *Gas detectors*

Thermometers :

These are too well known to the readers to justify space in this book. Temperatures are often stated in degrees Farenheit or centigrade and the conversion ratio is

$$\frac{C}{5} = \frac{F - 32}{9}$$

Where C denotes temperature in degrees centigrade and F denotes temperature in degrees Farenheit.

Absolute Zero : This is theoretically the lowest possible temperature and is the point at which all heat would be extracted from a substance if it can be cooled sufficiently. The absolute zero temperature is – 273.15°C or – 459.63°F. For calculation purposes these figures are taken as – 273°C and – 460°F.

Barometer :

It is an instrument to measure the atmospheric pressure. The standard atmospheric pressure, also called the mean or normal atmospheric pressure, is defined as that pressure which supports a column of mercury 760mm high at the sea level when the temperature of the mercury is 0°C.

The Fortin Barometer : This is the standard form of barometer and is of refined construction, being used in scientific work for accurate measurement of the atmospheric pressure.

It consists of a straight glass tube about 920mm long and about 8mm inside diameter, the upper end being sealed and the lower (open) end dipping into a small boxwood cistern of mercury having a soft chamois leather base.

An adjusting screw and plate beneath the flexible bottom enable the level of mercury in the cistern to be adjusted, (Fig. 2.1) and, when the instrument is to be moved, the plate may be raised until the lower end of the tube is sealed.

Before any reading is taken, the level of the mercury must be adjusted so that its surface just touches the tip of an ivory pointer P. This represents the level from which the height of the mercury column must be measured. Alongside the tube near the top, is mounted a scale and vernier, which enables a reading to be made to the nearest half millimetre..

The aneroid barometer : This instrument is much more portable and convenient than a mercury barometer for determining differences of level, and is much used by prospectors and also in certain types of ventilation surveys (Fig. 2.2). the term aneroid means that the instrument does not contain liquid. Its construction is based on a different principle of measuring atmospheric pressure. According to the Boyle's law a given mass of gas increases in volume as its pressure falls and the volume decreases as the pressure rises, temperature remaining the same. The aneroid barometer makes use of this principle.

It consists of a hollow, air tight, gas filled box made of thin springy corrugated metal and prevented from collapsing by means of a strong flat spring. The expansion and contraction movement of the cylinder is transmitted and magnified through a system of levers and links to a pointer or a scale which reads the atmospheric pressure. The scale is graduated to read pressure from 700 to 780mm of mercury.

The instrument, of course, must be calibrated (i.e. tested by comparison and the graduations marked) with a mercury barometer from time to time. When subjected to sudden alterations of pressure, it is liable to give inaccurate readings, and time must always be allowed for it to adjust itself to the new conditions before a reading is taken.

The barogarph or recording barometer consists of a series of thin exhausted corrugated metal shells acting together, thereby increasing the sensitivity of the barometer so that its movement can be used to operate a pen marker and give a continuous record of barometric pressure.

The barometric pressure increases with depth below the sea level and decreases with height above the sea level at a rate of nearly 1mm Hg difference for every 12m vertical difference.

The barometric pressure has a bearing on the safety in a mine. The noxious and inflammable gases like methane in the goaf of a coal mine expand when there is a sudden drop in the barometric pressure and overlfow into the mine workings. If the influx of such gases into the workings is excessive the normal ventilation current may not be able to dilute them sufficiently, thereby resulting in high percentage of the gases in the mine air. If the barometer is steady or rising, there is no overflow and no danger of fouling of the mine ventilating air current. D. G. M. S. Circular No. 84 of 1966 directs that barometers should be provided at least at every first class mine in which depillaring is done and/or which has sealed off underground workings.

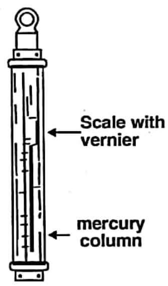
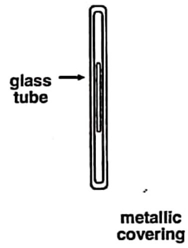
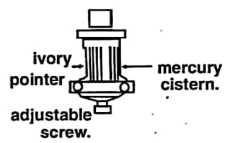


Fig. 21. Fortin barometer.





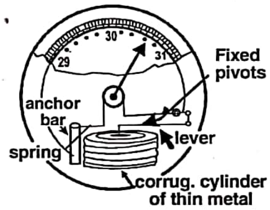
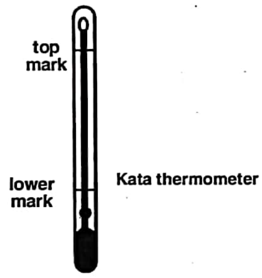


Fig. 2.2. Aneroid barometer.



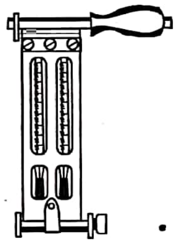


Fig. 2.3

Whirling hygrometer of N.M.C.

Hygrometer and relative humidity :

The atmospheric air has the capacity to absorb moisture and air containing it is called humid air. The atmospheric air, however, can absorb moisture only upto certain limit and air which is fully saturated with the moisture content that it is capable of absorbing, is known as saturated air. A given volume of moist air is lighter than the same volume of dry air. Under normal conditions air is never completely dry and the extent to which it is humid, is known as relative humidity.

$$\text{Relative humidity} = \frac{\text{mass of water vapour per m}^3 \text{ of air}}{\text{mass of water vapour required to saturate one m}^3 \text{ of air}}$$

The relative humidty of saturated air is 100%. The percentage of saturation, except in arid regions, is rarely less than 35% and is higher near the sea shore. In places like Calcutta and Bombay which are at the sea shore the relative humidity is maximum 96% during the rainy season on a hot day and the average R. H. on such day is nearly 80%.

The quantity of moisture present in air is chiefly dependent on temperature and pressure, other things being equal.

When a man works, his body temperature goes up and the body perspires. Normal human body temperature is 98°F (36.6°C) and a person feels discomfort if the body temperature goes up or below this figure by 1°C. During work if the perspiration given out by human body covers the skin, the latter will not be cooled in stagnant air and temperature of the body will rise, thereby causing discomfort to the worker. If the atmosphere in which man works is already full with moisture, it will have no capacity to absorb moisture of perspiration. The drier the air, the more comfortable it is for the worker, as evaporation of perspiration from his body is brisk and the body temperature does not rise to the limit of discomfort. The cooling effect of the air depends not only on the temperature and humidity, but also on the velocity of air.

The sources which contribute to the moisture content of mine air are:

i. Original moisture content of air.

ii. Moisture given off from the strata in wet downcast shafts.

iii. Wet roadways, working places and drains.

iv. Perspiration of men.

v. Water vapour given off during burning of lamps.

vi. Water introduced in the mine for wet cutting, water infusion, spraying on coal dust, *etc.*

Hygrometer : An instrument to determine relative humidity of air, *i.e.,* the extent to which it is saturated with moisture, is known as hygrometer, a typical example of which is the dry and wet bulb hygrometer. Essentially, it consists of two thermometers mounted side by side on a suitable frame. One thermometer has a dry bulb and it indicates the actual temperature of the surrounding air. The other thermometer has its bulb covered with a moist cloth which dips into a small bottle filled with water. Constant evaporation of moisture takes place from the wet bulb, thereby cooling it and bringing down its temperature. When the air is relatively dry, *i.e.*, it has a low relative humidity, there is a large difference between readings of dry bulb and wet bulb, When the air is nearly saturated, the two readings have hardly any difference.

A hygrometer convenient to carry underground is Whirling Hygrometer (Fig. 2.3). Two thermometers are placed on a frame and bulb of one is convered with wet cloth. When the frame along with the thermometers is whirled at 200 r.p.m. for about a minute the readings of dry and wet bulbs enable the operator to calculate the relative humidity of air from tables. Relative humidity of mine air in the temperature range obtained in Indian mines can be roughly calculated as follows :

Deduct from 100, 7% per °C temperature difference between dry and wet bulb readings above 25°C of dry bulb reading, 8% per °C difference for dry bulb temperature from 20 to 25°C, and 9% per °C difference for dry bulb temperature of upto 20°C.

If the relative humidity of air is high and the air velocity is brisk the combined effect is drying of the coal dust and creation of dusty condition in a coal mine. It is not a desirable feature when viewed in the context of coal dust suppression measures. Moreover, high velocity of air causes discomfort to the workers. For these reasons the maximum air velocity has been laid down by D. G. M. S. by Cir. No. 42 of 1974 (See Chapter on coal dust). The velocity of air at the working face at 0.5 m to 2.0 m/s is reasonable for comfortable working conditions.

In addition to these considerations the capacity of a worker to work in mines with humid conditions depends on the extent of his acclimatisation. An Indian worker may continue to work reasonably well in an underground atmosphere where the wet bulb temperature is 30°C or nearabout.

The temperature and relative humidity of air increase in the vicinity of a place of spontaneous heating of coal and if the hygrometer readings are taken regualrily they provide an indication whether or not heating is taking place.

Kata Thermometer : To judge whether a working place is suitable for a man to work efficiently and without discomfort, it is necessary to know the temperature of air at the working place, the relative humidity and air velocity. The joint effect of all these factors can be known with the help of kata thermometer which measures the cooling power by a combination of the above mentioned three factors at the instrument temperature of 36.5°C, the normal temperature of human body. Kata consists of an alcohol thermometer with two marks graduated on the stem at 35°C and 38°C. The instrument is used as follows :

The bulb is immersed in hot water carried in a thermos flask until the alcohol rises above the upper graduation. The bulb is wiped dry and the time required for the alcohol column to drop from the upper mark to the lower mark is noted. The procedure is repeated with a wet muslin cloth wrapped on the bulb.

An instrument factor is marked on every instrument. The kata factor of an instrument is the number of millicalories of heat which it loses per sq. cm. of surface area of the bulb on cooling from 38° to 35°C. It is nearly 480.

$$\text{Cooling power} = \frac{\text{Kata factor}}{\text{Time in sec. for alcohol to fall from upper mark to lower mark}}$$

The cooling power calculated is called dry if no wet cloth is used on the bulb, and wet if wet cloth is used. The cooling power is given in millicalories per. sq. cm. per second, and it is the relative value that is important. Minimum limits of the Kata thermometer cooling power for comfortable working are as follows :

	Dry Kata	Wet Kata
For sedentary workers	6	18
For light manual workers	8	25
For hard manual wokers	10	30

If the dry time is used, the result is the cooling power by conduction and convection. Cooling power by conduction, convection and radiation is given by using the wet bulb time.

A *temperature and humidity monitor* marketed by Uptron Ltd. is as follows :

The Sieger model TH1 monitor is an intrinsically safe, low cost device for continuously measuring the temperature and humidity in underground mines.

The instrument consists of three units, a control module, a remotely sited transducer complete with sensors, and a connecting cable, which can be up to 400 metres long. A number of alternative power inputs can be used, including 12V DC, 18V DC, or 15V AC. Intrinsically safe supplies, all of which will also charge the inbuilt nickel cadmium battery. This battery can be used as an automatic stand by supply if the input power is interrupted and also allows the instrument to be operated for many hours in a new development or areas where there is no power available. When used with an intrinsically safe power input a Special Junction box is used which prevents incorrect connection of the various leads.

The transducer unit contains separate temperature and humidity sensor together with voltage regulation and signal conditioning circuits, power for which is obtained from the control module. This is contained in a metal case and in addition to a long life, back up battery supply, also contains the main amplifier and display driver circuits associated with the LCD indicators on the instrument front panel. These give continuously readings of both temperature (°C) and humidity (in percentage RH), in addition to warning of low battery or fault conditions.

Air velocity meters : The meters used to determine the velocity of air in a mine are the commonly used instruments like anemometer, velometer and pilot tube. They have been described elsewhere in this book.

Measurement of air pressure :

The pressure of atmospheric air is measured by Fortin's barometer or aneroid barometer. A barometer records absolute pressure in mm of mercury column. The value of atmospheric pressure varies from place to place and depends further on climatic conditions. It is maximum at the sea level and gradually decreases with altitude above sea level. The standard atmospheric pressure, also called the mean or normal atmospheric pressure, is defined as that pressure which supports a column of mercury 760mm high at sea level when the temperature of mercury is 0°C. This is stated as absolute pressure of 760mm of mercury. In the metric units the value of standard atmospheric pressure is very nearly 1 kgf/cm². The internatinally accepted value of standard atmospheric pressure in SI units is 101,325 N/m², i.e. approximately 100,000 N/m². The unit bar (b) used by the meteorologists to express atmospheric pressure is equivalent to 10^5 N/m², and therefore very nearly equal to the atmospheric pressure. A 760 mm column of mercury is equivalent to $\frac{760 \times 13.6}{1000}$ metres of water column *i.e.* 10.34 m of water column (sp. gr. of mercury is 13.6). If the barometric pressure at a place is h mm of mercury, its conversion into SI units is simple.

$$\text{h mm height of Hg} = (101{,}325/760) \times \text{h N/m}^2$$
$$= 133.3 \text{ h N/m}^2$$

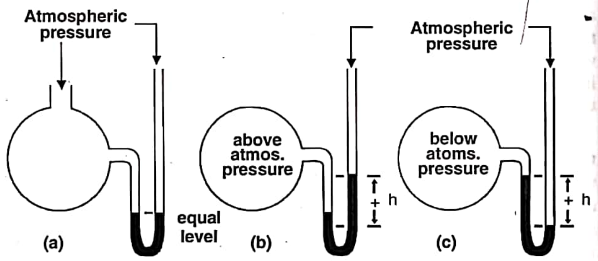


Fig. 2.4

Pressures of fluids are measured by pressure gauges of manometers. A manometer measures small pressure differences above or below the atmospheric pressure. It consists of a vertical U-tube containing a liquid and one limb of the U-tube is connected to the vessel containing the fluid under pressure while the other limb is open to the atmospheric air. The liquid in the U-tube is generally mercury but if the pressure difference is small, water or other lighter liquid is used. The difference between the levels of the liquid in the two vertical limbs of the U-tube is a measure of the pressure of the fluid above the atmospheric pressure (Fig. 2.4, b) or below the atmospheric pressure (Fig. 2.4 c). In Fig. 2.4 if the prevailing atmospheric pressure is P mm of mercury and if the liquid used in the manometer is mercury, the absolute pressure of fluid in the cylinder at b is P + h mm of mercury and at c, it is P – h mm of mercury. The amount by which a pressure is below the prevailing atmospheric pressure is referred to as vacuum and it is quoted in mm of mercury or water column. Fig. 2.4 c, represents a vacuum of h mm of mercury or water column. Fig. 2.4 c, represents a vacuum of h mm of mercury. Most pressure measuring devices indicate pressures above or below atmospheric pressure. Such devices indicate zero when exposed to the atmosphere; consequently pressures obtained from such instruments are reported as gauge pressures.

Absolute pressure = prevailing atmospheric pressure ± gauge pressure.

The minus sign is to be used when the pressure gauge records vacuum. Such pressure gauge recording vacuum is called a vacuum gauge.

If a manometer employed for finding out the pressure of a fluid uses water in place of mercury and the difference of water levels in the two limbs of U-tube is h mm, then absolute pressure in mm of Hg

$$= \frac{\text{barometric height of water column in mm} \pm \text{h}}{13.6}$$

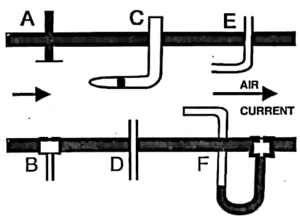
The pressure developed by a mine fan is small and it is measured by a manometer using water. If very small pressure differences are to be measured, say, upto 50 mm of water, inclined manometers are used in which the limbs of the U-tube are inclined to the horizontal.

Water Gauge :

The atmospheric pressure in an underground mine can be measured by the aneroid barometer. The difference of pressure between nearby points is however known by a water gauge which is essentially an ordinary glass tube of U-shape containing water and having one open end of U-tube connected to one point of low pressure and the other open end connected to another point having high pressure. The difference between the water levels of the two legs of the U-tube record the pressure difference between the two points. Water rises in the leg which is connected to the point of low pressure and falls in the other leg. A difference of one mm (stated as 1 mm water gauge) represents a pressure difference of 1 kgf/m². A water gauge is placed in the fan draft; one end connected to the atmosphere and the other in the fan drift very near the fan (towards the mine).

The term pressure is used in mine ventilation with the following refinements.

1. **Static pressure :** This is the pressure exerted by a moving fluid on a surface parallel with the direction of movement.
2. **Velocity pressure :** This is the pressure exerted by fluid by virtue of its motion and it varies with the square of its velocity.



A, B and C give static pressure,
D – not recommended
E – gives total pressure or fan pressure

Fig. 2.5 U-tube and various methods of measuring air pressure mercury is 13.6). If the barometric pressure at a place is h mm of mercury, its conversion into SI units is simple.

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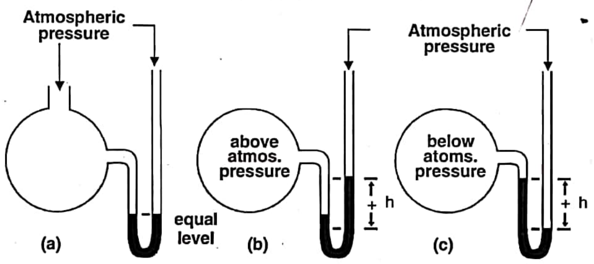

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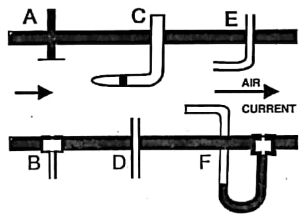


A, B and C give static pressure,
D – not recommended
E – gives total pressure or fan pressure

Fig. 2.5 U-tube and various methods of measuring air pressure

3. **Total pressure :** This is the algebric sum of (1) and (2). In the case of a fan the total pressure or fan pressure (or ventilating pressure) is the difference between the mean total pressure of the fan and the mean total pressure of the air entering the fan. For the fan located on the surface it is the difference between the atmospheric pressure and the total pressure in the fan drift. The total pressure (*i.e.* fan pressure or ventilating pressure) is measured by a facing tube *i.e.* with the open end of the tube facing upstream side of air current (*e.g.* at E in Fig. 2.5)

Measurement of Ventilating Pressure :

The static pressure ps may be measured in several ways by a U-tube gauge having one leg exposed to the atmosphere and the other in the fan-drift, as shown at A, B and C in Fig. 2.5 At A, the tube projects into the drift at right angles to the direction of air flow and is provided with a thin, sharp edged flat disc, called a d' Arcy tip. At B, a recess is cut in the side of the drift and is closed by a smooth metal plate fitted flush with the drift side and perforated with a small hole. At C is shown a hooked tube, sealed at the point and provided with four small holes without burr. This has the advantage that it furnishes the maximum reading when correctly aligned in the air current. All three methods give reasonably accurate results. On the other hand, a plain ended tube, like D, invariably gives incorrect results (too low a reading with a compressing fan and too high with an exhausting fan)

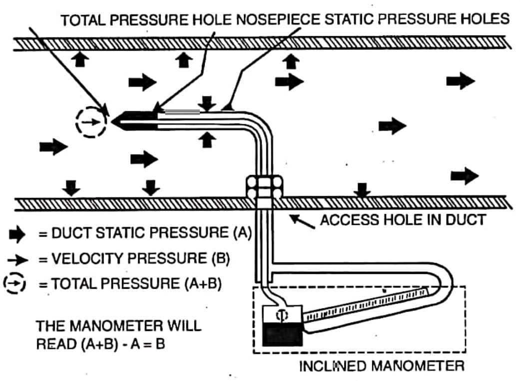


Fig. 2.5A Illustrating static, velocity and total pressure and use of an inclined manometer

The total pressure P (fan pressure) is measured correctly by a facing tube, as shown at E, i.e. with the open end of the tube facing to windward. It is this pressure that must always be used when calculating the 'H. P. in the air' or when determining the efficiency of a fan. It is sometimes, however, convenient and more accurate (owing to the turbulent and gusty nature of the airflow in a drift or airway) to measure the average static pressure and make allowance for the velocity pressure by calculation from the observed mean air velocity.

The water gauge produced by fans in our mines is small, varying between 25 mm and 150 mm, though it is higher for the deep mines. Very small water gauge as in the case of underground auxiliary fans or booster fans is measured by U-tubes with inclined limbs and necessary corrections applied to convert the observed inclined water gauge reading into vertical water gauge reading.

For the measurement of water gauge smaller than 50 mm an inclined water gauge is generally used (Fig. 2.6). The reading is converted into equivalent vertical w.g. as follows :

Vertical w.g. $= L \times \sin\alpha$, where α is the angle of inclination with the horizontal and L the inclined reading of manometer. The manometer can be made more sensitive by using a lighter liquid instead of water e.g. alcohol. The vertical w.g. is then

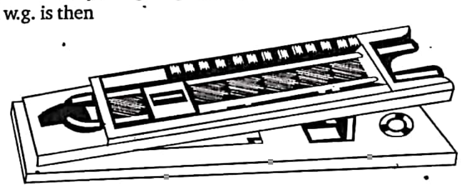


Fig. 2.6 Inclined manometer

vertical w.g. $= L \times \sin\alpha \times w$

where w is the sp. gr. of the liquid

The inclination of the water gauge is conveniently adjusted to 5°44' as the sine of that angle is 0.1.

Inclined water gauges can be read upto 0.2 mm and in some cases even upto 0.02 mm depending upon scale graduation.

The Pitot Tube :

A pitot tube is a device which can be used to measure the static pressure, the velocity pressure or the total pressure. The principle involved is shown at F in Fig. 2.5 which shows one leg utilised to obtain the static pressure and the other to obtain the total pressure, the actual reading recording the velocity pressure which is equal to the difference between the limbs of the U-tube. If the velocity pressure of air at any point has to be measured the two limbs shown at F should be near each other. In a pitot tube, as shown in Fig. 2.7 the inner tube is connected via nipple N_1 to one leg of the w.g. and is subjected to total pressure. The outer (static) tube is sealed at the end (as indicated by the shaded portion) but is pierced as indicated with seven holes. It is connected via nipple N_2 to the other leg of the w.g. and the instrument is placed in the airway so as to face the air current. The relation between the observed w.g. and the velocity of the air is given by the following equation.

$$V = 4.43\sqrt{\frac{w.g.}{w}} \quad \text{or, at standard air density}$$

$$V = 4.015\sqrt{w.g.}$$

$$= 4\sqrt{w.g.} \quad \text{(Approximately)}$$

where, V = Velocity of air in m/sec

w.g. = Pressure in mm of w.g. or in kgf/m^2

w = Density of air in kgf/m^3

A pitot tube should be used in conjunction with a sensitive inclined manometer.

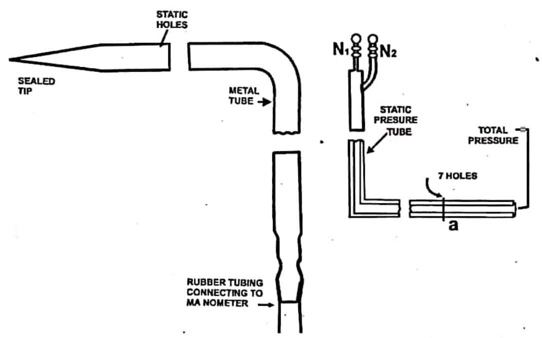


Fig. 2.7 Pitot tube

Airflow meter of Nanda Manufacturing Co. comprises:
(a) Portable inclined manometer. (b) Pitot static tube.

Portable Inclined Manometer :

Pitot readings are very small for the common range of air velocities. Ordinary U-tube manometers are not useful for measurement of gas velocities below 12 m/s. For this use inclined tube manometer is suggested. These manometers are well type with, single limb indication. NMC inclined manometer can be swivelled in 4 positions - 1 in 20, 1 in 10, 1 in 5 and vertical. Maximum limit of pressure measurement is 250 mm Wg. Flexible PVC connecting tubing in contrasting colours, each of 9 m length, is provided. A bottle of manometer fluid with the density labelled on it and a funnel for topping up the reservoir fluid is included among the scope of supply. A table showing the conversion of wg. into Pascals is supplied.

Levelling is acurately and conventinelty done by means of screw threaded feet in conjunction with spirit levels.

Pressure range : 0-125/250/500/2500 Pascals (N/m^2)
0-12.5/25/50/250 mm wg.

Velocity range : 0-14.2, 0-28 m/sec.

Gas Detectors :

These have been described in the earlier chapter on mine gases.

PRODUCTION OF VENTILATION :

Air flows from a region of high pressure to a region of low pressure. The difference of pressure may be caused by

1. Purely natural means. It is then called natural ventilation, or
2. By a fan; it is then called mechanical ventilation.

The ventilation in a mine using a fan is a combination of natural and mechanical means.

Natural Ventilation : 4unit

How is natural ventilation produced? Consider that a level roadway joins two shafts equal in dia. and of equal depth from a level ground surface. If the air in both the shafts is at the same temperature and pressure, there are two columns of air equal in weight which balance each other and there will be no flow of air from one shaft to the other.

If, however, density of air in one shaft is more, the column of air in that shaft will have a higher weight and the difference in pressure at the bottoms of the shafts will cause flow of air from high pressure shaft to low pressure shaft. The difference in density between of the two shafts may be produced by any of the following factors :

i. Presence of firedamp, or steam purposely introduced in one of the shafts, which renders the air lighter.

ii. Presence of fire in one of the shafts. The fire heats up the air which has then less density.

iii. Passage of cold water down one of the shafts; the cold air is more dense.

iv. Movement of case in the shaft.

v. Unequal depth of shafts.

The last factor is important and may be explained by reference to Fig. 2.8. Consider two shafts BC and FD at different surface levels and joined at their bottoms by a level roadway DC. If we consider a horizontal imaginary plane AF passing through the point F and the point A, an extension of the shaft BC, the air at point A and F has same temperature and pressure as that of the atmosphere. It is, therefore, only the air columns below the point A and F which should create a difference of pressure if natural ventilating pressure is to cause a flow of air through the shafts and through their underground connecting roadways. At C the air pressure is due to the air column AB above the pit top and the column BC; at D the air pressure is due to the air column FD.

The temperature of the rocks below ground is not the same as the temperature at the surface. The day-to-day variation and the seasonal variation in atmospheric temperature affects only the rocks near the surface but whatever be the surface temperature, at 15m below ground the temperature of rocks is constant. It is 27°C in Jharia field. Beyond that depth of 15m the temperature of the rocks increases with depth at a definite rate. This rate of increase is known as Geothermic gradient and varies in different areas. In Jharia field this rate increase is 1°C for every 38 m depth below the constant temperature line. In Kolar Gold field it is 1°C for 68 m; and it is 1°C for every 140 m in Gold Mines of Rand in South Africa. In Mosabani copper mines the geothermic gradient is 1°C for 52 m but indications are that it is getting steeper as the temperature recorded at 30th Level was 47.8°C against 46.2°C expected as per the rate stated above.

In Fig. 2.8 the constant temperature line is shown at 15m below the surface and it is always parallel to the surface contour. Below that line the air in the shafts gets heated mainly due to conduction of heat from the strata. Let us consider that atmospheric temperature in Jharia Field on a summer day is 40°C and on a winter day 20°C. The average temperature of column AC is that due to AB, and BC.

In winter column AB is colder than column FE; BC is also colder than ED which is at a greater depth from the surface. Thus AC is colder than FD and weight of air in AC is more than that of air in FD. Therefore, air at a point C is at a higher pressure than at D and current of air flows from C to D so that BC acts as a downcast shaft and FD as upcast shaft.

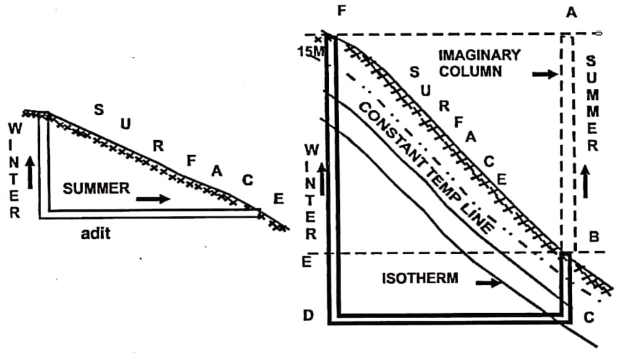


Fig. 2.8 Natural Ventilation

In summer the average temperature of air in AC is high compared to that of FD. Temperature of AB is atmospheric (40°C) and higher than that of FE. In this particular example BC, being a shallow shaft, is a small length compared to AB and though BC is colder than ED, the mean temperature of AC is higher than the mean temperature of FD. The air in AC is therefore less dense than in FD. Hence, air flows from D to C. There is thus a reversal of air currents in winter and in summer. In all cases, the entrance at the lower surface level is the downcast in winter.

The natural ventilation therefore causes the mine air to travel in opposite directions in summer and in winter. In deep mines, however, the air will travel in the same direction throughout the year provided the constant mean temperature of the air in the upcast shaft is higher than in the downcast shaft.

If two shafts have the same depth and are at the same surface level, and if a current of air once begins to flow for any reason whatever, the cool air enters one shaft and warmer air goes up by the other, and this results in keeping one shaft cooler and the other warmer. This causes flow of air by natural ventilation as long as difference in temperature continues and even when the surface fan is stopped. There are various ways in which the air may be induced to flow, e.g. by the action of wind at one shaft mouth, and other causes mentioned earlier.

Natural ventilation in a shallow mine assists the flow of air caused by the fan for part of the year and opposes the pressure difference produced by the fan in the remaining part of the year. In those mines where there is much difference between day temperature and night temperature, natural ventilation flow during night may be opposed to that during day. When the temperature difference of the two air columns is maximum the N.V.P. is maximum and it ceases altogether when there is no temperature difference.

The natural ventilation pressure produced in the above example may be measured by a water gauge placed at the bottom of the two air columns AC and FD. The water gauge should be placed on a partition erected across the connecting road CD; one leg of the water gauge should be exposed to the air column AC and the other to the air column FD.

It should be noted that in all questions concerning N.V.P. (1) The two air columns should be considered between two imaginary horizontal planes, one at the shaft collar of the higher level shaft and the other at the lower point of the two shafts. When one of the outlets is not a shaft but an inclined drift or an adit, the same principle applies. (2) Total N.V.P. can be calculated considering the two air columns stationary. To cause flow of air through the shafts, some of the water gauge of the N. V. P. is expended in overcoming resistance of the shafts and the connecting roadways. If air flows through the shafts and underground roadways, a water gauge connected at shaft bottom between UC and DC shaft across a partition door will record the total N.V.P. minus the pressure spent on causing air flow through the shafts and it is the balance N.V.P. recorded on the water gauge which will be available for circulating air to the inbye workings. In contrast to this when a mine is ventilated by a surface fan, the maximum water gauge developed by it and available for ventilation of the mine including the shafts is that which is recorded on a U-tube placed at the fan drift (one leg exposed to the atmosphere and the other exposed to the air in the fan drift).

Motive Column : In the case of N. V. P. it is the excess weight of air in D. C. air column which gives rise to the N. V. P. The height of this excess weight of air column of DC shaft, 1 m² in cross section which gives rise to N. V. P. is called motive column. In other words, the motive column, when referring to N. V. P., is the preponderant or unbalanced part of the whole D. C. column, 1 m³ in cross section, i.e. that part of the D.C. column which is not balanced by the UC air column. The following example will make it clear.

Example : Mean air temp. in a D.C. shaft 400 m deep is 28°C and in the U.C. shaft is 38°C. Calculate (i) the motive column, and (ii) the N.V.P. assuming average barometric pressure in D.C., shaft to be 750 mm of Hg.

Answer : The height of the motive column is given by the formula,

$$h = \frac{Tu - Td}{273 + Tu} \times D$$

where h = height of motive column (m)

Tu = average temp. in upcast shaft (°C)

Td = average temp. in down cast shaft (°C)

D = depth of column between top of the higher level shaft and bottom of the deeper shaft (in m)

N. V. P. = Motive column × Density of air in D.C. shaft

$$\text{Motive column, } h = \frac{Tu - Td}{273 + Tu} \times D$$

$$= \frac{38 - 28}{273 + 38} \times 400 = \frac{10 \times 400}{311}$$

$$= 12.8 \text{ m}$$

$$\text{Density of air in the D.C. shaft} = \frac{0.4645\ B}{273 + Td}$$

Where B is the barometric pressure in mm of Hg.

$$\text{Density, } w = \frac{0.4645\ B}{273 + Td}$$

$$= \frac{0.4645 \times 750}{273 + 28}$$

N. V. P. = motive column × density of air in D. C. shaft

$= 12.8 \times 1.157 = 14.81$ kgf/m² = 148.1 Pa

Density of air (mass per unit volume) varies with its temperature, pressure and moisture content. The formula for density of moist air in SI units is as follows :

$$\text{Density } w = \left(\frac{B - 0.378\,e}{287.1\,T}\right)10^3 \text{ Kg per m}^3$$

where B = barometric pressure in kPa

T = temperature of air in K

e = water vapour pressure in air in kPa

The various causes which tend to heat the air as it travels through a mine are :

1. Conduction of heat from the strata.
2. Compression of the air due to depth in the D.C. shaft and dip working.
3. Burning of lamps.
4. Oxidation of carbonatceous material.
5. Heat given out by men.
6. Heat given out by machinery and subsiding strata.

Of these, the first three are the most important, the others being of relatively minor consequences except in confined places.

The various causes which tend to cool the air are :

1. Evaporation of moisture from wet shafts and roadways, or from the coal itself.
2. Expansion of the air as it rises up the U.C. shaft, or up the rising roadways, to a higher level.
3. Local cooling effect due to expansion of compressed air at the exhaust of compressed air motors.

FAN VENTILATION :

Artificial ventilation is produced in a mine by fans. A fan may force the air in a mine (forcing fan) or it may suck up the air from it (exhaust fan). The main fan which deals with all the air in a mine has to be situated on the surface under the Mining Regulations. In that case, it is not easily affected by an explosion underground and is easily accessible for supervision and mainteance. Every fan installation on the surface should be capable of reversing the ventilation and if it is electrically driven, there should be two independent electric circuits for power supply to it. In very gassy coal mines a standby ventilator is required by DGMS cir. 42 of 1972.

The fans used are of two types :

1. The centrifugal fan. 2. The air screw or axial flow fan.

Both these types can be arranted to act (a) as forcing fan or (b) as an exhaust or suction fan. The exhaust fan, as it sucks the air is situated at the mouth of the upcast shaft on the surface and the forcing fan at the mouth of the downcast on the surface.

Centrifugal fan : A centrifugal fan essentially consists of a wheel carrying blades or vanes at the periphery. Consider a particle of air at a in Fig. 2.9. When the blade moves with the rotation of the fan, it tends to drive the particle in a circular path towards b. The particle, however, has intertia and it therefore tries to move in a straight line in the direction ac. The effect of these two motions is that it follows an intermediate path and is finally thrown clear beyond the tip of the blade in a direction approximately tangential to the circumference as shown by the arrow de. As the blades act simultaneously on all the air in contact with them and are driving the air beyond the periphery during rotation, there is a suction effect at the centre of the fan wheel where air enters to take place of the air driven out at the periphery. The speed of the fan varies from 100 to 300 r.p.m. In actual practice the blades of the fan are not exactly radial, but either curved backward or curved forward, though the backward curved blade types are common.

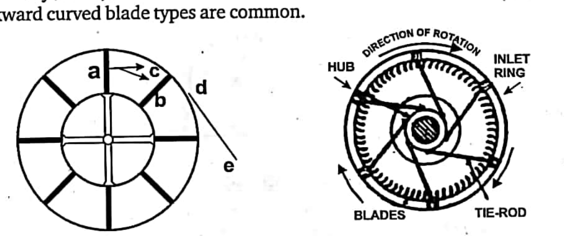


Fig. 2.9 (a) Principle of working of centrifugal (b) Right wheel of a sirocco fan

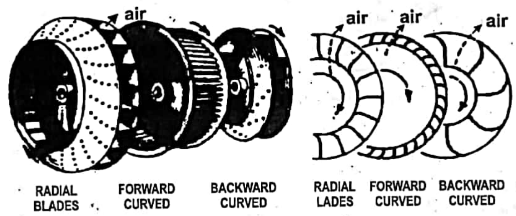


Fig. 2.10 Centrifugal fan with radial forward curved and backward curved blades.

Sirocco fan : This is the common type of centrifugal fan used in many of our mines. The fan wheel is cylindrical having 64 blades in single inlet and 128 blades in a double inlet fan. The blades are nearly 160 mm deep and are cup shaped (fig. 2.10) Fig. 2.11 shows the installation of a Sirocco fan. The purpose of the spiral casing is to enclose the fan wheel and prevent re-entry of the discharged air. The cross sectional area of the spiral casing gradually increases to accommodate the area of the spiral casing gradually increases to accommodate the progressive increase in the quantity of air discharged by the fan wheel. The evasee which is a passage of increasing cross-section for the air discharged by the fan, is provided to reduce the final velocity of discharge to a minimum. This helps smooth flow of air and some of the velocity energy in it is transformed into pressure energy. The water gauge which the fan motor has to develope is, therefore, reduced to that extent and the expenditure of power is accordingly saved. A well designed evasee thus increases the efficiency of the fan and reduces cost of power for a given quantity of air.

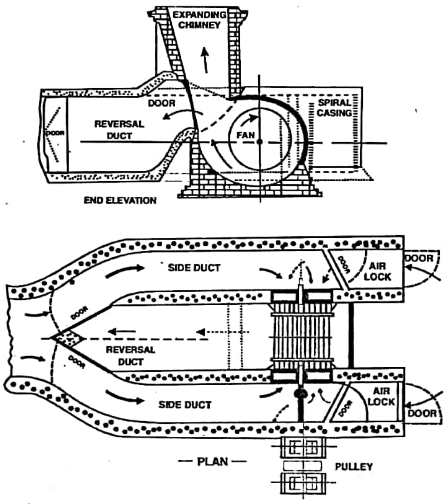


Fig. 2.11 Instillation of a centrifugal fan, (Sircocco fan) double inlet. Firm arrows indicate air travel; the position of doors at the fan drift and the air travel after reversal are indicated by broken lines.

Air Screw of Axial Flow Fans :

An air screw fan or an axial flow fan pushes the air forward in the direction parallel to the axis i.e. axially, without changing the direction of air current unlike in the centrigual fan. Consider a long bolt fitted in a nut at half its length. If the nut is confined to one place and made to rotate, the bolt travels along its axis and when the nut is rotated in reverse direction, the travel of bolt is in opposite direction. In the case of an air screw fan its radial blades can be compared with the nut which has to rotate in one place are the air column on both the sides of the plane of the blades can be compared to a long bolt. As the shaft and the blades mounted thereon rotate, the air, because of the shape of the blades, is pushed axially, *i.e.*, in line parallel to the fan shaft so that on one side of the blades the air is under pressure and on the other side, under suction. The above principle of air screw fan has been further refined in what is known as aerodynamic principle adopted in the manufacture of propeller blades of aeroplanes and the present day axial flow fans are based on aerodynamic principle. The blades are set at an angle to the plane of rotation. The w.g. developed and quantity circulated depend on the speed of the blade tips and as compared to the centrifugal fan a much higher peripheral speed is needed for creating the same water gauge. The higher the peripheral speed of the blades, more is the noise and for fans near residential areas the permitted limit of the speed of blade tips is 4200 m/min. At this speed the pressure that may be generated is nearly 12.5 cm of water gauge.

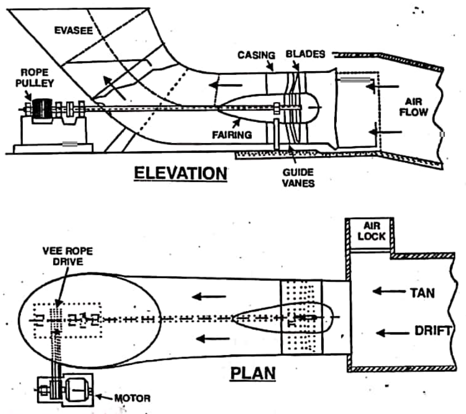


Fig. 2.12 Installation of air screw fan.

An axial flow fan consists of a rotor carrying a number of blades, somewhat similar to those of an aeroplane propeller and revolving within a cylindrical casing. The air which leaves the fan blades has a rotary movement and to counteract it, the fan is fitted with fixed guide blades at the outlet side so that the motion is partly restored to axial direction. Direction of travel of air current to the mine can be changed simply by changing the direction of motor rotation but the quantity of air reduces to nearly 40% on reversal.

The blades of the rotor of an air screw fan, as already stated, are set at an angle to the plane of rotation. This angle or pitch can be varied by 10° to 15° in either direction from the normal setting and the hub on the shaft is marked suitably to indicate the pitch. By varying the pitch of the blades, the quantity of air circulated can be varied but pressure generated remains nearly the same.

Each set of rotor blades along with its stationary guide vanes constitutes one stage. The water gauge developed can be increased by fixing two or more stages on the fan shaft if the fan is designed for it. The action may be compared with the head available in a pump by using a number of stages in it. By increasing the number of stages of axial flow fans, the same water gauge can be produced with less fanspeed. The w.g. developed per stage rarely exceeds 130 mm.

Fan Drives : An electric motor is practically the standard driving arrangement for fans which have to run continuously. The motor is usually of constant speed with V-belt drive arrangement and the speed of the fan is varied by changing the gearing ratio of the driving and driven pulleys. With AC electric supply synchronous induction motor is preferred for large fans as it has an advantage of improving the power factor. The main mine fan has to run for all 365 days of the year, 24 hours a day. It is estimated that in a coal mine nearly 5 te of air has to be sent down a mine for every te of coal raised. The rigorous duty a fan has to perform can be judged from this and the need for a driving arrangement and transmission of high efficiency can be well appreciated.

Air Velocity : Low velocities in coal mines are dangerous from the point of view of methane layering and can cause uncomfortable environment conditions of working. High velocities are undesirable as they raise dust and give a feeling of "cold" to the workers. Air velocity at the working face should be between 0.5 m and 2.0 m/s. Carbide lamps can withstand an air velocity of 5 m/s but for safe working the velocity should not exceed 2.5 m/s at places where carbide lamps have to be used. Modern flame safety lamps can withstand an air velocity of 15 m/s. It is a good practice not to exceed an air velocity of 1.7 m/s at the face; 3 m/s in conveyor roads, loading points and transfer points; 5 m/s in main haulage roads; 7.5 m/s in smooth lined airways not

used for haulage purposes and shafts with rope guides; and 12.5 in shafts with rigid guides as well as fan drifts. High air velocities in shafts with flexible guides may cause too much swinging of cages. Air velocities in diesel locomotive sheds and battery charging stations should not be less than 0.75 m/s.

Control of quantity of air delivered :

The air circulated by a fan can be varied by the following methods:

1. By removing some of the blades (for less quantity).
2. By changing the pitch of the blades on the hub of the rotor shaft in the case of an air screw fan.
3. By installing two or more stages on the shaft if the installation permits of it. Such arrangement is possible only with an axial flow fan. This increases the water gauge and therefore for the same mine, circulates more air.
4. By changing the fan speed. This is applicable for both centrifugal as well as axial flow fans.
5. By placing air regulating dampers in the fan drift to reduce the quantity immediately in case of an emergency.
6. By a controlled leakage via surface airlock. This is possible by partly opening the surface airlock doors.
7. By changing the mine resistance by installing air regulators in some ventilation districts or by withdrawing them, or by arranging ventilation districts/splits in parallel (to reduce the resistance).
8. By replacing the fan.

Air screw Vs. Centrifugal fan :

1. An air screw fan requires less space in installation compared to a centrifugal fan.
2. Reversal of ventilation can be easily achieved with air screw fans, merely by reversing the direction of rotation. The water gauge developed and the quantity circulated is, however, much less, 40 to 50% of the original figure on reversal.
3. Air screw fans as well as centrifugal fans have an overall efficiency varying from 70 to 85% but the manometric efficiency of axial flow fans is low, only 20 to 30%.
4. Quantity of air can be varied within wide limits in the case of an air screw fan by varying the pitch of the blades and they run efficiently over a wide range of mine resistance e.g. in the case of a developing mine. A centrifugal fan with backward curved blades also maintains a high efficiency over a wide range of mine resistance.

5. The air screw fan is noisy at high speeds of the blades.
6. An air screw fan is more suited for low water gauge and high volume but a centrifugal fan, more suited for high pressure and less volume. An air screw fan, however, can be conveniently used for developing high water gauge by installation of stages.
7. An air screw fan is convenient as an underground booster fan due to ease of installation and the requirement of space.

Forcing vs. Exhuast Fans :

1. A forcing fan requires an airlock at the D.C. shafts but an exhaust fan has to be provided with an airlock at the U. C. shaft.
2. The pressure of forcing fan has the effect of keeping the gases of the goaf and old workings confined to their places. If the fan stops or breaks down the air pressure in the mine falls to normal atmospheric pressure and the gases in the goaf and old workings expand and overflow into the mine workings.
3. A forcing fan handles clean atmospheric air and has a longer life compared to the exhuast fan.
4. In the case of skip winding there is formation of more dust during the loading of the skip and it is a general practice to install the skips in the upcast shafts. If the mine has exhaust ventilation the upcast shaft is to be provided with airlocks and this is inconvenient due to the need to keep the doors of the airlock closed. Under such circumstances, a forcing fan at the DC shaft is advantagesous unless other considerations require installation of an exhaust fan.

Laws of mine air friction :

When air passes in a mine it has to overcome the friction of the walls of the shafts/inclines and also of the floor, roof and sides of the roadways, the props, bars and other objects in its passage. The laws which govern the passage of air in a mine are as follows :

$$P = \frac{KSV^2}{A} \quad \text{or} \quad P = \frac{KSQ^2}{A^3}$$

where P = pressure absorbed in kgf/m^2 or mm w.g.
K = coefficient of friction, in kilomurg
S = rubbing surface in m^2 (i.e. perimeter × length)
V = velocity of air in m/sec.
A = area of cross-section of roadway in m^2
Q = quantity of air flowing in m^3/sec.

Coefficient of friction : In the above formula $P = \frac{KSV^2}{A}$

if S = 1, V = 1, and A = 1, then K = P

The coefficient of friction, **Kilomurg**, written as $k\mu$ is the resistance of friction of an airway that absorbs a pressure of 1 kgf/m^2 (i.e. one mm watergauge) when 1 m^3 of air flows through it per sec. (air density taken as 1.2 kg/m^3). The coefficient of friction in Germany is Weissbach (w) and has the same value as kilomurg. The kilomurg is often too large unit for measuring mine resistance; the resistance of a roadway 50 m long, 6 m^2 cross-section being equal to 0.0025 kilomurg. Therefore, along with the basic unit of resistance, kilomurg, another unit of friction is used and it is called the murg (μ).

1 kilomurg ($k\mu$) = 1000 murg (μ)

The resistnace of the roadway mentioned above then becomes 2.5 murgs.

The unit of resistance in S. I. units is $Ns^2 m^{-8}$. Unlike names of other units like Weissbach and Murg, the unit in S. I. has no name and it may be called "resistance unit".

1 Weissbach = 9.81 $Ns^2 m^{-8}$

1 Murg = 0.00981 $Ns^2 m^{-8}$

The coefficient of friction in S. I. units is $Ns^2 m^{-4}$. Its value (average) is as follows in units of $Ns^2 m^{-4}$ or kgm^{-3}.

Smooth lined roadway (brick or concrete lined)	0.00216
Unlined roadway in sedimentary rock or coal	0.01118
Shaft, bricklined	0.00618
Staple pit without bunton	0.02256

It is necessary to emphasize that the value which indicates the difficulty or ease with which a roadway can be ventilated is not the coefficient of friction but the friction or resistance R. Roadways can have the same coefficient of friction but if their cross sections are differnet, the larger roadway will be easier to ventilate than the narrower one. The resistance of a roadway is high at the bends and at places having sudden change in cross section.

The relation between pressure, resistance and quantity is expressed by the formula –

$$P = RQ^2$$

where P = pressure in kgf/m² or waterguage in mm

Q = quantity of air flowing in m³/s

and R = resistance of the airway in kilomurg.

Example : What is the watergauge required to pass 1500 m³ of air/min. through an airway of 2.5 m × 3 m cross section and 2000 m long.

Answer : Let coefficient of friction for the airway be assumed to be equal to 0.001 kilomurg.

then $A = 7.5\ m^2$

$S = 11 \times 2000 = 22000\ m^2$

$Q = 1500/60 = 25\ m^3/sec$

$$P = \frac{0.001 \times 22000 \times 25^2}{(7.5)^3}$$

= 32.6 mm water gauge

= 326 Pascals approx.

Resistance of roadways in series and a parallel

Series roadways : $R = R_1 + R_2 + R_3 ..... + R_n$

Parallel roadways : $\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + ... + \frac{1}{\sqrt{R_n}}$

where R is the equivalent resistance of the system covering all the roadways and R_1, R_2, R_3 are the resistance of individual roadways of that system.

A comparison of these formulae with formula on electrical resistances may be of interest.

For resistance in series, $R = R_1 + R_2 + R_3 + ... + R_n$

For resistance in parallel, $\frac{1}{R} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3} .... + \frac{1}{R_n}$

Law relating to quntity of air, water gauge and fan speed :

Let Q = Quantity of air in m³/sec.

N = r.p.m. of fan

V = Peripheral speed of blade tips in m/sec.

Law No. (1) – Q varies directly as fan speed, i.e. $Q \alpha N$ or V

Law No. (2) – Water gague developed varies directly as square of the fan speed or of the quantity i.e. Water gauge αN^2 or Q^2 or V^2.

Law No. (3) – Horsepower required to drive the fan varies as the cube of the fan speed or of the quantity i.e. H. P. αN^3 or V^3 or Q^3.

Example : In the above example if the quantity of air has to be increased from 1,500 m³/min to 3,000 m³/min, what will be the effect on H. P. of the fan ?

Answer : Suppose H. P. required for 1500 m³/min is H_1, and H.P. required for 3000 m³/min is H_2, then

$$\frac{H_2}{H_1} = \frac{(3000)^3}{(1500)^3} = 2^3 = 8 ; \text{ or } H_2 = 8 \text{ times } H_1$$

It will be obvious that increasing the speed of the fan (which in turn increases the H.P. of the motor) increases the H. P. to 8 times and is a very wasteful process. In many cases it is not possible to increase the speed of the motor as fans are driven by constant speed motors (mostly synchronous A. C. motors) and the speed of the fan can be increased only by other means like changing the ratio of driving and driven pulleys. Other methods of increasing the quantity of air lie in reducing the friction of time roadways by enlarging their size, arranging underground airways or splits in parallel, etc.

Definitions of terms used in connection with fan water gauge :

(A) The theoretical depression of a fan is the pressure difference that would be produced by a mechanically perfect fan running under ideal conditions and connected to an evasee chimney of infinite height. Its value is a calculated one and depends on the speed of the blade tips and the shape of the fan blades and it varies as the square of the speed.

The theoretical depression of a centrifugal fan is given by the formula

$$H = \frac{U (U \pm V_1 \cot \theta)}{g} \quad \text{(see Fig. 2.13)}$$

where H = Theoretical depression in m/sec of air column

U = Peripheral speed of the blade tips in m/sec.

V_1 = Radial velocity of air leaving the fan in m/sec; (also called velocity of flow)

θ = External angle between the fan blade and tangent at the periphery.

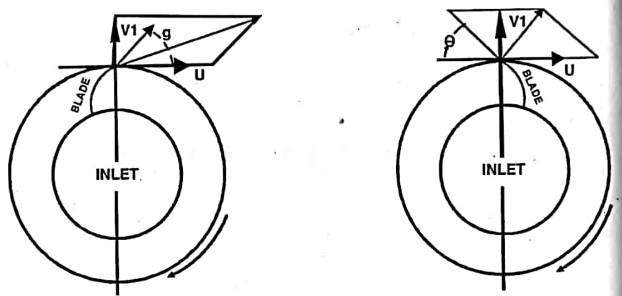


Fig. 2.13

The + sign is applicable for forward bladed fan and the – sign for backward bladed fan. For a radial bladed fan, the theoretical depression will be

$$H = \frac{U^2}{g} \quad \text{(True for air screw fan also)}$$

Example 1 : A mine is ventilated by a fan producing 6000 m³ per min at 75 mm w.g. The fan runs at 300 r.p.m. and absorbs 160 B. H. P. To increase the volume of air flowing through the mine, the fan is speeded up to 400 r. p. m. Calculate - (i) Volume of air at new fan speed, (ii) New w. g., B. H. P., and efficiency of the fan.

Answer : From fan laws, $Q \propto N$,

$$\therefore \text{ New volume} = 6000 \times \frac{400}{300} = 8000 \text{ m}^3/\text{min}$$

Again w. g. $\propto N^2$

$$\therefore \text{ New w. g.} = 75 \times \left(\frac{400}{300}\right)^2 = 133.3 \text{ mm.}$$

Also B. H. P $\propto N^3$

$$\therefore \text{ New B. H. P.} = 160 \times \left(\frac{400}{300}\right)^2 = 379$$

$$\text{Air h. P.} = \frac{PQ}{75} = \frac{133.3 \times (8000/60)}{75} = 237$$

$$\therefore \text{ Efficiency of the fan} = \frac{237}{379} \times 100 = 62.5\%$$

Example 2 : Calculate the w.g. developed and the quantity delivered by a backward bladed centrifugal fan having the following specifications -

Fan dia. 3.6 m,	R. P. M. 300
width at periphery 1.5 m,	Blade angle 40°
velocity of flow 4.5 m/sec,	Air density 1.2 kg/m³

Answer : Blade tip speed $= \frac{\pi DN}{60} = \frac{\pi \times 3.6 \times 300}{60} = 56.57$ m/sec

$$\text{Theoretical depression } H = \frac{U(U - V\cot\theta)}{g}$$

$$= \frac{56.57\,(56.57 - 4.5\cot 40°)}{9.8}$$

= 295.5 m of air column

= 295.5 × 1.2 kgf/m²

= 354.6 kgf/m² or mm w.g. = 3546 Pa

Peripheral area of fan outlet

= circumference × width

= $\pi DW = \pi \times 3.6 \times 1.5 = 16.97$ m²

∴ Quantity = area at periphery × velocity of flow

= 16.97 × 4.5 = 76.37 m³/sec

= 4582 m³/min

Example 3 : Calcualte the w.g. produced by a 3m dia. fan running at 250 r.p.m. and delivering 6000 m^3/min of air, if the blades are :

(a) radial, (b) bent backward at 35°, (c) bent forward at 35°,

Assume velocity of flow = 3 m/sec.

air density = 1.2 kg/m^3

Answer : Blade tip speed $\frac{\pi DN}{60} = \frac{\pi \times 3 \times 250}{60} = 39.3$ m/sec.

(a) For radial bladed fan,

theoretical depression, $H = \frac{U^2}{g} = \frac{(39.3)^2}{9.8}$

= 157.6 m of air column

= 157.6 × 1.2 kgf/m^2

= 189.1 kgf/m^2 or mm w. g. = 189.1 P

b) For backward bladed fan,

theoretical depression, $H = \frac{U(U - V \cot \theta)}{g}$

$= \frac{39.3\,(39.3 - 3 \cot 35°)}{9.8}$

= 140.35 m of air column

= 140.35 × 1.2 kgf/m^2

= 168.4 kgf/m^2 or mm w. g.

= 1684 Pa

c) For forward bladed fan,

theoretical depression, $H = \frac{U(U + V \cot \theta)}{g}$

$= \frac{39.3\,(39.3 + 3 \cot 35°)}{9.8}$

= 174.47 m of air column

= 174.47 × 1.2 kgf/m^2

= 209.4 kgf/m^2 or mm w. .g.

= 2094 Pa

It will be observed that a forward bladed fan produces more depression.

(B) The effective depression is the pressure actually developed by the fan and expended on overcoming the resistance of the mine. It is the depression measured by and shown on the fandrift w.g. with one limb of U-tube facing the air current. For a given fan speed, the effective depression varies with the quantity of air flowing i.e. it depends on the equivalent orifice (or mine resistance) on which the fan is working. Each type of fan has its own pressure volume characteristics curve which shows the relationship between the effective w. g. and the volume of air when this changes due to a change in the mine resistance.

(C) The manometric efficiency is the ratio between the effective w. g. and the theoretical w. g. produced by a fan with the same dimensions and running at the same speed. It is therefore a variable figure unless it relates to a particular point on the characteristic curve. Quoted values generally refer to the design point of the fan at which it runs at its maximum mechanical efficiency but other operating points may be selected if desired, e.g. when the fan is working on a closed fan drift.

(D) H. P. of ventilation : The useful power developed by an air current or the "H. P. in the air" is found from the following equation.

Air h. p. $= \frac{PQ}{75} = \frac{RQ^3}{75}$, where

P = ventilating pressure or total pressure in kgf/m^2

Q = quantity in m^3/sec

R = resistance in kilomurg

Power of ventilation or air power is the rate at which work is done to maintain the air flow through a system and is given by the formula

$AP = RQ^3 \times 10^{-3}$ kW where S. I. units are used.

Here AP is air power in kW

R is resistance in S. I. units

Q is in m^3/sec.

(E) Mechanical efficiency : No fan or engine is perfect, i.e. it never gives out as much useful work as is put into it; and the ratio (or fraction) obtained by dividing the power output by the power input is termed the "mechanical efficiency". It is always less than unity i.e. less than 100%. It is not, of course, a constant figure for a given fan but varies with the resistance (or equivalent orifice) of the mine on which the fan is working. A fan only maintains a high efficiency over a certain range of mine resistance.

Now the useful power output of a fan is the "H. P. in the air" calculated as already explained, whilst for the input, we may consider either (a) the power given to the fan shaft, or (b) the power put into the driving engine or motor. In practice, it is easier to measure the power input either from indicator diagrams (in the case of steam engine) or from instruments (voltmeter and ammeter, or a wattmeter) in the case of an electric motor. Knowing the air H. P. and also the H. P. input to engine or motor, we can find the ratio :

$$\text{Overall mechanical efficiency or useful effect} = \frac{\text{H. P. of ventilation}}{\text{H. P. input to engine or motor}}$$

$$\text{Fan efficiency} = \frac{\text{H. P. of ventilation or Air H. P.}}{\text{Fan shaft H. P.}}$$

If the fan is direct driven, the Brake H. P. (or output) of the motor or engine may be accepted as the fan-shaft H. P. If the fan is driven indirectly through gearing (belt, rope, chain or toothed gearing) the efficiency of the drive must be estimated to arrive at the fan-shaft H. P.

Equivalent orifice of a mine :

The equivalent orifice of a mine is the area of an imaginary opening in a thin plate which offers the same resistance to the passage of air as is offered by the mine itself. It is the area of opening through which would pass the same quantity of air as passes through the mine under the same pressure. It may be calculated by the formula :

$$A = \frac{0.385\,Q}{\sqrt{P}}$$

where A = Eq. orifice in m^2

Q = Quantity of air flowing in m^3/sec

P = Pressure absorbed in kgf/m^2 or in mm of w.g.

The above formula is very nearly equal to the formula used when SI units are given. The formula in SI units is

$$A = \frac{1.2\,Q}{\sqrt{P}} \quad \text{i.e.} = \frac{1.2}{\sqrt{R}}$$

Where A is area of orifice in m^2

Q is quantity of air flowing in m^3/sec

P is pressure causing flow, in Pascals.

R is resistance of mine in units of $N\,s^2\,m^{-8}$

Example (S. I.) :

In a coal mine a longwall district is supplied air by a main gate road. The return airway is parallel to it. The two roads are 30 m apart and interconnected at 50 m intervals. The stoppings in the interconnections allow some leakage so that the quantities meausred at the inbye and outbye ends of the main intake are 5 and 7.5 m^3 per sec. respectively. Calculate the pressure across the main intake and return at the outbye end if the resistances of the main intake, main return and the working district are 7.5, 7 and 0.5 resistance units (SI) respectively.

Answer :

Pressure drop in the main intake = $7.5 \times 5 \times 7.5 = 281.25$ Pa.

Pressure drop in the main return = $7 \times 5 \times 7.5 = 262.5$ Pa.

Pressure drop at longwall face = $0.5 \times 5^2 = 12.5$ Pa.

Therefore total pressure drop = $281.25 + 262.5 + 12.5$

= 556.25 Pa.

Characteristic Curves :

A characteristic curve is a curve which shows how the magnitude of one quantity varies with changes in some other related quantity. In the case of a fan for every speed, a set of curves can be drawn to show variation of fan-drift pressure, B. H. P. of the prime-mover and mechanical efficiency of the fan, with changes in volume of air circulated by it. These curves are indicative of the performance or the characteristic of the fan and as such, are referred to as the "characteristic curves" of that particular fan. Sometimes the term "fan characteristic" is used to denote the first (pressure-volume curve) of the set of these curves.

Mine characteristic curves :

Just as the main fan characteristic curve is a pressure volume (P-V) curve showing the relationship between (a) the effective fan-drift w.g. produced by a fan and (b) the volume of air delivered by it at constant speed as the equivalent orifice increases (or the mine resistance decreases), so the mine characteristic curve is one showing the relationship between fan drift w. g. and the volume of air flowing in a mine of given equivalent orifice (or mine resistance) and based on $P = RQ^2$

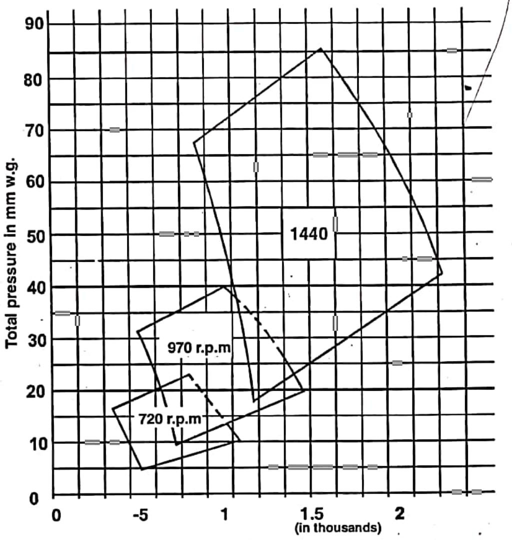


Fig. 2.14 Pressure volume characteristic of fan; type V. F. 1200

Fans in series and parallel :

The results obtained by installing fans in series or in parallel, depend

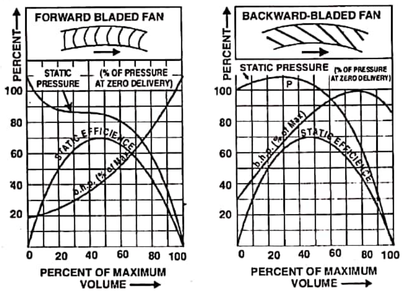


Fig. 2.15 Characteristic of centrifugal fans

very greatly on the characteristic of the fan and the mine concerned, as shown by their characteristic curves. Let us consider two identical fans for series/parallel running (Fig. 2.16)

Let XY = the characteristic curve of a single centrifugal fan.

Let ZY = the series characteristic

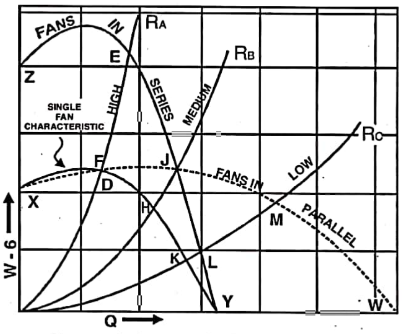
Let XW = the parallel characteristic

Let R_A = a mine of high resistance

Let R_B = a mine of medium resistance

Let R_C = a mine of low resistance

The series characteristic ZY is obtained from the single fan characteristic and is such that the pressure generated by two fans in series for any given volume of air flowing is twice that for the same air flow through a single fan.



Characteristic curves for fans in series and in parallel.

Fig. 2.16 Characteristic curves for fans in series and in parallel.

The parallel characteristic XW is plotted on the basis that the quantity of air flowing with two fans in parallel is twice that for the same pressure with a single fan.

It will now be seen that with a single fan, the operating point of the fan is a D in the case of mine, R_A, H in the case of mine R_B; and K in the case of mine R_C.

Fans in parallel :

In mine R_A, the operating point moves from D to F, showing that both w. g. developed and quantity flowing remain almost unaltered. In mine R_B, the operating point moves from H to J, exactly as in the series case, so that the same results are obtained on this mine by series and parallel arrangements alike. In mine R_C, the operating point moves from K to M, showing that the w. g. developed is now more than doubled (due to the fans now operating at a higher efficiency under the new conditions) and the quantity of air flowing is increased by about 50%.

Fans in series :

The delivery of air to the mine is given by the intersection of the series fan characteristic with the mine characteristic concerned. In mine R_A the operating point of the fans moves from D to E, and this shows that the w.g. developed by the two fans is about 1.8 times that of a single fan and the quantity of air flowing is increased by more than 30%. In mine R_B the operating point moves from H to J, showing that the w.g. developed is about 1.35 times that of a single fan, and the quantity of air flowing is increased by less than 20%. In mine R_C the operating point moves from K to L giving only a negligible increase in w. g. and quantity.

Conclusions to be drawn : It will now be evident that

1. In a mine of high resistance, the series arrangement gives a considerable increase in quantity of air flowing (possibly more than 30% increase), whilst the parallel one gives a negligible increase, or even, in some cases, a decrease depending on the shape of the fan characteristic curve.
2. In a mine of medium resistance both the series and parallel arrangements give the same result in the region of about 20% more air.
3. In a mine of low resistance, where the single fan is operating at a very low efficiency on an unsuitable part of its characteristic curve, the series arrangement gives a negligible increase in quantity, and the parallel arrangement is to be preferred.

Table 1 : Fan Specifications

The specifications of axial flow fans manufactured by M. A. M. C. for auxiliary ventilation in underground mines, and for ventilation of a sinking pit or tunnel are given below.

Technical Specifications of AV Series Fans of M. A. M. C.		
Type	$AV_1/500-2S$	$AV_3/500-2S$
Effective continuous rating of the electric motor, kW	11	12.5
Electric supply	500/550V / 400/440 V, 3 phase, 50c/s	500 / 550 V, 3 phase, 50c/s
Speed, R. P. M.	2910	2910
Number of stages	2	2
Impeller diameter, mm	510	510
Pressure kg/m² (mm of W. G.)	240 - 50	250 - 60
Capacity m³/min	145 - 225	180 - 370
Max. efficiency, %	70	70
Approx. wt. of fan, kg	310	336

Table 2 : Specifications of main mine fans of M. A. M. C.

Technical Specifications of MV Series Fans	Units	$MV_1 - 1.8$ Single stage	$MV_2 - 1.8$ Double stage	$MV_1 - 3$ Single stage	$MV_2 - 3$ Double stage
Rotor dia.	mm	1800	1800	3000	3000
Speed	RPM	750 1000	750 1000	500 600	500 600
Air Quantity	m³/sec	17-70 23-93	17-70 23-93	43-222 52-265	43-222 52-265
Water gauge	mm	132-47 235-83	265-93 470-165	186-61 270-90	312-95 450-135
Consumed air power	kW	19-95 45-225	50-190 100-450	70-492 120-852	116-820 200-1420
Max. total efficiency	%	80	80	80	80

The fans are provided with a unique arrangement for varying the pitch of the blades without any need to tighten or loosen nuts and bolts.

The fans possess a non-overloading type of power characteristic. The air quantity on reversal of the fan is nearly 35-40% of the normal quantity.

Model standing orders in the event of stoppage of the main mechanical ventilator in mines :

Coal Mines Regulation, 134 and Metalliferous Mines Regulation, 133 give directions on the procedure to be followed when the main ventilation fan is to be stopped, or has stopped. The fan attendant has to inform the attendance clerk when the fan stops. The overman/foreman, Asst. Manager and the engineer also should be informed. The officers should ascertain more facts about fan stoppage and should take a decision whether to withdraw the workers from the mine. If the fan stoppage is likely to last a long period, it is essential to withdraw the workers out of the mine and stop electric supply to underground mine. The fan attendant should open the doors of the main fan house on the surface. Instructions relating to procedure in the event of fan-stoppage, should be displayed at prominent places on notice boards, both above and below ground.

Rock dust in mines :

The underground environments, to be healthy and suitable for efficient working, have to be free from rock/coal dust beyond some permissible limits. The effects of rock/coal dust and the manner of their control have been described in other chapters.

QUESTIONS

1. What is relative humidity ? How does it affect the working efficiency of a worker in an underground mine ? What steps should be taken in a mine to keep the relative humidity within tolerable limits for efficiency in working ?
2. How is nautral ventilating pressure produced? Explain by sketch "motive column" as applied to N.V.P.
3. **Define the following terms :**
   i. *Theoretical depression,*
   ii. *Ventilating pressure of a fan,*
   iii. *Static pressure,*
   iv. *Manometric efficiency of a fan,*
   v. *Mechanical efficiency of a fan,*
   vi. *Kilomurg, and*
   vii. *Equivalent orifice of a mine.*

4. How does the speed of a mine fan affect the air quantity, water gauge and H. P. of the fan motor ?

   What are the different ways to vary the quantity of air going down the mine ?

5. Compare the forcing fan with an exhaust fan.

6. What is meant by fan characteristics? What is a non-overloading characteristic ?

7. Draw up the code of Standing Orders to be observed in the event of stoppage of the main ventilating fan of a gassy mine.

8. What do you understand by "adequate ventilation" in a coal mine? What should be the air velocity at the working face and other important places in a coal mine ?

9. A 200 m long longwall face is ventilated by two gate roads each 300 m long. Calculate the quantity that will flow along the face when a ventilating pressure of 150 Pa is applied acros the gate roads at the outbye end. The face has a resistance of 0.6 $Ns^2 r^{-8}$ per 100 m length. Neglect leakage between gate roads.

O O O

III

CHAPTER 3

DISTRIBUTION OF AIR AND ITS CONTROL

Air in a mine is coursed to the working places by the use of brick stoppings, doors, brattice cloth, air-crossing, air pipes and regulators. The leakage of air should be avoided and it should be noted that a good proportion of air leaks through faulty arrangement of cover the shaft mouth and its lifted up by the detaching hook when the cage ascends to the surface. The best arrangement is an air lock which encloses the shaft fully. Conventional air lock has two doors on each outlet so that when one is opened, the other remains closed. The walls of the air lock have large windows with glass for natural light.

The various devices used for distribution and control of air in a mine are shown on the mine plane by signs, of which the most common are shown in the appendix of this Part A.

4 **Ventilation Stoppings** have to be made of incombustible material and therefore are of brick or stone. The thickness of a ventilation stopping is minimum 38 cms of brick or stone in lime or cement and plastered to prevent leakage of air (D.G.M.S. Cir. No. 17 of 1964).

4 **Aircrossings** are constructed where intake air and return air currents have to cross each other. These should be leakage-proof, fireproof and have ample cross-section. Normally an air crossing is constructed at a place where it has reasonably long life and the ground free from rock movement. Air crossing m gassy coal mines (deg. 2 and 3) should be explosion-proof (Fig. 3.1).

~~4~~ **Doors** are used on those roads which are required for haulage or travelling purposes. If a door is no longer required for ventilation it should be taken off the hinges & kept away. The ventilation door must be arranged to close automatically and with this object the frame of the door is so fitted that the top leans about 50mm–75mm forward in the direction of air pressure. The door, of course, should open against the intake air so that the air pressure normally keeps it closed.

4. How does the speed of a mine fan affect the air quantity, water gauge and H. P. of the fan motor ?

   What are the different ways to vary the quantity of air going down the mine ?

5. Compare the forcing fan with an exhaust fan.

6. What is meant by fan characteristics? What is a non-overloading characteristic ?

7. Draw up the code of Standing Orders to be observed in the event of stoppage of the main ventilating fan of a gassy mine.

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O O O

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CHAPTER 3

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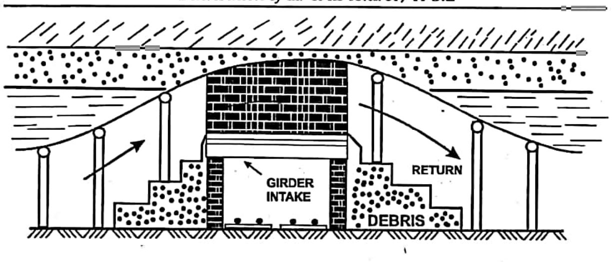


Fig. 3.1 Aircosting

A regulator is a window of adjustable opening in a brick stopping. The shutter of the regulator can be locked in position to prevent tampering by workers. Introduction of a regulator in a roadway increases the resistance to air current and it should, therefore, be used on the return side of a district whose ventilation has to be reduced and fixed in a place where all the air of the district has to pass. Obviously a regulator cannot be placed on haulage roads. The regulator has the effect of reducing the air flowing in the regulated split and at the same time increasing (although not to the same extent) the volume of air flowing in the unregulated split. If a return airway is common to two districts and one of the districts has to be regulated, the regultor must be placed in the intake airway of the split to be regulated. It such intake airway is to be used for traffic of tubs or workers it is not possible to fit the regulator in a stopping and it should therefore be fitted in a ventilation door.

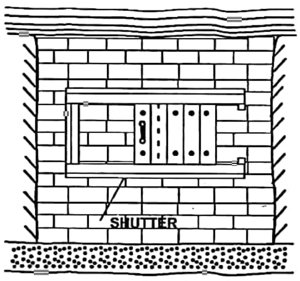


Fig. 3.2 A regulator

Brattice partitions are erected to course the ventilating current upto the working face in a narrow heading e.g. in the main heading or companion heading in bord and pillar working. Only the roads inbye of the last connecting gallery or roadway should be fitted with a brattice partition which is, however, useful only for a limited length. Long stone drifts and headings, if partitioned by brattice cloth, will not give adequate quantity of air at the face and an auxiliary fan is usually the convenient arrangement for properventilation.

Ascensional and descensional ventillation :

Ascensional ventilation implies taking the intake ventilating air to the lowest point of a district or face and allow it to travel to higher levels to ventilate the district or face before it goes to the return. Descensional ventilation implies taking the air to the rise side of a district and allow it to travel to lower levels as it ventilates the working places. In Fig. 3.3 the split A has ascensional ventilation.

Ascensional ventilation is preferable because :

1. Firedamp being lighter than air, is readily carried to higher levels.
2. Natural ventilation pressure assists the fan ventilation because the air gets hot during its travel in the mine and has a tendency to go to higher levels.
3. If the fan should stop, the air will continue to flow in the same direction by natural ventilation.

Descensional ventilation has some advantages in hot deep mines. The main advantage is the air has not to pass over water drains of the dip side and it reaches the working face in drier and cooler condition. At one deep mine in Raniganj field worked by advancing longwall and hydraulic sand stowing the dip side face of the double unit longwall face had a relative humidity at 93% with ascensional ventilation because the intake air was passing over drains of water coming out from stowed goaf. When that particular face was ventilated by descensional ventilation, the relative humidity came down to 65%.

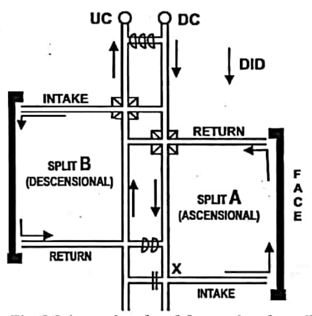


Fig. 3.3 Ascensional and descensional ventilation.

Homotropal and Antitropal ventilation :

When the air and mineral flow in the same direction, the ventilation is known as homotropal ventilation.

When air and mineral flow in opposite directions the ventilation is said to be antitropal. This would apply to split A in Fig. 3.3. where coal flows on conveyors in the direction opposite to air current (conveyor is assumed in intake road).

With homotropal ventilation the velocity of air relative to coal is less as compared to that in antitropal ventilation. The amount of coal dust at the face is therefore less with homotropal than in case of antitropal ventilation. Homotropal ventilation has other advantages in respect of humidity and dealing with fire in mines having longwall faces.

Splitting :

With a view to have fresh air in a district, unpolluted by human breathing or gas emission in other districts, a branch of air current (also called split) is taken from the main air current which travels inbye from the DC shaft. Each ventilating district is ventilated by a separate split. **A ventilating district,** as defined in the Regulations, means such part of a mine below ground as has an independent intake airway commencing from a main intake airway, and an independent return terminating at a main return airway. If an intake airway has two, three or more splits branching off it, the intake airway has the splits in parallel and the combined frictional resistance of the main intake airway and splits is much less than that of the main intake airway alone. The resistance follows *nearly* the same laws as are applicable to electrical resistance in parallel and in series.

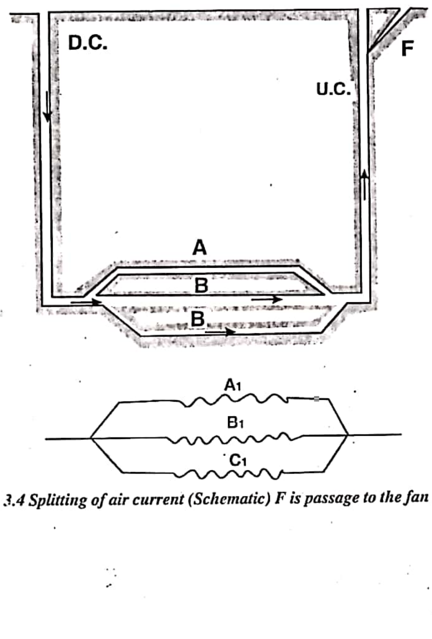


Fig. 3.4 Splitting of air current (Schematic) F is passage to the fan drift.

Splitting supplies fresh air to each ventilating district. As the overall resistance of the mine is reduced, for the same water gauge produced by fan and the same H.P. of motor, a much larger total quantity of air circulates through the mine. Air velocities are reduced in the roadways and at the face and coal dust formation is lessened. The reduced velocity of air has reduced drying effect on the coal dust. Fire, gas emission, on roof fall in a district, resulting in derangement of its ventilation does not affect other districts. Splits reduce the doors on haulage roads but increase the number of aircrossings.

In a mine a large number of splits result in proportionately less quanity in each district and consequently reduced velocity which may not be able to clear off gas or may not provide adequate ventilation to the district.

The number of splits should be such that the air velocity is not unduly low and that the ventilation is *adequate* as defined in the mining Regulations. As far as practicable all the splits should be of the same resistance but some split may be required to carry more air because of its extensive working faces or more gas emission. The minimum air velocity at the working faces is stated in earlier chapter (see mining Regulations).

The combined resistance of an underground system of roadways having one or more splits is given by the formula.

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} \ldots\ldots.. + \frac{1}{\sqrt{R_n}}$$

where R is the combined (or equivalent) resistance of the system of roadways and R_1, R_2, R_n etc. are the resistance of individual splits. The effect on the quantity flowing is illustrated by the following examples.

Example 1 :

A total quantity of 100 m³/min of air is passing through two splits. One airway is 2.5m × 1.5m and 100m long, and the other, with similar lining, is 2m × 1.5m and 125m long. Calculate the quantity of air passing in each split.

Answer : Since the two splits are subjected to the same pressure, and the nature of lining is the same, from the equation.

$$P = \frac{KSQ^2}{A^3}, \text{we have}$$

$$Q \alpha \sqrt{\frac{A^2}{S}} \text{ or } Q \alpha \sqrt{\frac{A^3}{\text{perimeter} \times \text{length}}}$$

$\therefore$ Relative quantity in split No. 1 $= \sqrt{\dfrac{(2.5 \times 1.5)^3}{2\,(2.5+1.5) \times 100}}$

$= 0.2567$

Relative quantity in split No. 2 $= \sqrt{\dfrac{(2 \times 1.5)^3}{2\,(2+1.5) \times 125}}$

$= 0.1757$

Sum of the relative quantities = 0.4324

$\therefore$ Actual quantity of air in split No. $1 = \dfrac{0.2567}{0.4324} \times 100$

$= 59.37\ m^3/min.$

$\therefore$ Actual quantity of air in split NO. 2 = 100 – 59.37

$= 40.63\ m^3/min.$

[Check : quantity in split No. $2 = \dfrac{0.1757}{0.4324} \times 100$

$= 40.63\ m^3/min$

Example 2 :

Three splits in parallel, of similar cross-section and same type of roadway surface, are respectively 300m, 600m and 900m long. Calculate the quantity of air which would flow in each if the total quantity is 200 m³/min.

Answer : From the relation $P = \dfrac{KSQ^2}{A^3}$

$$Q^2 = \frac{PA^3}{KS} = \frac{PA^3}{K \times L \times \text{Perimeter}}$$

Since A, K and perimeter are constant here,

$$Q \propto \frac{1}{\sqrt{L}}$$

$$\therefore \quad Q_1 : Q_2 : Q_3 \ \frac{1}{\sqrt{300}} : \frac{1}{\sqrt{600}} : \frac{1}{\sqrt{900}}$$

or $\quad Q_1 : Q_2 : Q_3 : \dfrac{1}{\sqrt{1}} : \dfrac{1}{\sqrt{2}} : \dfrac{1}{\sqrt{3}}$

or $\quad Q_1 : Q_2 : Q_3 :: 1 : 0.709 : 0.579$

Sum of the relative quantities = 1 + 0.709 +0.579 = 2.288

$\therefore$ Quantity in split No. $1 = 200 \times \dfrac{1}{2.288} = 83.03 m^3/min$

Quantity in split No. $2 = 200 \times \dfrac{0.709}{2.288} = 61.97\ m^3/min$

$\therefore$ Quantity in split No. 3 = 200 – (88.03 + 61.97)

$= 50m^3/min.$

[Check : Quantity in split No. $3 = 200 \times \dfrac{0.579}{2.288}$

$= 50.6\ m^3/min$]

Leakage of Air :

The sources of leakage of air that goes down the mine are :

1. Doors of the fan drift and air lock. If the air lock is provided with glass windows to admit natural light at the pit top, a broken glass pane causes heavy leakage.
2. Where air lock is not provided, the space between the cages and shaft walls and also between the cages and pit top leading level is a source of leakage. If the pit top landing level is covered by a wooden lid which is lifted by the ascending cage, the arrangement permits of substantial leakage and it is heavy when the lid is lifted by the winding rope capel and the cage is resting at the pit top.
3. Ventilation stoppings, ventilation doors and air crossing.
4. In the longwall method of coal mining, the roadside pack walls if the goaf is not solid stowed.
5. Broken or crushed pillars of coal.
6. Wrong siting of underground booster fan.

Because of leakage the total quantity of air that reaches the working faces is only 25 to 50% of that circulated by the main fan. The bord and pillar method of coal mining is notorious for poor ventilation at the working faces compared to the longwall method.

Under average coditions 45 to 55% of air circulated by fan reaches the working face in an in-the-seam-worked coal mine, slightly higher (55–65%) in the case of coal mines worked by horizon mining, and still higher in metal mines. Periodical ventilation surveys for the quantity can give an idea of the leakages which can be reduced, if not completely avoided, by the following measures.

1. Air locks at the pit top should be of proper design.
2. Doors of the air locks and of the fan drift should have rubber lining for leakage-proof closing.
3. Precautions should be taken to see that both the doors of an air lock are not opened stimultaneously and this point should be impressed upon the workers. If possible the doors should be mechanically interlocked so that when one is open, the other can-not be opened.
4. Have the underground intake and return as far apart as possible and have very few connections between them. If possible, the main return and main intake of the mine should be kept in different seams.

   Care must be taken to locate major airways in strong undisturbed ground to reduce leakage.
5. All the underground ventilation doors, ventilation stoppings and aircrossing should be well constructed and maintained.
6. In longwall method of coal mining roadside packwalls should be well constructed to avoid leakage through them.
7. Where the pillars of coal/mineral are broken or cracked, sometimes due to heavy roof pressure as in the vicinity of depillaring/stopping area or near fault zones, they should be coated by a spray of cement-mortar.
8. For reducing leakage it is preferable to use a large number of low pressure fans in series than a single fan producing high pressure.

A system of ventilation normally adopted in metal mines is the *boundary ventillation system*. This is possible where the D.C. and U.C. shafts are located at opposite ends of the property and the air from the intake to the return side is practically eliminated. The fresh and cool intake air goes to the lowest levels where rock temperature is the hottest.

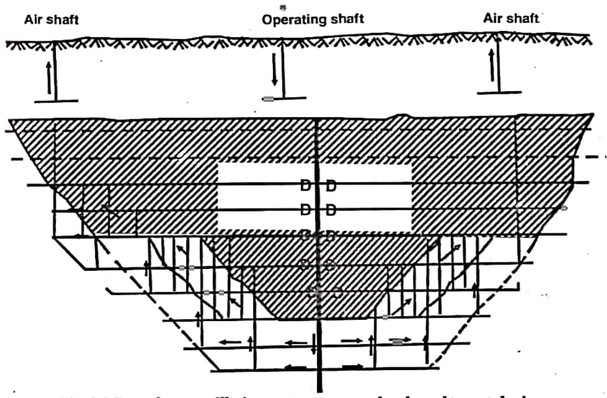


Fig. 3.5 Boundary ventillation system commonly adopted to metal mines.

In most of the coal mines, however, the standard practice is to use adjacent shafts as intake and return shafts and very often the main intake and return airways underground are also adjacent, resulting in much leakage and necessitating use of many controlling and regulating devices.

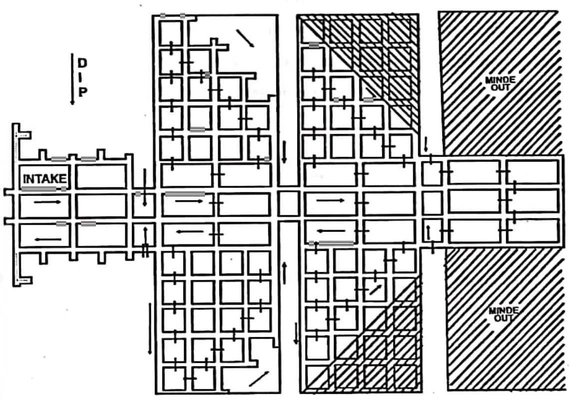


Fig. 3.6 Air distribution in bord-and-pillar workings.

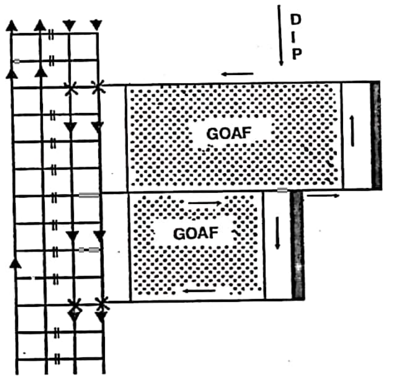


Fig. 3.7 Air distribution in longwall workings; double-unit face.

4 # Air Locks :

At a number of mines the type of air lock provided consists of only a simple covering at the top of a shaft which is lifted up by the upcoming cage. In this design heavy leakage of air, as much as 30% of the quantity of air circulated by mechanical ventilator, takes place when the cage is resting at the pit top. Such a design of the airlock therefore cannot be considered as suitable.

The suitable airlock designs are :

1. Standard type of air-lock at the top of a shaft enclosing part of the pit top.
2. Guillotine type of doors which are provided in a vertical steel box fitted within the headgear.
3. German type of a airlock which forms an airlock inside the shaft.

The German type air-lock below banking level.

Compared to the other air locks the German type air-lock is not constructed above the banking level but below it. The pit top is completely covered with steel joists and thick wooden planks except for two rectangular openings for passage of the cages. In these openings two hollow boxes of M.S. plates, open at the top and bottom, are suspended rigidly from the above mentioned shaft top covering and their length is equal to the height of the cages. The pit top banking level is flush with the pit top covering and the space between the shaft walls and pit top wooden covering is closed by rubber linings.

The hollow steel boxes suspended from the pit top covering are also lined with rubber sheets throughout the length and also at the top and bottom openings.A trapezoid shaped covering of aluminium to cover the bridle (suspension) chains is provided and it has a small opening for the passage of safety hook. This trapezoidal shaped aluminium box rests on the pit top covering the corresponding cage at the pit bottom but can be lifted by it when ascending. The small opening at the top of the trapezoidal aluminium body is covered by a separate wooden lid with a small hole for the winding rope. When the ascending cage approaches the banking level the safety hook first lifts the wooden lid over the aluminium boxing which is itself lifted, a second later, by the ascending cage. The space of guide ropes between adjacent cage is also covered with a wooden frame lined with rubber-sheets and small openings are provided in the frame for the requirment of guide rope shoes as the cage moves up and down. The rubber linings at various openings prevent leakage of air.

An air lock of this type is provided at Chinakuri Colliery.

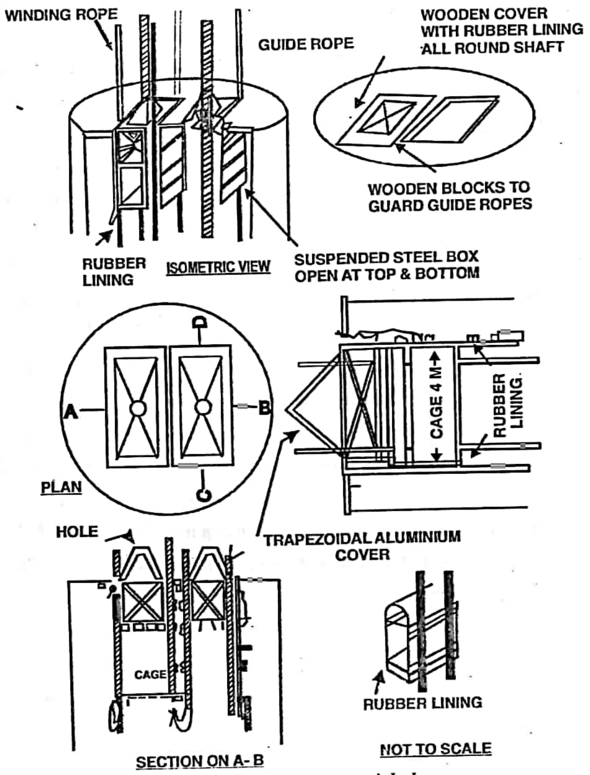


Fig. 3.8 The German type air lock.

Booster and auxiliary fans :

A common arrangement to improve the quantity of air in one or more districts having high resistance, without increasing the total quantity of air circulated by the main surface fan, is the installation of a booster fan.

A booster is a more or less permanent installation designed to pass the whole of the air circulating in the district or districts concerned. An auxiliary fan, on the other hand, is a more or less temporary installation designed to pass only a proportion of the air circulating in the district and it is used for

i. long headings or stone drifts which are carried in advance of the normal ventilating current and thus constitute dead ends, or

ii. clearing of roof fall which has obstructed the normal air supply.

A booster fan may be placed in the return to act as an exhaust fan or it may be placed in the intake to act as a forcing fan. It is usually of axial flow type and the manner of its installation is shown in Fig. 3.9.

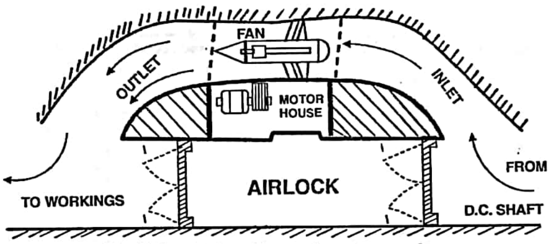


Fig. 3.9 Installation of booster fan : air screw fan.

The axial flow fan placed in the byte pass of an intake air way is shown in Fig. 3.9. It is a forcing fan. A door provides access to the motor room and an air lock provides for passage of men and materials (if equipped with haulage track). The installation advantages, viz. (i) compactness (ii) the fan can be mounted directly in the path of ventilating air as air flows axially through it (iii) a small size axial flow fan up to 0.5m diam. fitted with a direct driving motor can be mounted in the canvas ventilation tube used for ventilating a long split.

Auxiliary : 1 As already stated an auxiliary fan is used in the mines for the ventilation of development headings, for narrow workings in coal and for stone drifts which are carried in advance of the normal ventilating current. Axial flow fans are preferred to centrifugal fans as auxialliary ventilators. An auxiliary fan should be so installed that there is no possibility of recirculation of the air supplied by it to the working place. The quantity of air taken by an auxiliary fan should not exceed 1/3rd of the quantity in the air current from which the fan takes its supply. (D.G.M.S Cir. No. 82 of 1963 places this limit at $\frac{1}{2}$ of the air current from which the fan takes its air supply.) The site of installation should be a sufficient distance outside the actual heading it is intended to ventilate. A forcing fan therefore should be placed on the intake side and an auxiliary exhausting fan on the return side of the drift it has to ventilate and the minimum distance between the fan and the corner of the drift/road to be ventilated should be 5m as shown in the Fig. 3.10.

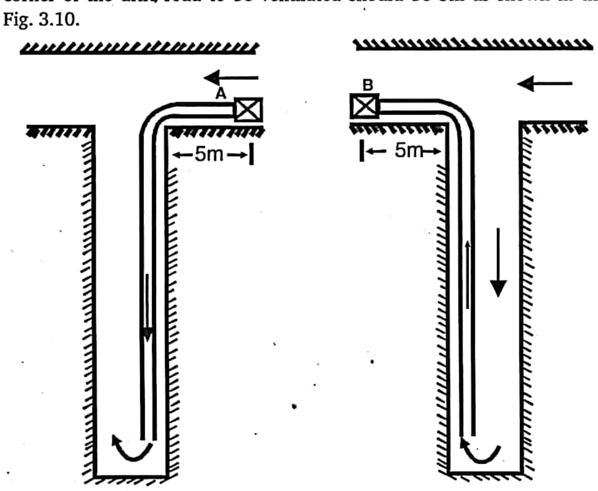


Fig. 3.10 Installation of auxiliary fan, A – a forcing fan; B–an exhaust fan.

All metallic parts of an auxiliary fan installation should be carried to avoid build up of electrostatic charges which can ignite methane by sparking.

The quantity of air to be circulated by an auxiliary fan depends upon the rate of gas emission in the drift/roadway and varies from one installation to another. In general terms it may be said that the quantity per minute should be $7m^3$ per m^2 of the working face.

Auxiliary ventilators, compared to boosters, are much smaller in size and are used for ventilating shafts, drifts, tunnels and long development headings, particularly in metal mines, where the small size of working place does not permit bratticing.

Auxiliary fans are not provided with arrangement for volume control as it would make the fan heavy but on some large fans an inlet-vane or damper control is provided to regulate the air quantity.

Contra-rotating Axial-Flow fan :

This fan consists of two impellers rotating in opposite directions. The impellers are enclosed within a cylindrical casing and each impeller is driven by its own motor. The downstream impeller assists in restoring axial flow of the air which has received circular motion when passing through the upstream impeller. This helps the efficiency of the fan, and it can develop upto times the w.g. developeed by a single impeller without guide vanes. The fan w.g. can be regulated by allowing downstream impeller to be idle. When reversed the fan produces nearly 60% of the normal air quantity. These fans can be coupled together in series for getting higher water gauge. (Fig. 3.11).

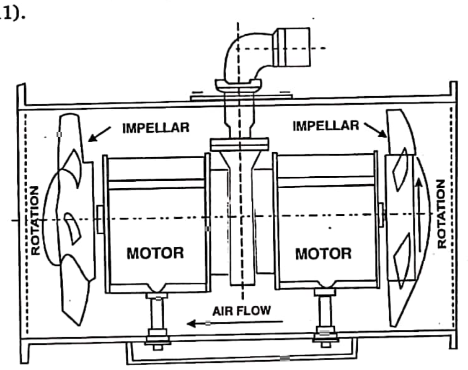


Fig. 3.11 Contra-rotating axial-flow fan.

Air Tubes :

The air of the auxiliary fan (forcing type) is supplied to the working face through sheet steel air-tubes or through canvas tubing which is made of fabric impregnated with rubber to make it airtight and resistant to dampness and mine gases. The canvas tubing is light, easy to handle and can be rolled up for easy transport. Normally it is available in lengths of 8 or 16m. Just before short-firing the last few lengths near the face should be withdrawn to avoid damage by blasting. In the case of exhausting auxiliary ventilator the usual canvas tubing cannot be used and only the sheet steel air-tubes have to be installed; the air reaches the working face through the entire cross-section of the road and foul air returns through the sheet-steel tubing to the exhausting fan. Flexible canvas or PVC ducting incorporating wire armouring embedded in the fabric is available for exhaust ventilation. Such ducting is expensive but does not collapse under suction. The air tubes of an auxiliary fan can be suspended from hangers fixed in the roof-supporting steel arches, or they can be supported on long steel spikes inserted into the sides of a drift/roadway.

D.G.M.S Circular No. 82 of 1963 contains the following instructions on the use of auxiliary fan underground.

"With a view" to preventing re-circulation of air which could lead to a dangerous situation in a mine, it is recommended that the following rules should be observed in all mines where auxiliary fans are installed.

1. To prevent recirculation of air the quantity of air taken by an auxiliary fan shall not exceed one half of the quantity in the current from which the air passing through the fan is drawn provided that this rule shall not apply where the inlet and outlet ends of the duct are separated by doors or seals.
2. All auxiliary fan installations, which draw air from an intake airway and feed it into a return airway, shall be examined once at least in every week by the Ventilation Officer to check that the quantity of air passing in the intake airway on the inlet side of the fan or fans (if more than one fan is drawing air from the same airway) is sufficient for the proper ventilation of the inbye workings where fan/fans is/are running.
3. Before any auxiliary fan is installed the quantity of air flowing in the airway at the point where it is proposed to install it, shall be measured (a) to avoid the possibility of recirculation, (b) for the proper ventilation of inbye district when the fan is running.
4. (i) If it is necessary to regulate an auxiliary fan, it shall be done in such a way as to prevent unauthorised or inadevertant alternation.
   (ii) Fans delivering air through flexible ducting shall not be regulated by constricting the ducting.
   (iii) Fans with rigid ducting shall not be regulated by placing loose obstructions such as bricks or stones in the ducting.
5. No person other than an official of the mine, ventilation officer or a person authorised by the manager to do so shall regulate the quantity of air passing through or delivered by any auxiliary fan.

The main provisions of the above instructions are included in the amendments to mining Regulations in 1972.

Booster fan and neutral line :

A booster fan, as already stated is installed to increase the quantity of air passing in a split or ventilating district of high resistance, Such split may be either a long single split or one in parallel with another split. When the fan is installed the w.g. on the return side of the split, at least for some distance, increases in relation to that on the intake side and there is some position in the ventilation system of the split where, after the installation of the booster fan, there is no pressure drop between the intake and the return. Such position is known as the *neutral line* and it is shown as the line NL in Fig. 3.12. The Fig. shows a single district which has to be ventilated by a booster to be installed on the return side. Let us assume that drop of pressure from the junction of intake and the split, X, to the junction of the split with the return, Y, is 50mm. A is the midpoint of the longwall face.

Before the installation of a booster every point on the intake is at a higher pressure than any point on the return side and the pressure drop, for the sake of simplicity, may be assumed to be at uniform rate from X to A (25mm) and from A to Y (25mm), the total drop from X to Y being 50mm, represented on the graph by the dotted line XA and AY.

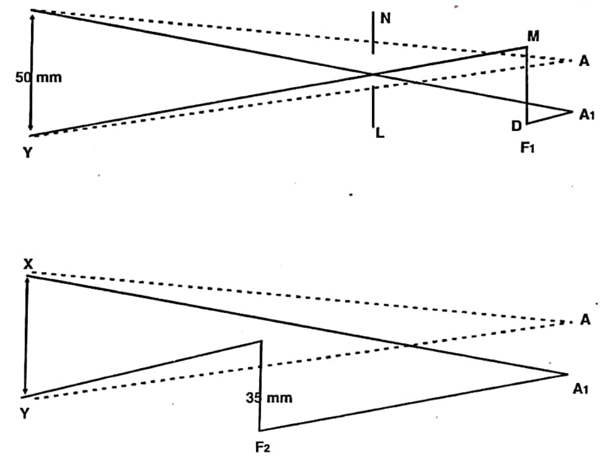


Fig. 3.12 Booster fan and the neutral line.

F_1 and F_2 are two alternative positions of the booster.

After the installation of the booster at the point F_1, the general effect is to

(a) increase the volume of air in the split,

(b) increase the pressure drop from X to A to some value over 25mm as $P = RQ^2$. The higher pressure drop from X to the face A is shown by a steeper line XA_1. The same quantity flows from the face to the booster and therefore the rate of fall of w.g. represented by the line A_1 D from the face to F_1 is the same as from X to A_1. At D the booster boosts up the water gauge by, say 35mm, represented by the line DM. The total air leaving the booster at F_1 passes to the point Y and the rate of fall of water gauge, say, per m length of road is the same as from X to A_1, A_1 to D, and from M to Y. The point P lies on the neutral line NL and at this point there is no pressure difference between intake and the return. At any point between F_1 and P the return w.g. is higher than intake w.g. and a leakage of foul return air can take place on the intake across the ventilation stoppings and other places of leakage. If the booster is placed close to the neutral line NL between P and F_1 the leakage is negligible as the pressure difference between intake and return is then small. If the booster is placed at some other point outbye of NL, say at F_2, the neutral line is not developed and the intake pressure always exceeds that of the return. One disadvantage, however, is that there is considerable pressure difference between the intake the return and the leakage, if it takes place, is maximum (from intake to return).

Example (S.I.) :

Two ventilation splits A and B pass 15 and 20 m³/s of air respectively with a pressure drop of 500 Pa across them. The trunk airways consume a pressure of 300 Pa. If the air flow in the two splits is to be equal by installing a regulator, calculate the size of the regulator, assuming the fan pressure to remain constant before and after installation of a regulator. What will be the air flow in the mine after fitting of the regulator.

Answer :

$$\text{Resistance of split A, } R_A = \frac{500}{15^2} = 2.22 \text{ units}$$

$$\text{Resistance of split B, } R_B = \frac{500}{20^2} = 1.25 \text{ units}$$

$$\text{Resistance of trunk airways, } R_T = \frac{300}{35^2} = 0.245 \text{ units}$$

Let the new quantity in the mine after installation of the regulator in split B be Q m³/s.

Let resistance of regulator to R_R units.

When the flow in the two splits is equal, $R_B + R_R = R_A$

Or, $R_R = R_A - R_B = 2.22 - 1.25 = 0.97$ resistance units.

$$\text{Area of regulator} = \frac{1.2}{\sqrt{R_R}} = 1.22 \text{ m}^2$$

$$800 = R_T Q^2 + R_A\left(\frac{Q}{2}\right)^2 = 0.245\, Q^2 + \frac{2.22}{4} Q^2$$

$$\text{or} \quad Q^2 = \frac{800}{0.8} = 1000 \quad \text{or} \quad Q = \sqrt{1000} = 31.6 \text{m}^3/\text{s}$$

Example (S.I.) :

A mine fan produces a pressure of 500 Pa and passes 25m³/s of air in the mine. The mine has trunk airways and two splits A and B. Air flow in split A is 15m³/s and in split B, 10 m³/s. With a view to increase the air flow in split B to 15 m³/s a booster is to be installed in it. Calculate the size of the booster if the resistance of the shafts and trunk airways is 0.2 resistance units.

Answer :

Let R_A = Resistance of Split A,

R_B = resistance of split B,

R_T = resistance of shafts and trunk airways,

P_B = pressure generated by the booster fan,

Q_A = quantity in split A after booster installation

Q_B = quantity in split B after booster installation.

and Q_T = total quantity flowing in the mine after booster fitting.

Before installation of the booster.

$$500 = R_T \times 25^2 + R_A \times 15^2 = R_T \times 25^2 + R_B \times 10^2$$

or $R_A = 1.67$ resistance units

and $R_B = 3.75$ resistance units, since $R_T = 0.2$ units.

After installation of the booster,

$$500 = R_T Q_T^2 + R_A (Q_T - Q_B)^2$$
$$= 0.2\, Q_T^2 + 1.67\, (Q_T - 15)^2 \text{ since } Q_B = 15\text{m}^3/\text{s},$$

By solving the above equation, we get

$Q_T = 29\ m^3/s$, neglecting the negative value of Q_T.

In relation to flow in split B,

$$500 + P_B = R_B Q_B^2 + R_T Q_T^2$$
$$= 3.75 \times 225 + 0.2 \times 29^2$$

which gives $P_B = 512$ Pa.

The points to note from the above examples are :

1. An increase in the air-flow in the boosted district increases the total quantity passing in the mine.
2. The increased quantity results in a greater pressure loss in the shafts and trunk airways and thus reduces the pressure available for air-flow across the splits.
3. Horse power remaining the same, the main mine fan generates less pressure when circulating a larger quantity of air in the mine.
4. This causes a reduction in the quantity of air in the unboosted split.

Too large a booster in one split can cause stoppage of air current or even reversal of air current in the other splits, particularly, if the trunk resistance is high compared to the resistance of the splits. If there are two splits in parallel and the pressure of one is boosted by a booster, in theory, the booster has a **critical pressure** at which the pressure difference at the point of splitting drops to zero. All the air then passes through the booster while the air passing in the unboosted split-in parallel is nil. The selection of the size of a booster and its location need, therefore, careful planning.

Example (SI) :

Find the critical pressure for a booster fan to be installed as per the following data.

The main surface fan at a mine developes a pressure of 1.2. KPa Of this 0.8 KPa is consumed in the shafts and trunk airways and 0.4 kPa is used up in ventilating two splits A and B. Air flow in split A is 15 m³/s and in B, 10m³/s. What should be the critical pressure of the booster to be installed in split B ?

Answer :

Installation of booster in split B will reduce the flow of air in split A to zero.

Now the quantity passing through the trunk airways before installation of booster = 10 + 15 = 25m³/s.

Resistance of trunk airways and shafts = $800/25^2 = 1.28$ units.

After installation of booster in B, the whole of the main fan pressure is consumed in overcoming the resistance of shafts and trunk airways. If Q is the quantity flowing through trunk airways as well as through split B after installation of booster, and assuming main fan pressure to remain unchanged.

$$1200 = 1.28\ Q^2 \text{ or } Q = 30.62\ m^3/s.$$

Resistance of split B = $400/10^2 = 4\ Ns^2\ m^{-8}$

Therefore critical pressure of the booster = $4\ (30.62)^2$

= 3750 Pa = 3.75 kPa.

The general considerations in installing a booster should be :

1. Is the installation of a booster in a particular split justified ?
2. What should be its site and what effect it will have on the general system of ventilation ?
3. Will it cause recirculation of the foul return air ?
4. What w.g. should it develop ?

If the booster is placed too for inbye of the neutral point in either intake or return, a zone for recirculation will inevitably occur if the booster is placed too far outbye of the neutral line the pressure difference between intake and return will be increased and excessive leakage will occur in the normal direction. Though this will not be dangerous, it will be inefficient.

For installation of booster seven days notice has to be given to the J.D.M.S.

Compressed air jets :

Where compressed air is available underground, as in most of the metal mines, quantity of air at the working face can be increased by admission of only a small compressed air jet in an air tube. The small quantity purposely introduced in the air tube increases the air flow by about 20% at the face. A venturi-tube blower having a convergent-divergent tube is more efficient in this respect and increase the flow by nearly 30%.

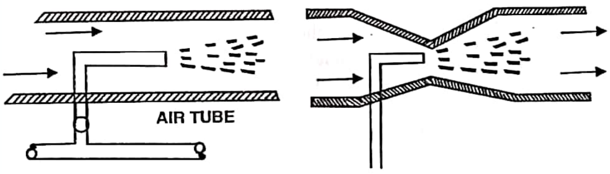


Fig. 3.13 Compressed air jets

Removal of gas accumulation in a large cavity :

An overman or Assistant Manager has sometimes to deal with such cases. A place is considered to be dangerous due to gas if it is 1.25% or more, or measurable by the flame safety lamp. Suppose the cavity is 15m long and 4m high above rail level on an intake and haulage road (Fig. 3.14).

The steps to be taken are as follows :

1. Withdraw all workers from the inbye side.
2. Cut off electric supply to the inbye workings where the intake air is passing from the cavity.
3. Install ventilation tubing 0.6m dia. made of galvanised iron sheets extending from the cavity to nearest stopping by puncturing a hole in the stopping.
4. The firedamp is sucked gradually to the return through the air tubes and passes straight away to the U.C. shaft

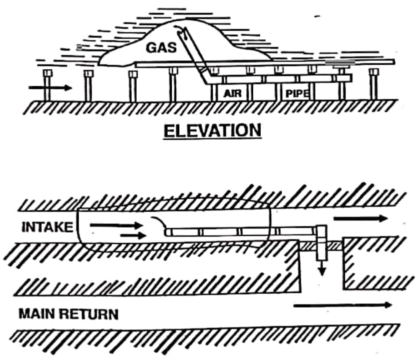


Fig. 3.14 Removal of gas accumulation in a large cavity

5. Inform the manager about the gas accumulation and its clearance.

To prevent firedamp accumulation in such a place again it is better to support the high cavity and pack it with boiler ash or sand. If this is not possible, leave the air tube permanently in position.

If the cavity were on the return side, it would suffice to erect a hurdle screen on the road so that air has to pass to the cavity over the hurdle screen and clear up the gas.

VENTILATION SURVEY :

A ventilation survey is carried out to investigate the ventilation system of a mine and to find out the adequacy of the ventilation system, the points of leakage and the extent of leakage, and the steps necessary for further improvemet. The investigation is generally done in the following three ways.

1. *Quantity surveying :* This involves the measurement of the air velocity and the quantity of air flowing in various parts of the mine.
2. *Pressure surveying :* This involves the measurement of the air velocity and the quantity of air flowing in various parts of the mine.
3. *Qualitative surveying :* This involves the determination of the freedamp content at different strategic points in the mine and chemical analysis of the air samples, if necessary. If the firedamp content in the general body of return air of ventilating district exceeds 0.75%, the ventilation is inadequate.

Quantity Surveying :

Such surveying is necessary (i) to ascertain the distribution of air in the main roads, ventilation districts and at the working faces, (ii) to locate leakages between intakes and returns, (iii) to know the efficiency of fan during fan tests.

The quantity of air going down the mine and at other points has to be measured periodically as per requirements of the Mining Regulations. (Once in 14 or 30 days).

Quantity of air passing per minute at a place is given by the formula

$$Q = A \times V$$

where A = cross-sectional area of the roadway in m^2

and V = average velocity of air in m/min, at the place.

The instruments that are used for measurement of air velocity are the anemometer, the velometer, the pitot tube (already described in earlier chapter) (For profilometer sea at the end of chapter).

In a mine the quantity of air measured at the entrance or at the working places is always less than the quantity at the outlet of exhaust fan in upcast shaft because of (i) warming of the air during passage along the roadways, (ii) admixture with gases given off from the strata, and (iii) expansion of the air due to reduction of the barometric pressure. The increase in the quantity of air due to expansion in the upcast shaft can be roughly taken as 1% for every 100m depth and provision should be made for this when estimating the quantity of air required to be circulated by the fan.

Anemometer :

It is an instrument to determine the distance travelled by air in a given time and is used where the air velocities are between 60m and 1000m/min. One type of anemometer (Fig. 3.15) consists of a small fan having its vanes at 40° to 50° to the direction of air flow. The travelling air rotates the vanes and through gearing arrangement the pointers on the dials of the anemometer record the distance travelled. There are usually one large and 3-4 small inner dials and a little practice is required in taking readings of the instrument. The gears of the anemometer can be engaged or disengaged by a clutch.

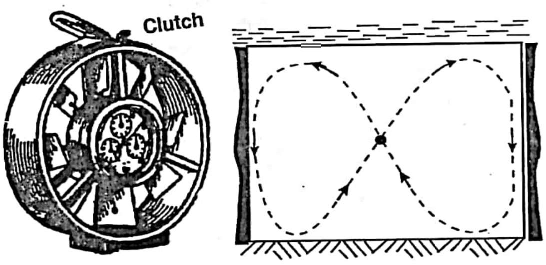


Fig. 3.15 Left-Anemometer. *Right Dotted line indicates path of anemometer movement*

Anemometer Range : 0 – 1.00.000m

Possible measuring wind speed 1 – 15 m/s

When using the instrument in an underground roadway it should be held away from the body of the observer and the plane of rotation of the vanes should be as near as normal to the direction of air flow. To determine the velocity of air a stop-watch is essential in conjunction with an anemometer. The instrument can be held at the end of a stick to avoid obstruction by the body of the observer.

To determine the average velocity of air at any point of a roadway note the reading of the pointers of the instrument and with the instrument in declutched position hold it in the roadway away from the body. Keep stop-watch ready at hand. At the desired moment set the stop-watch in motion and stimultaneously engage at the gears of the anemometer by the clutch arrangement. The instrument now records the distance travelled by the air.

Move the instrument throughout the cross-section of the roadway as shown by the path in the Fig. 3.15. After 2, 3 or 4 minutes declutch the instrument and simultaneously stop the stop-watch. Take the reading. The difference indicates the distance travelled by air in the time recorded by the stop-watch. The average velocity of air is then calculated.

In some models of anemometer the pointers on all the dials can be brought to zero by operation of a lever before using the instrument for purposes of calculating air travel.

The places where average velocity of air has to be measured should be selected on the following considerations.

1. The roadway should have nearly uniform cross-section for nearly 15m on either side, and it should be straight.
2. The cross-section should be such that its area can be easily calculated.
3. It should be away from bends, junctions and places having sudden changes in cross-section, and free from obstructions which may cause turbulent air flow.

Electronic Anemometer :

Nanda Manufacturing Company is marketing an electronic anemometer which possesses the following features :

1. Solid state circuitry with integrated circuits.
2. Sensing head : non contact type fully encapsulated electronic transducer using principle of change in capacitance.
3. Indicator unit : measuring accuracy : ± 2%
4. Measurement of air velocity in 3 ranges.

   (i) 0.25 – 25 m/s (ii) 0.25 – 5 m/s (iii) 0.25 – 10 m/s.

   The battery used is Eveready type 276-P two nos.

Velometer :

The velometer is an instrument which directly indicates the air velocity in m/s at any point of observation. In an underground mine to find the average velocity the spot readings should be taken at a number of equally spaced points over the cross-section of the roadway and the average value should then be calculated.

Nanda manufacturing Company is marketing a velometer which directly reads the flow of air or gases. The use of micro-circuits makes the instrument not only compact and trouble free but also permits measurement of air flow to an accuracy of ± 2%. Besides the advanced electronics, the design incorporates many outstanding features like the specially designed vanes pivoted on jewels which make it extremely sensitive to air flow.

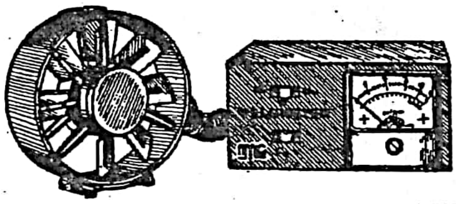


Fig. 3.16 Velometer, Directly reads air Velocity. Range 0 – 15 m/s (N.M.C.)

The instrument consists of a sensing head and an indicator unit. The head is either mounted on the indicator unit or a portable head with extension cable and a polarised connector is available.

The head incorportaes a rotating vane type non-optical, non magnetic and non-contact transducer which is unaffected by virbations, dust, temperature or humidity. This transducer generates electrical signals having a frequency directly porportional to the rate of air flow through the head. These signals are then converted to a direct current, which drives an indicating meter calibrated in terms of flow rate per minute or meters per second.

BA 4 air velocity monitor :

It consists of the following parts :

Detector Head : The detector head measures air velocity between 0 and 10 m/s using a vortex shedding sensor which has no moving parts. There are three ranges 0.2 m/s, 0.5 m/s and 0.10m/s, selectable by a switch in the Control Unit. Each range is individually calibrated and therefore for best accuracy the instrument should be used on the most sensitive range appropriate to the air velocity being measured. Connection to the Control Unit is by way of a 6-way socket and a cable which is available in various standard lengths.

Control Unit : This unit houses an indicating meter, an alarm lamp and switches to control the BA4 system. The meter is scaled for the three ranges mentioned above. The controls are :

1. A power switch is OFF, battery Check, and ON>
2. A range switch which selects one of the three ranges.

3. A time constant switch which selects one of three time constants, 15, 30 or 60 secs. to 63% of total step change.
4. An alarm set point adjutment for 0-100% of the range selected, with an alarm operating on a falling level.

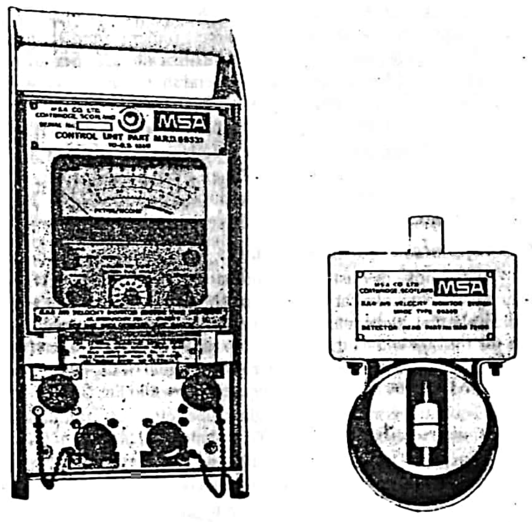


Fig. 3.17 BA-4 air velocity monitor. (Right) Detector head.

In normal operation an integral lamp flashes once every 16 seconds and in alarm condition, once every second, Some fault conditions cause the lamp to cease flashing.

In addition to sockets for connection of the Detector Head and Power Supply Unit, there is a recorder socket which provides a 0.4 – 2.0 volt D.C. signal corresponding to zero to full scale in each range for the recorder unit and an alarm socket which provides a volt free relay contact which closes and opens as the alarm lamp flashes on and off. In addition the alarm socket provides an analogue output of 0.4–2.0 volt D.C. signal corresponding to zero to full scale on each range.

Recorder Unit : This unit contains a Rustrak Recorder, a time marker and clock battery. The recorder normally indicates the air velocity being measured but every hour the signal is disconnected and the Recorder indicates below the offset zero. The chart speed is approximately 12.7 mm/hr and reference should be made to the time event marks for the exact time place.

Battery Packs : Two battery packs are available.

1. A 7Ah which may be float-charged underground by means of a flameproof power supply with intrinsically safe output to enable instrument operation during week-end shutdown periods when mains power supply is disconnected. When surface charged, this battery will supply sufficient power for approximately five days.
2. A 20Ah battery surface-charged only, to supply power for approximately twelve days. As an alternative to battery packs a flameproof mains supply with intrinsically safe output can be provided to power the BA4 directly.

Smoke generator for low velocity:

The smoke or dust can sometimes be used to measure air velocity. In a metal mine, smoke can be easily produced by burning a fuse. Two men standing about 100 m apart with stopwatches in hand, measure the time taken for the smoke or other visible vapour to travel from position of one person to the position of another person. In a coal mine, the visible smoke cloud is produced by a simple arrangement shown in Fig. 3.18.

Smoke generator consists of glass tube containing granulated pumice stone saturated with anhydrous tin or titanium tetrachloride.

Glass tube is fitted with a rubber aspirator bulb by rubber tubes at one end and the other end is in the form of a tip covered by a rubber cap.

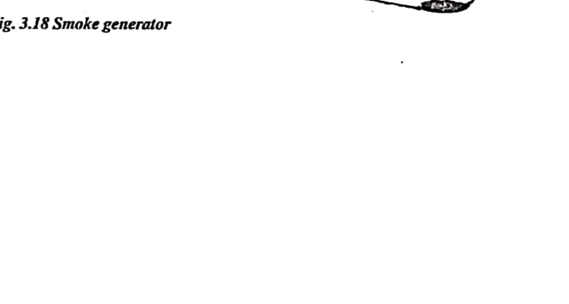


Fig. 3.18 Smoke generator

Smoke generator :

When the smoke generator is to be used, the tip of glass tube is broken off. Now bulb is squeezed. Air charged with tetrachloride vapour issues from the tip immediately forming a while vapour. The cloud consists of fine particles which are suspended in the atmospher longer than ordinary dust and are readily carried along by the air current. When the apparatus is not in use the tip is convered by the rubber cap.

Precaution :

It is advisable that before using the smoke generator upper opening and lower opening of the glass tube is cleaned by any pin or hard wire. This is essential because, when the apparatus is not in use, the chemical vapours choke the tube stem.

Applications :

(a) For determining low velocity below the range of an ordinary anemometer (below 1 metre per second).

(b) For testing air tightness of seals or stoppings.

Pressure Surveying :

The basic principle behind the pressure survey in a mine is Bernoulli's theorem which states ; when a fluid flows through a passage of varying cross-section, the total energy of the moving steam remains constant, assuming no friction losses.

The total energy is the sum total of the kinetic energy and pressure energy and so it follows that a reduction in the velocity energy is accompained by a corresponding increase in the pressure energy.

A pressure survey can be carried out in two ways :

1. By ascertaining the total pressure of the air at each point with aneroid barometers and then calculating the pressure difference.
2. By using a very sensitive inclined manometer.

Fig. 3.19 Portable manometer

The first method involves the use of two sensitive, recently calibrated aneroid barometers which should be taken underground at least 24 hours before the start of the pressure survey, so that they may adjust themselves to the underground pressure conditions. This method is not much adopted in our mines.

The other method uses sensitive inclined manometer (containing alcohol), is simple and is generally adopted. The other accessories required for carrying out the pressure survey with the help of inclined manometer are : two flexible rubber hose pipes, each 100m long; spirit level; measuring tape; ventilation plan; a tripod. (Fig. 3.20).

The stations at which pressure drops have to be measured are marked on the ventilation plan after a reconnaissance of the mine. The considerations that apply to selection of stations for measurement of air quantity with the help of anemometer apply in this respect also. The day on which pressure survey (and in fact, any ventilation survey) is to be conducted should be a rest day in which ventilation appliances in the mine remain undisturbed. The fan speed also should be constant during the period of the ventilation survey. It is essential that ventilation conditions should be maintained as steady as possible during the pressure survey (and also during the quantity survey).

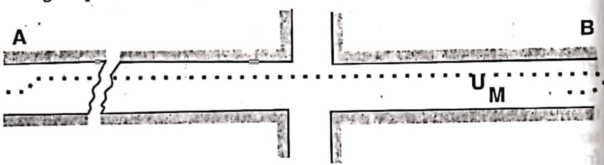


Fig. 3.20 Pressure survey with inclined manometer. A and B are stations; dotted line indicates, rubber tubing; M-inclined manometer.

To note the total pressure drop between the two adjacent stations spread flexible hoses, connect them to inclined manometer placed midway between adjacent stations and keep the hose ends facing the air current. See that (i) manometer base which rests on the tripod is level, and (ii) there is no leakage at the joints of hose pipes with the limbs of the manometer.

Note the readings of the manometer and record them in a meausrement book.

Generally pressure survey and quantity survey are carried out by the same team (usually of 3 men) at the same time; therefore the readings of the ventilation survey are tabulated as follows :

Station Length, m	Air Velocity m/sec.	Quantity of air m^3/sec	Gauge reading mm Gauge reading, mm of w.g.	Velocity Correction	Corrected Pressure drop. mm of w.g.

A complete and systematic ventilation survey of a mine may be carried out in late summer and in late winter if there is no rapid change in the mine characteristic. In shallow mines working with natural ventilation only, a quarterly survey is however, desirable. In very deep mines only the annual survey should suffice. Before a booster fan is to be installed underground it is essential to carry out a complete ventilation survey to study the possible effects of the booster installation. Quantity measurement are to be made, under the Indian Coal Mines Regulations, once in 14 days in gassy coal mines (deg. 2 and deg. 3 mines) and once in 30 days in deg. 1 gassy mines.

The points where the measurement have to be made for a pressure survey and a quantity survey are marked underground as permanent stations.

Ventilation of deep mines and airconditioning :

Ventilation of deep mines over 600 m pose the following problems :

1. The strata temperature gradually increases, raising the temperature of the mine air.
2. Deep mines are dry, so the air current carries much underground dust.
3. In coal mines, the deeper mines are known to be more gassy than the shallow ones; the quantity of gas liberated therefore increases and needs large quantity of fan air.
4. The resistance to air current is high due to the large depth of the shafts and powerful fans developing large w.g. are required.
5. Capital investment on shaft sinking, shaft winders, etc. is high. Interest on such heavy capital investment and depreciation forms a sizeable amount of cost per tonne and to keep the cost of production low, deep mines have to be planned for large outputs which require circulation of large air quantities. The main roadways have therefore to be of ample cross-section for the large volumes of air.

The circulation of large quantities of air at a high w.g. requires powerful mechanical ventilators. Moreover high velocity of air current is uncomfortable to the underground workers and to keep the ventilation adequate with reasonable velocity of air current, the following measures are sometimes adopted :

(a) Methane drainage in the case of coal mines,

(b) Binding of dust instead of wetting it as wetting increases the humidity of air,

(c) Sending conditioned air underground by reducing its temperature to nearly O°C and eliminating its water content as far as practicable,

(d) Use of descensional ventilation.

(e) Arranging winding of coal/mineral in the return airway and in the upcast shaft.

(f) Use of comp. air operated equipment instead of electrically operated machines. The compressed air is cooled and its moisture content considerably reduced by proper devices before sending it underground. The Compressed air mains are located in the D.C. shaft and intake airways and the water condensed in these compressed air mains is drained out automatically by suitables devices fitted on the mains at intervals.

(g) By covering the undergound drains to minimise their water evaporation and entry into circulating air current.

(h) Use of protable air coolers or semiportable spot coolers in hot stopes or development districts of metal mines.

(j) Installation of booster fan underground in air splits with high resistance.

In deep mines like the gold mines at Rand in South Africa, the gold mines at Kolar Gold Field and some coal mines of Germany (over 1200m deep) conditioned air at nearly 3°C is sent down the mine. The air conditioning plant consists essentially of the following units.

1. The refrigerant circulation system,
2. The brine circulation systm,
3. Coolant circulation system

In the refrigerant circulation system, the refrigerant (usually ammonia gas) is compressed by a compressor and the gas, at a high temperature and pressure resulting from compression, passes through an oil separator which extracts any oil picked up from the compressor cylinder. The oil-free gas passes outside the tubes of a condenser where it is cooled to nearly atmospheric temperature by circulating water. This results in liquefaction of most of the refrigerant gas which then passes thorough a float regulator and when passing through its needle valve, the gaseous portion suddenly expands resulting in drastic lowering of its temperature to nearly - 12°C. The liquid portion of the refrigerant being in contact with the expanded gaseous refrigerant, also cools down to nearly the same temperature. The refrigerant, most of it in the liquid state and part of it in the gaseous state at nearly –12°C goes to evaporator (also called vaporiser).

A brine pump circulates brine (a weak solution of calcium chloride at a sp. gr. 1.22) through the network of tubes placed in the evaporator and extending in the path of the atmospheric air forced down the D.C. shaft by a forcing fan. The chilled refrigerant passing through the evaporator absorbs the heat of the brine and the cold brine cools the atmospheric air on its way to the D.C. shaft. The liquid refrigerant is converted into gaseous state by the heat extracted from the brine and is sucked into the compressor cylinder for further recirculation.

The condenser consists of a network of tubes outside which the gaseous refrigerant (under a high temperature and pressure as a result of compression) passes and the internal surface of the tubes is in contact with the circulating water (called coolant) at nearly atmospheric temperature. The water absorbs heat from the refrigerant and the resultant hot water is pumped to form sprays which dissipates the heat to the atmosphere. The cooling pond collects the water of the sprays for circulation.

A good refrigerant should have a large latent heat of evaporation and should evaporate and condense as near the atmospheric temperature and pressure as possible. The refrigerant is usually ammonia in the industrial type refrigerant plants used on the surface. Its critical temperature is 131°C and critical pressure 113 atmospheres. The gas is cheap and has the largest heat of evaporation compared to other refrigerants but is toxic and corrosive for brass and copper. However, leakage of the gas in a refrigeration system can be easily detected because of its smell and corrosion can be avoided by selection of suitable alloys for construction of the refrigeration system.

Other refrigerants include freon, carbon dioxide, methyl chloride, etc. For underground air cooling plant the gas Freon-12 is used as a refrigerant (dichloro-difluoro-methane; chemical formula CF_2 Cl_2). It is a harmless, colourless gas and does not affect C.I., steel and other metals. At atmospheric pressure its boiling point is 29.4°C. Its disadvantages are : (1) It is about 7 times as costly as other refrigerants like ammonia. (2) As it is odourless, its leakage cannot be detected, unlike ammonia and the joints in the plant should be tight.

Although carbon di-oxide can reduce compressor size due to its low specific volume, it is not suitable for use at mines because of its low critical temperature (32°C) and toxicity.

In confined places of underground workings ammonia should not be used as it is toxic and forms and explosive mixture with air, if present 30% by volume.

Brine is preferred to water when the air has to be cooled to a fairly low temperature (nearabout the freezing point of water) because brine has a lower freezing point and hence is less likely to freeze in pipe lines.

Calcium chloride and sodium chloride solutions are the commonly used brines as they remain liquid under all temperatures commonly encountered. They have a relatively high specific heat and are fairly stable.

The capcity of an airconditioning plant is expressed in tonnes. A 1-tonne plant removes 3024 kcal/hr heat from the air passing through it.

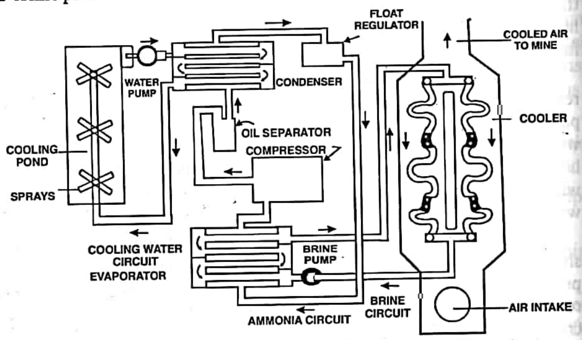


Fig. 3.21 Schematic diagram of the air conditioning plant at the surface of Giffords shaft at Kolar Gold Field (800 te cap).

Fig. 3.21 shows the airconditioning plant which was installed on the surface at Gifford shaft of Champion Reef Mine, K.G.F. The following description relates to it when it was in use. It is a 800–te capacity plant and cools the intake air from 21°C (wet bulb temp.) to nearly 3°C saturated. The plant consists of a reciprocating compressor, an oil separator, a condenser (shell and tubes type), an evaporator (shell and tubes type), cooling pond with water sprays, water pump, brine pump, a cooler. The cooler consists of a large coil of 38mm bore steel pipes.

Ammonia gas is compressed by a reciprocating cmpresor driven by a 300 kW, 300 r.p.m. motor. The compressed gas, after passing through an oil separator, passes on to a condenser where it is cooled to liquification by circulating cold water by three 22kW pumps, each of 3 m³/ min. capacity. The cooling water extracts the heat of compression of the gas and also the latent heat of evaporation of ammonia. The liquified ammonia passes to the evaporator through the float regulator which so adjusts the flow that a constant level of liquid ammonia is maintained in the evaporator. The compressor compresses the gas during the onward stroke and during the return stroke a partial vacuum is created in the cylinder, as a result of which a part of the liquid ammonia in the evaporator is vaporised an drawn into the cylinder and is then compressed. After the extracttion of heat from ammonia the cooling water gets warm and is cooled again in a spray pond.

In the evaporator a weak calcium chloride solution (sp. gr. 1.22) is circulated by 3 brine circulating pumps, each of 4.9 m³/min capacity. The brine on leaving the evaporator, gets cooled and the cold brine is then circulated through the cooler. A 2-stage axial-flow fan, 2400 mm dia., pushes the air to be cooled over the cooler pipes. The brine, after cooling the air, gets warm and has to be cooled again by the vaporisation of liquid ammonia in the evaporator.

Surface airconditioning plants are popular due to :

1. Posisibility of using a cheap refrigerant like ammonia gas resulting in cheapness of the plant.
2. Ease of disposal of waste heat.
3. Ease of getting water at surface temperature and facility of its circulation in unconfined space.
4. Convenience of operation, inspection and maintenance.

Surface plant produces large N.V.P. because of large temperature difference of U.C. and D.C. air columns.

Underground air conditioning plants are used for circulating only a part of the air going to deeper levels. CO_2 or freon gas is generally used.

The limiting depth of a mine is determined by the air temperature and humidity, apart from considerations of strata pressure and roof control, as well as strength of hoist ropes.

Spot Cooler :

In Mosabani copper mine and some other mines small capacity spot coolers are used underground to provide cool air to places which are uncomfortably hot. A spot cooler is a semiportable, small sized air conditioning plant with a fan installed underground with the limited purpose of supplying conditioned air to few working faces. Its cooling capacity varies from 37,800 to 126,000 kcal/h. There are two types of spot coolers : (a) where the direct evaporation of the refrigerant cools the air, and (b) where an intermediate coolant such as water is used. The compressors are usually of the reciprocating type and the cooling of the compressed refrigerant is done in air-cooled condensers. Through the spot coolers supply cool air at the working faces, they have the disadvantage of leaving the rest of the mine hot.

Profilometer :

In a mine the cross-section of a roadway is rarely of a regulator shape. For Measuring the area of a roadway of irregular shape, a suitable device is a profilometer.

Construction :

The profilometer comprises a tripod stand with adjustable legs and a quick clamping swivel head permitting rotation in both horizontal and vertical planes. A hook is provided at the bottom of the swivel head for suspending a plumb line for centring purposes. On the top of the swivel head is screwed a horizontal bar. On one end of the bar 290mm away from the centre of the swivel head is fixed a circular scale graduated in degrees with its plane at right angles to the axis of the horizontal bar. On a horizontal axle central to the circular scale is mounted a hub on bush bearing. The hub can be rotated and clamped in any position by a clamping knob.

The hub carries a light radial arm of tubular aluminium, 9.3mm in diameter and 1.06 m in length. Into this, slides a solid aluminium extension arm graduated in centimeters. A clamping screw can clamp the extension arm in any desired position. A pointer attached to the hub exactly opposite to the radial arm reads the angular position of the arm on the circular scale.

The horizontal bar is suitably counterweighted against the weight of the radial arm. It carries on top two vertical pins at the two ends for the purpose of alignment.

Operation : The tripod is set in the roadway section to be measured, and its position and height so adjusted that the centre of the circular scale lies roughly at the centre of the roadway cross-section and its plane in the plane of the cross-section. The horizontal bar is rotated so as to lie along the axis of the roadway with the help of the alignment pins. It is then levelled with the help of a spirit level mounted on it.

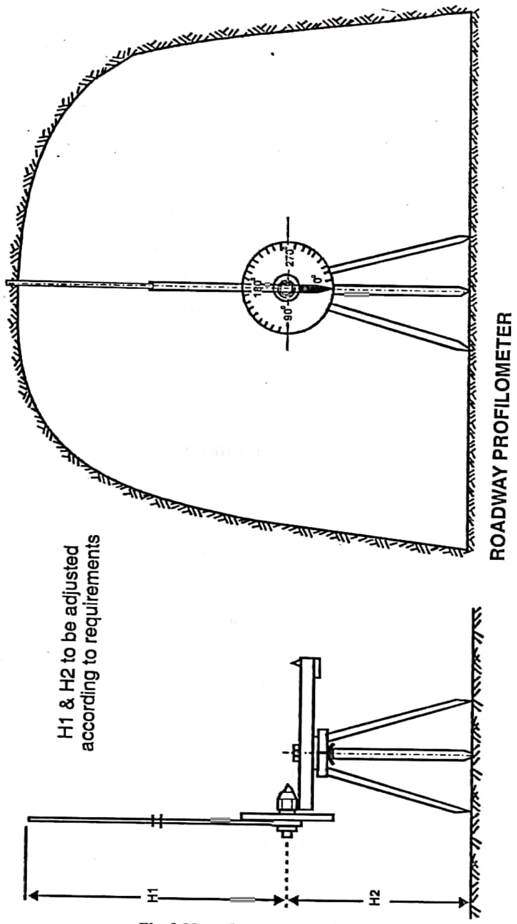


Fig. 3.22 roadway profilameter

The radial measuring arm is then rotated and clamped at the zero of the circular scale. The sliding arm clamping screw is loosened and the arm extended till it touches the roadway perimeter. The reading on the sliding arm is taken. This reading added to the length of the fixed arm gives the radial distance from the centre of the circular scale to the roadway perimeter at zero degrees.

The sliding arm is then loosened and telescoped into the fixed arm to the extent necessary. Simultaneously the hub carrying the fixed arm is loosened and the arm rotated unit the tip of the sliding arm comes to a point of significant change in the roadway profile. The sliding arms as well as the hub are now clamped tight and the linear readings on the sliding arm and the angular reading on the circular scale noted. The process is repeated until the entire perimeter of the road-way is covered.

Determination of roadway perimeter and cross-sectional area :

The radial distances are then plotted to a suitable scale at the corresponding angles. The locus of these points gives the profile (perimeter) of the roadway. While the perimeter of the roadway is obtained by adding the distances between these points, the area is measured by a planimeter or by counting the number of squares contained in the profile if the plot is made on a suitable square gird graph paper or by measuring the ordinates at close uniform interval defining the roadway profile. The area is obtained from the Simpson's rule :

$$\text{Area} = \frac{L}{3}\ (\text{Sum of first and last ordinates}) \\ +\ 4\ \text{times the sum of even ordinates} \\ +\ 2\ \text{times the sum of odd ordinates}$$

where, L = distance between ordinates.

QUESTIONS

1. What are the various devices used for coursing the mine air ? Give the formula for calculating the area of a regulator.
2. What is the difference between an auxiliary fan and a booster fan ? Show by sketches the installation of an auxiliary fan of (a) forcing type (b) exhaust type.
3. In relation to a booster fan what is (a) neutral line, (b) critical pressure ?

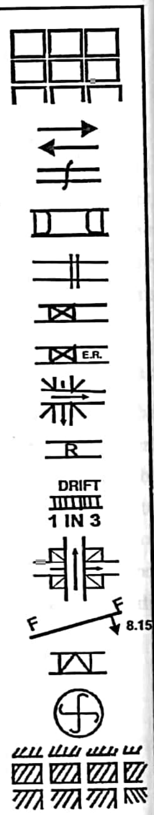
   What points should be borne in mind when installing a booster ?
4. How will you conduct a quantity survey in an underground mine ?

   Describe the procedure.
5. What problems arise in the working of a deep mine ? What are the different refrigerants used in an air-conditioning plant ? State their advantages and disadvantages.

O O O

Conventions for signs on ventilation plans

Name	Colour
Pillars and galleries	in black
Direction of air current	Intake in blue Return in Red
Branttice	In red
Doors	In red
Brick or stone ventilation stopping	In red
Fire dam, seal or stopping	In red
Explosion proof stopping	In red
Air crossing	In black
Regular	In red
Drift	In burnt sienna; Gradient in black
Explosion proof air crossing	In black
Fault	Red
Water dam	In red
Auxilary fan	In red
goaf	



CHAPTER **4**

MINE FIRES AND SPONTANEOUS HEATING

To start a fire the following conditions are essential :

1. Presence of a combustible material.
2. Presence of a source of ignition of sufficient intensity of heat.
3. Presence of oxygen.
4. Contact of combustible material and source of ignition for some time.

For the fire to continue after it starts a sufficient supply of oxygen or air must be available. In the absence of oxygen the fire gradually dies down.

Classification of Fires :

Indian Standards Specifications Classifies fires as follows :

1. *Class 'A' fires :* These fires involve combustible materials *e.g.* timber, coal, rubber, conveyor belt, other carbonaceous material.
2. *Class 'B' fires :* These fires involve inflammable liquids *e.g.* lubrication oils, diesel, petrol and other fuel oils, greases, *etc.*
3. *Class 'C' fires :* These fires involve gaseous fuels like LPG gas, butane, *etc.*
4. *Class 'D' fires :* These are metal fires such as melting iron, *etc.*
5. *Class 'E' fires :* These fires involve live electrical equipments such as electric motors, generators, cables, oil-filled transformers, circuit breakers, electronic equipments *etc.*

Class 'A' fires are generally quenched by water. Stone dust and sand may be used if the fire is on a small scale and in its early stage. Overhead fires cannot however be tackled by stone dust and sand.

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Air crossing	In black	
Regular	In red	R
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Class 'A' fires are generally quenched by water. Stone dust and sand may be used if the fire is on a small scale and in its early stage. Overhead fires cannot however be tackled by stone dust and sand.

Class 'B' fires are quenched by an extinguishing agent which has a blanketing or smothering action and foam extinguishers are best suited for this purpose. Water, if used for extinguishing, spreads the oil and the fire along with it.

Class E fires require use of extinguishing agent which is not a conductor of electricity. Water, with its normal impurities is a conductor of electricity and should not be used. A foam extinguisher is also not recommeded but sand is suitable for smothering such fires in the initial stages on a small scale and the best extinguisher is the carbon gas. One of the advantages of CO_2 extinguisher is that it has no adverse effect on the insulation or other working parts of the equipment.

Portable fire extinguishers :

The common arrangement for dealing with a fire is provision of boxes filled with sand and a few buckets full of water. Under the Mining Regulations these arrangements of portable fire extinguishers have to be provided.

(a) at every entrance to a mine,
(b) at every landing and shaft bottom in use,
(c) at every engine room,
(d) at every other place where timber, brattice cloth, grease, oil or other inflammable materials are stored,
(e) at suitable places at the entrance to every district of a mine.

Portable fire extinguishers are of the following types :

1. Soda acid 2. Water CO_2 3. Foam 4. CO_2 gas
5. Dry powder 6. C. T. C. 7. B. C. F.

Soda acid extinguisher :

It consists of a lead coated steel cylinder full of solution of soda bicarbonate and a hermetically sealed bottle of sulphuric acid placed inside a cage in the extinguisher. A spring loaded plunger protruding outside the cylinder is so placed that when the plunger is struck against a hard surface it breaks the glass bottle of acid and the sulphuric acid reacts with sodium bicarbonate solution generating CO_2 gas. The gas under pressure forces the mixture of water and sodium bicarbonate up a tube and a nozzle to a distance of 6–8 m

for initial 60 seconds in a 9 litre extinguisher and the total duration of discharge is 60 – 120 sec. The CO_2 expels the soda bicarbonate solution partially dissolved in the water and helps in the quenching of fire when it is liberated as the water evaporates. Cooling action of water is the main factor in quenching the fire; CO_2 helps it only partially. The jet of the solution should be aimed at the base of the flames.

Such fire extinguisher is suitable for only class 'A' fires and is not recommended for class 'B' fires. It is prohibited by the mining regulations for quenching class 'E' fires.

The extinguisher can be operated either in an inverted or upright position depending upon the model and instructions for operation are stated on the extinguisher itself. In general soda acid extinguishers of conical shape should be held in upright position and of cylindrical shape in inverted position during use. The conical shaped extinguishers are gradually going out of use. The extinguisher can be refilled by the user at the site if refills are kept in stock.

An extinguisher of 9-lit. capacity weighs 15 kg when full the soda acid extinguisher is also available as wheel mounted mobile model. One model of 45 lit. Capacity has 10-12 m range of discharge for initial 100 sec. and total duration of distcharge is 100-150 sec. During operation the mobile model has to be kept upright. The hose provided on it directs the jet in any direction.

Water CO_2 extinguisher :

It consists of a steel cylinder coated inside with copper and containing water and a charge of high pressure CO_2 in a separate small sealed container of copper placed within the main cylinder. A spring loaded plunger protruding outside the main cylinder is provided a the top of this small CO_2 container. To operate, strike the plunger against a hard surface. This punctures the CO_2 container and the gas pressure forces the water out of the cylinder to nearly 12 m.

It is effective for class 'A' fires only but not recommended for class 'B' fires and totally prohibited for class 'C' fires. The small copper cylinder containing CO_2 under pressure forms the refill and during use, specially in underground mines, the extinguisher can be recharged with water and the refill.

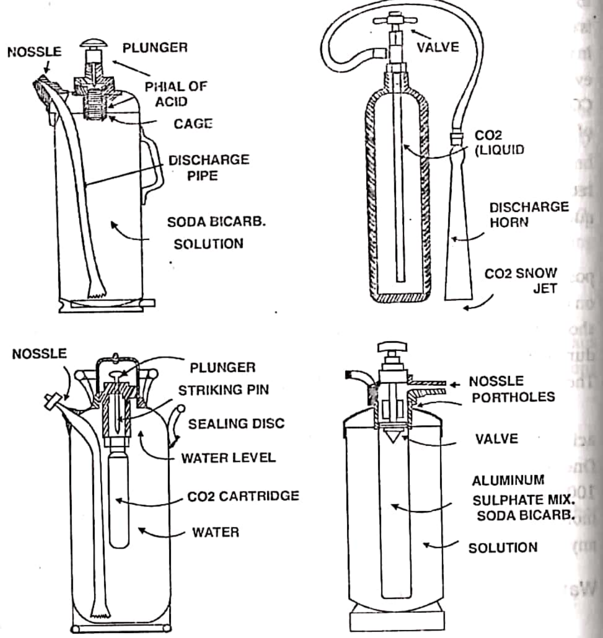


Fig. 4.1 Top Left -- Soda acid extinguisher
Top Right -- Carbon dioxide extinguisher
Bottom Left -- Carbon dioxide water extinguisher
Bottom Right -- Foam extinguisher

Foam Extinguisher :

In this extinguisher the foam produced during its operation forms a blanket on the flame and smothers the fire by stopping access of air. A foam type extinguisher is not permitted for dealing with class 'C' fires as the foam is a conductor of electricity but it is well suited for class 'A' and class 'B' fires.

There are two types of foam type fire extinguishers :

1. Chemical foam extinguishers and
2. Air foam extinguishers.

The former type is more common and it consists essentially of a vessel containing a solution of soda bicarbonate enclosing a sealed container of aluminium sulphate solution or similar foam forming solution along with a stabilising agent. During use the seal of the inside container (containing foam forming chemicals) can be released by turning a spring loaded plunger rod attached to it. When the seal is thus released and the extinguisher turned upside down chemicals of the two containers react to give CO_2 in the form of foam bubbles which are forced out of the nozzle under pressure. The foam has a smothering and wetting action with the flame. all foam type extinguishers have to be held upside down during use. The commonly used extinghisher of 9- litre capacity produces about 75 litres of dense foam. In "Firex" foam type extinguishers the foam refill basically consists of two packets of powder i.e. aluminium sulphate and sodium bicarbonate. Refilling can be done by the user at the site. The total duration of discharge of foam in a 9 - litre extinguisher is 50 - 90 seconds with an effective range of discharge 6 - 8 m for initial 40 seconds. The wheel mounted mobile extinguishers are of 45 litre or 135/150 litre capacity. The 45 - litre capacity extinguisher produces 360 litres of dense foam and has a duration of discharge as 75 - 120 seconds and effective range of discharge 11 - 14 m for initial 75 seconds. The 135/150 - litre extinguisher has 90-135 seconds duration of discharge and the effective range is 11-17 m for initial 100 seconds.

CO_2 extinguisher :

It consists of a cylinder having one to five kg of CO_2 in liquified form under high pressure (70 kg/cm²) which can be released during use by operation of a wheel valve mounted on its top as in the case of the familiar Indane gas cylinder used for domestic cooking. This is by far the simplest portable fire extinguisher and has the advantage that it can be used for fire of any class including electrical fires. No residue is left on the extinguished material. The gas penetrates to otherwise inaccessible places. CO_2 has the property of quenching the fire and the gas leaving the extinguisher is at a very low temperature so that it exerts a cooling effect on the burning material.

All CO_2 gas type extinguishers are provided with funnel shaped discharge horns and the CO_2 leaves the extinguisher in the form of a dense but very cold white cloud. Such extinguisher can nowever be used where air current is not brisk. In a well ventilated underground roadway the CO_2 may be carried away by air current rendering the extinguisher ineffective. The refilling of CO_2 can be done only at the manufacturer's factory. During operation the extinguisher has to be held upright. One should not enter the premises affected by the immediately after use of the CO_2 gas extinguisher.

Dry powder type extinguisher :

An extinguisher of this type can be used for dealing with fire of any type including electrical fire. It contains a fire quenching powder and a container of high pressure CO_2 both placed inside a cylinder. The dry powder is a mixture of chemicals consisting of mono ammonium phosphate as the main ingredient and other chemicals which have antisetting and water repellant properties. The powder has very little cooling effect on the fire but it forms an adhering glassy crust of metamorphic acid over the combustible material and retards further combustion. The fire quenching powder goes by the trade name 'Firex-Nugas' in the Firex make fire extinguishers. The powder is water repellant, non corrosive, non toxic, non conductor of electricity and will not deteriorate or cake when stored. It is free flowing at all temperatures. The extinguisher can be recharged on the spot by the user with suitable refills. In some models, e.g. Minimax make, the CO_2 gas cartridge is specially designed for withdrawl and inspection.

Dry chemical fire extinguishers are available in capacities of 1 to 12 kg of dry powder. The trolley mounted units are available in 25 and 70 kg capacity models.

To put the extinguisher in operation, strike the knob or plunger rod against a hard surface. This releases the high pressure CO_2 into the chamber containing dry chemical powder which is shot out of the cylinder under the gas pressure. The powder is discharged in a fan shaped cloud and it settles on the surface of the burning material or liquid and stops access of oxygen to it. The "squeeze grip" valve at the end of a hose pipe enables the operator to control the discharge of the powder and in some smaller models pressure may be released without further discharge of powder by inverting the extinguisher. For quenching the fire the extinguisher has to be held erect and not upside down.

C. T. C. Fire Extinguishers :

Carbon tetrachloride is a liquid at normal temperature but it evaporates easily. Its cooling effect is high and when the liquid comes in contact with burning material the cooling effect quenches the fire. C. T. C. is a non-conductor of electricity and therefore the C. T. C. type extinguisher can be used for electrical fires in generators, sub-stations, transformer rooms, etc. When evaporated C.T.C. forms fumes which are toxic; hence the extinguisher should not be used in confined space and in underground mines. The C.T.C. fire extinguishers are used on motor cars or automobiles as handy units in 2-litre capacity models. The liquid is discharged by gas kept under pressure inside the cylinder.

These extinguishers are being gradually withdrawn from use.

B. C. F. Fire Extinguishers :

In recent years a liquid called B. C. F. (bromochlorodifluoromethane) is gaining popularity as a fire extinguishing medium in place of C. T. C.; B. C. F. is a clear, colourless liquid at normal atmospheric temperature with the following properties.

Formula : $CBrClF_2$
Molecular weight : 165.4
Boiling point : 80°C
Freezing point : minus 4°C
Odour : Similar to the halogen group of compounds
Liquid sp. gr. at 21°C : 1.83

Effect on human flesh : BCF does not present any hazard by absorption through the skin. Having a low boiling point the liquid evaporates quickly and gives a chilling sensation if spilled onto the skin out it cannot cause low temperature burns as CO_2 does.

Effect on metals and non-metals : BCF is chemically very stable and it shows less tendency to cause corrosion like CTC does. In the absence of water BCF can be stored satisfactorily in containers made from most common metals. BCF causes natural rubber to swell but its effect on nitrited rubber is very small and these can be used satisfactorily for gaskets and joint rings.

Electrical conductivity : BCF is a non-conductor of electricity and can therefore be used to deal with class E fires. B.C.F. is also known as Halon. Halon 1211 is used for extinguishing fires.

The suitability of different types of fire extinguishers is summerised as follows : Class A fire – Soda acid type; Class B – Foam type; Class C – Dry chemical powder; Class D – Special dry chemical powder; Class E – CO_2 type and BCF. Halon 1211 BCF type is recommended also for fires involving electronic equipment. It is imported and may not be available easily these days.

FIRES IN MINES :

Fires in mines may be at the surface or in underground part of a mine. Surface fires in coal mines are in

1. Store yards, engine/transformer houses, offices and places.
2. Overburden dumps in coal mines.
3. Exposed coal in quarries.
4. Coal stacks.
5. Coal in bunkers of coal handling plants.

Surface Fires :

1. *Fire in store yards, engine houses, etc. :* The causes of such fires are generally.

i. *Electrical :* Short circuiting of electrical wiring, sparks from welding apparatus or electric motors and equipment.

ii. *Embers of coal or other fires :* Insufficient care or negligence causes spread of fire due to these reasons.

iii. Carelessly thrown cigarette, biristubs at places containing kerosene oil, lubricating oil, grease, oil-soaked cotton waste, paper and other combustible material.

iv. *Hot surfaces :* Hot surfaces of electrical motors, steam engines or internal combustion engines in contact with combustible material; hot wooden brake blocks of winders and other engines.

2. The possibility of fire in overburden heaps near quarries is generally ignored. Inferior coal and shale is very often thrown into overburden dumps. In a large number of cases, the shale bands so dumped have some bright coal attached to them. In the overburden dump these carbonaceous materials catch fire due to spontaneous heating. Another reason for fires in overburden dumps is the burning of coal over these dumps in winter nights by quarry workers to get warmth. The heat of such fires penetrates down the debris and ignites the carbonaceous matters concealed below the surface. If the overburden dump touches the coal seam, as is not uncommon, the latter also catches fire.

In mines where soft coke is manufactured by the usual crude method, the process is carried on, sometimes, over the levelled surface of overburden dump. The heat results in ignition of rejects and carbonaceous matter inside the overburden dump. The hot ashes, after the soft coke is prepared are allowed to remain at the site and they also contribute towards heating of the carbonaceous matter in the overburden dump.

Sometimes hot boiler ash is dumped on the overburden heap.

Fires in overburden dumps are noticeable at many quarries in Jharia, Ranigunj, West Bokaro (Kedla-Jharkhand) and other Coalfields. If such dumps are on the cracked surface of a goaf area the heat travels to the underground mines through the cracks resulting in ignition of coal left in the underground mine during depillaring.

The remedy to prevent such fires in overburden dumps is obvious. The D. G. M. S. Circulars direct that overburden heaps should be atleast 15 m away from the coal face of a quarry. If all the coal of a bench right upto the sand stone floor is extracted and the distance of 15 m or more between overburden dump and coal bench is maintained fire from overburden will not

spread to coal bench. If a quarry is abandoned the coal bench and overburden dump should be separated by digging a trench 6 m to 10 m wide upto the non-carbonaceous and incombustible rock (like sand stone) below the coal seam. A better arrangement is to flood the trench with water. Hot boiler ashes or other hot materials dumped at the outcrop of the coal seam or on quarry floor can also set the coal seam on fire which engulfs not only the quary coal but also spreads to underground parts.

Fire in an adjacent mine often spreads to an unaffected mine through the barrier between the two. Such examples are common in the Jharia Field near Phularitand, Angarpathra, Ena, Khas Jharia, Jogta and many other places.

(3) & (4) The fires in surface coal stocks are mainly due to spontaneous heating. Results on some field investigations are : Kajora Seam coal (Ranigunj Field) catches fire in stocks of 70-80 te within 2-3 weeks; Jhingurda seam (Singrauli Field) coal catches fire in 10-15 days' time even in the coal benches; in Bishrampur Colliery slack coal stock of 2 m height took nearly 55 days of incubation period before it burst into flame. A 2.4 m high stock of slack coal of Jambad Seam took nearly 1.5 months to reach a temperature of 85-90°C where it remained fairly steady for a week (upto 90-95°C) and then burst into flame within the next 48 hours.

Preventive Measures :

Spontaneous heating in coal stock can be prevented by adopting the following measures :

i. The stacking ground should be hard and firm, free from growth of any vegetation. A concrete surface is ideal.

ii. Coal should be screened and a stack should contain coal of one size only. Fires easily take place in coal heaps where the coal of different sizes (R. O. M. coal) is stacked together.

iii. Shale pieces in contact with coal are known to accelerate spontaneous heating. For example, in the coal at Jambad seam (Ranigunj Coalfield). They should be picked out.

iv. The coal stack should not exceed 200 tonnes and its height should not be more than a critical height. The critical height varies for different coals and is generally between 1.5 m and 3m. It can be ascertained only after experience or after knowledge of coal seams in adjacent mines.

v. The coal stacks should be cleared on the basis of first come first removed so that only fresh coal is available in the stacks.

vi. The coal stack should be compacted by dozers from time to time as it builds up. This retards the process of oxidation.

vii. If possible, the coal should be stacked over a net work of criss-cross perforated iron pipes through which water can be circulated at the slightest detection of heating.

viii. In the coal stacks iron pipes 50 mm dia., pointed and closed at the lower ends should be fixed at intervals. A thermometer can be inserted in the pipe to note the temperature of the coal stock. If the pipes attain a temperature too hot to touch by hand (nearly 60°C), it is an indication of accelerated spontaneous heating and steps should be taken to tackle it without delay.

ix. Water is the best fire fighting medium for surface fires. Water pipes of 100 mm bore, should be laid near the coal stocks, and branch pipes to accommodate 50 mm canvas hose pipes should be provided on the water mains.

x. To cut down supply of oxygen the surface of coal stocks may be coated with mud (mixed with cowdung) or coal tar. A coating of asphalt or mobile emulsion is also helpful.

At Singrauli Colliery, Jhingurda Seam coal is so prone to spontaneous heating that it catches fire in about a fortnight in the coal bench itself. C. F. R. I. had suggested the following remedial measures.

i. Accumulation of slack/coal in the floor of the quarry benches should be avoided.

ii. A suitable protective coating (tar/burnt mobile oil, etc.) be applied to the face of the coal bench where fissured and crushed coal layers are met with, making them virtually impervious to air entry.

An emulsion of burnt oil and bitumen in the ratio of 3 : 1 was spread over an experimental face. The treated area, over 1000 m², showed no signs of heating for over 6 months though many untreated areas of the vicinity had caught fire. The coating fairly withstood moderate rain showers. Recoating was made only once in 6 months.

iii. Covering of coal stacks with polythene sheets 0.2 mm thick restricts air feed to the coal.

iv. A fire resistant sealant of the MOWOTWOS as used in Poland for avoiding air leakages in longwall faces may, with certain modifications, be useful in giving fire resistant air seal/coating over the patches of fires in the quarry benches. In some cases slack coal has been used successfully for covering the coal stock.

v. Coal should not be stocked near any source of heat like boiler house, boiler ash heap, steam pipe range, etc.

Dealing with fires in surface coal stock :

i. Steam rising from a coal stock is a warning that heating is taking place. Presence of moisture on surface of coal stock early in the morning or late in the evening should not be mistaken for "dew". These are indications of the start of heating. Observation of temperature is of course the most reliable way to detect heating.

ii. Seat of heating in a coal stock is nearly 0.6 to 1.5 m below the surface of stock.

iii. The coal stock in which heating has started should be drenched with water jets under pressure. The quenched coal should be dug up and spread in thin layers about 0.6 m thick at another place and allowed to cool down. Such cooled down coal should be the first to be despatched.

5. Fire in the coal of bunkers (Coal Handing Plant) can be prevented by keeping a watch on the heating which may take place if the coal is not despatched due to shortage of wagons. When such heating is detected the coal should be desptached by trucks to a convenient place where it should be treated with water and the quenched coal should be spread on the ground for further cooling in thin layers.

SPONTANEOUS HEATING OF COAL :

Spontaneous combustion of coal or other carbonaceous matter may be defined as the process of self heating resulting eventually in its ignition without the application of external heat. The term is used generally for coal as spontaneous heating of other minerals is very rare.

When coal is exposed to air it absorbs oxygen at the exposed surface. Some fractions of the exposed coast substance absorbs oxygen at a faster rate than others and the oxidation results in formation of gases, mainly CO, CO_2 and water vapour alongwith evolution of heat during the chemical reaction. The process takes place even at normal atmospheric temperature but it is slow and the heat evolved is not perceptible as it is carried away by the air unless the latter is stagnant. If, however, the rate of dissipation of heat is slow compared with the evolution of heat by oxidation, there is a gradual build up of heat and slow rise in the temperature of coal. At the raised temperature the process of oxidation is slightly accelerated and some other fractions of coal become susceptible to oxidation. A stage is reached when the build up of heat and the rise of temperature reaches the ignition point of coal which then catches fire. A good air current will effectively prevent undue increase of temperature, absence of air will prevent oxidation ; and somewhere between these two extremes conditions may permit marked heating to take place. Once the coal reaches its ignition point (as distinct from slow oxidation), the air supply to it will only increase the combustion. The ignition temperature of bituminous coal is nearly 200°C and of anthracite coal, nearly 398°C. The coal may be smouldering in the beginning but it may soon break up into flames if sufficient oxygen of fresh air feeds the hot coal. This process of self heating of coal resulting ultimately in its combustion is known as spontaneous combustion.

Factors governing spontaneous heating :

1. **Chemical composition of coal :** High moisture and high volatile coals are more susceptible to spontaneous heating. all bright coals with 25% or more of V. M. and 7 to 15% of moisture are prone to spontaneous heating. Moisture does not assist directly in oxidation of coal but as it dries up the coal disintegrates and the disintegrated coal presents more surface for contact with air and oxidation.

High rank coal with high carbon content (say, over 85% d. m. m. f. basis) is less liable to spontaneous heating. Lignite and bituminous coals are susceptible to spontaneous heating but fires are practically unknown in anthracite coal mines.

The proneness to spontaneous heating of coal decreases with decreasing oxygen content in the V. M. of coal. With oxygen content of 2% or less the coal is not liable to spontaneous heating.

2. **Banded constituents of coal :** The bright bands of coal viz. vitrain and clairain are more liable to spontaneous heating than the dull constituents like durain and fusain. Durain is hard and difficult to fracture and resistant to self heating. Fusain consists very largely of resistant materials but it is a porous, flocculent powder which presents a large surface to the air. It therefore oxidises rapidly at low temperatures but it forms only about 5-6% of the coal and is thus not so important factor in spontaneous combustion as vitrain or clairain.

3. **Friability :** Coal which is easily crushed and borken into smaller size is more liable to spontaneous heating than hard coal. Disergarh seam (Ranigunj field), and XIV and XV (Jharia Field) seam coals are typical examples.

4. **Presence of iron pyrites :** Coal containing iron pyrites in disseminated form is much liable to spontaneous heating, e.g. some coal seams in Pench Valley coalfield. Iron pyrites, during oxidation swell and cause the coal to disintegrate. The broken coal presents more surface for air circulation and is also slightly warmer due to the heat of oxidation of pyrites. This results in acceleration of the process of spontaneous heating.

5. **Nature of adjoining strata :** Thermal conductivity of coal measure shales is only 1/3rd that of the sandstones. If a coal heap is covered by loose shales, the heat of oxidation of coal is not dissipated as fast as in the case of coverage by sandstone and the former heap is more liable to spontaneous heating.

6. **Depth of seam :** The strata temperature and crushing effect of superincumbent rocks of a coal seam increase with increasing depth. Both the factors accelerate the process of spontaneous heating.

7. **Thickness of seam :** It is difficult to remove all the coal in a thick seam during depillaring by caving. The difficulties in extraction of coal by caving often results in nearly 50% extraction only and the remaining coal is left underground in the form of stooks, coal in the roof or coal in the parting between adjacent section. Such coal lying in the goal provides suitable material for spontaneous heating. Coal of the high stooks is easily crushed and such crushed coal is readily attacked by the oxidation process. Another contributory factor for spontaneous heating in thick seams is the slow velocity of air current in high galleries; due to slow velocity the heat of oxidation is not removed fast enough.

Slack coal or inferior coal, purposely left underground due to poor marketability, is also responsible for spontaneous heating.

8. **Geological disturbances :** Near a fault plane the coal and other strata are usually crushed and not hard enough. Such crushed and weak, friable coal has to be left in-situ for support near a fault zone to prevent a rock slide along the fault plane. Such coal is more liable to spontaneous heating than the comparatively harder coal at places away from the fault zone. In the Jharia field typical examples are the coals in seams at Bhulanbararee, Sudamdih, Chasnala, Bhowrah and other collieries due to the effect of Bhulanbararee thrust.

Incubation Period :

This is the term to denote the period which elapses between the time when the coal is first subjected to conditions favourable for spontaneous heating and the time of indications of heating.

In a coal mine having depillaring with caving the coal left in situ in stooks is burried under broken rocks of roof after the first major roof fall takes place. In a seam thicker than 3 m it is not always possible to extract all the coal upto the roof and if the latter consists of immediate shaly band, coal is left intact against the roof during development as one of the measures in roof support. Such roof coal falls in the goaf after withdrawal of roof supports during retreating in the caving method and it gets buried after the first major roof fall. The first major roof fall therefore creates conditions under which the coal oxidation heat cannot be dissipated fast enough with the result that the heat of oxidation builds up leading ultimately to spontaneous heating. The term incubation period has therefore special significance in a depillaring mine as all the depillaring operations should be over in a mine or a panel during the incubation period so that the depillared area can be sealed off by stoppings.

The incubation period cannot be measured by any instrument and has to be judged by experience gained at the mine or adjacent mines working the same seam. It varies from seam to seam, and is observed to be as follows for some seams.

Jambad seam (Ranigunj field)	..... 15.20 weeks
Churi colliery (North Karanpura field)	..... 15 weeks
Makum coalfield (Assam)	..... 10 to 21 weeks
Neyveli lignite	..... 8 to 10 weeks
Bisrampur seam	..... 8 weeks

Some coal seams are so liable to spontaneous heating that the coal stocked in wagon loading bunkers catches fire if not removed within 2-3 weeks. Even the exposed coals in mechanised quarries sometimes catch fire within the incubation period, much less to speak of coal stacked on the surface.

Symptoms of spontaneous heating in underground mines :

The symptoms can be generally considered in the following stages :

(a) Initial stage of heating :

i. *Faint haze :* This is due to moisture given off during oxidation of coal. The moisture collects as small globules in the cooler air away from the actual seat of heating.

ii. *Moisture deposition :* This is due to the condensation of moisture and its deposition as beads on the cooler surfaces (roof, sides, timber and metal surface).

iii. *Faint odour known as gob stink :* The odour resembles the smell of decaying timber. There is also slight discomfort due to increase in air temperature and humidity.

iv. Cricket and other small insects show increased activity and chirping.

(b) Intermediate stage :

Previous symptoms are intensified and there is a further pronounced petrol like odour indicating the beginning of distillation of coal.

(c) Last stage of heating approaching ignition :

The petrol like odour changes into tarry odour, sometimes known as 'Firestink' which is also due to distillation of coal. Further stage is the actual appearance of smoke which may travel sometimes against the air current in the intake air way also.

It may be noted that the various symptoms stated above usually overlap and if the ventilation is strong enough the air current may carry off or dilute the smells which are noticeable in the initial stages of heating. At times, the smell of other materials may obscure the actual smell which arises out of self heating. Such smells are due to :

i. decay of wood in warm damp places,

ii. tarred brattice cloth,

iii. lubricating oil, etc.

If systematic records of dry and wet bulb hygrometer are maintained for the return airway of the mine such records give an indication of initial stages of heating, because increased temperature, especially of the wet bulb, generally indicates incipienheating.

Analysis of return air gives indication of various stages of heating in underground coal mines. But these involve collection of air samples and tests in the laboratory which take quite some time and therefore, the supervisory staff should carry out regular inspections of underground districts to detect any unusual odours and symptoms in the return air. It is truly said that "there is no scientific instrument to equal the nose". The underground observations by the regular supervisory staff and the results of gas analysis should go side by side.

Interpretation of mine air samples :

Interpretation of mine air samples is necessary to determine

1. Percentages of various gases like oxygen, CH_4, CO, CO_2, N_2 in the return air of a normally working mine, and at other places of suspicion.
2. Existence of fire or spontaneous heating in some part of a mine.
3. The inflammability (or explosibility) of the atmosphere in the return side of a fire when the latter is being tackled by direct methods of fire fighting.
4. The condition of fire in a sealed off area.

In a normally working mine the slow oxidation of coal produces small quantities of CO and CO_2. The percentages of these gases in the return air remain nearly constant in the samples taken periodically from the return air and any unusual increase in the percentages needs immediate investigation.

The CO_2 is produced not merely by slow oxidation of coal but by various other factors but CO is produced mainly by slow and incomplete oxidation of coal if diesel locomotive or other internal combustion machinery is not used underground. The percentage of CO in a normally working mine may be about 0.005 or even lower in the return air. Though an increase in the normal percentage of CO should cause anxiety, it is the ratio, CO produced/oxygen consumed, (also called CO/oxygen deficiency), which has been found to give indication of spontaneous heating and its stage. The ratio CO produced/oxygen consumed is usually between 0.1% and 0.5% in the main return of a working mine with adequate ventilation but it may be higher, even upto 1%, for samples taken at the working coal faces. In the case of human breathing, burning of lights and decay of timber no CO is produced and the ratio CO/oxygen consumed is nil.

With spontaneous heating of coal in underground mines the ratio CO/O_2 deficiency gradually increases and in general terms it may be summarised as follows :

CO/O_2 deficiency : 0.1% to 0.5% is normal to a coal mine (in extreme cases only it may be upto 1%)

: 1% indicates existence of spontaneous heating.

: 2% indicates heating in advanced stage approaching active fire.

: 3% or more indicates active fire.

One significant feature of CO/O_2 ratio is that it is independent of dilution of samples by CH_4 given from the strata and coal seam.

The following example shows how the CO/O_2 and CO_2/O_2 ratio is calculated :

Example 1 : The percentages of various gases in the return air of a normally working mine are as follows :

Oxygen	19.95
Nitrogen	78.72
Methane	0.93
Carbon dioxide	0.39
Carbon monoxide	0.005

Calculate the CO/O_2 deficiency and CO_2/O_2 deficiency ratios.

Ans. : In the atmospheric air which goes down the mine the percentage of oxygen is 20.93, of nitrogen (including inert gases like arogon) is 79.04 and of CO_2 is 0.03.

As the percentage of CO is very small it is calculated to the third place of decimal and other gases, to second place of decimal.

The oxygen corresponding to 78.72 parts of N_2 will be

$$\frac{20.93}{79.04} \times 78.72 = 20.85\%$$

Oxygen absorbed = 20.85 – 19.95 = 0.9%

CO_2 produced = 0.39 – 0.03 = 0.36% and this, expressed as a percentage of the oxygen absorbed = (0.36 ÷ 0.9) × 100 = 40%

CO produced = 0.005%

Percentage of CO/O_2 absorbed = (0.005 ÷ 0.9) × 100 = 0.56

Example 2 : The following is an analysis of a sample of return air in the same mine at one stage. (percentage). What do the figures indicate?

Oxygen	19.90
Nitrogen	78.67
Methane	1.00
Carbon dioxide	0.40
Carbon monoxide	0.03

Ans. : In the above example by calculation we get $\frac{CO_2 \text{ produced}}{O_2 \text{ absorbed}}$ as 40% and CO produced/O_2 absorbed is 3%. The results indicate dangerous heating. Note the low value of CO percentage as compared with the relatively high value of CO/O_2 absorbed ratio.

Methods of working coal seams liable to spontaneous heating :

1. If the seam is to be worked by bord and pillar method without stowing during the depillaring stage, the mine should be divided into districts or panels and each panel should have only such number of pillars as can be extracted during depillaring within the incubation period. If the mine is already developed without panels artificial panels should be formed by brick walls.
2. The dimensions, size and shape of the pillars should be such that no crushing of pillars takes place when they are formed or when they are under extraction.
3. The thickness of barriers between adjacent panels should be of minimum one pillar corresponding to the depth of the place.

4. If depillaring is not to be undertaken for a long time after formation of pillars they should be treated with stone dust or guniting to reduce oxidation, and the panel sealed.

5. If the mine is to be worked by longwall method of mining without stowing, retreating longwall should be adopted.

6. The longwall faces should be in panels which can be sealed off in case of emergency.

7. Main intake air road and main return air road should be kept as wide apart as possible to prevent air leakage.

   Sufficient ventilation should be provided at low water gauge.

8. The entries to the panels should be minimum and provision should be made for construction of isolation stoppings at the entries. Sufficient supply of bricks and other construction materials should be kept at sites of isolation stoppings. Recesses should be cut in advance in the roof, floor and pillars for speedy erection of isolation stoppings.

   If the width of road is more the stopping should be partially built to permit just enough width for passage of tubs and men. Pre-fabricated isolation stoppings are recommended.

9. If the work of coal extraction in a panel is over, remove all the equipment and seal off the panel by isolation stoppings to prevent entry of air. Before such sealing the area should be heavily stonedusted.

PREVENTION OF UNDERGROUND FIRES :

Apart from taking steps against spontaneous heating of coal in the case of a coal mine, other measures to prevent accidental fires in underground part of a mine are :

1. Check the workers, before they proceed underground, for match box, cigarettes, biris, cigarette lighters and other contrabands. This is required under the coal mining Regulations.

2. Do not allow burning of fires inside a mine (non-coal mine) and within 15 m of an incline/pit.

3. Avoid welding of the headgear pulley or the headgear frame unless proper precautions are taken. Such welding in the case of a coal mine requires previous permission of the J. D. M. S.

4. Avoid welding in underground repair shop without adequate precautions. If any repair shop/workshop is to be provided underground in a coal mine prior approval of the J. D. M. S. is essential in case the repair shop is to be equipped with welding arrangements.

   The important conditions which are usually associated with such permission are ; (a) The repair shop should be in the main intake airway near the d. c. shaft (b) Ventilation should be adequate. (c) Methanometers should be used to keep a watch on the percentage of firedamp during the time of welding (d) The welding operation should be conducted in the presence of a responsible official, preferably a senior overman, an Assistant Manager, or a Ventilation officer, and (e) Fire fighting equipment should be ready at hand.

5. Restrict the storage of inflammable and combustible materials like oil, grease, timber, etc. underground.

6. Remove all wood cuttings, oily and greasy cotton waste out of the mine.

7. Install the electrical cables and equipment with adequate care and maintain them properly with regular inspections. A roof fall can damage a cable and rupture it resulting in short circuit and spark. Cables should therefore be laid along routes which are normally expected to be free from danger of roof-fall. Avoid cables in roads with rope haulages to prevent damage by derailed tubs.

   Use electric motors, switchgears and transformers of flame-proof type in underground coal mines. These should be of adequate size and rating. A motor of insufficient power and rating, if worked for long hours, gets too hot and the heated surface can provide a source of ignition.

8. Use only approved safety lamps which should be taken underground in locked condition. Test the lamps for their defects in the lamp cabin and again before the worker enters the pit. Any damage to a safety lamp must be reported to the mining sardar or overman by the user immediately.

9. Do not use any naked lights, or stove in underground part of a coal mine.

10. Machinery used underground should be properly assembled and operated so that during use it does not cause dangerous sparks or hot surfaces. At Chinakuri a diesel locomotive flame trap which was not suitably assembled was believed to have been the source of ignition resulting in the explosion.

11. Brake blocks of underground machinery like haulage engines, locomovites, etc. should be adjusted periodically to avoid their overheating.

12. Avoid accumulation of dangerous electrostatic charge on equipment using comprssed air by earthing.
13. Ensure that the conveyors are in proper alignment and their idlers and rollers moving freely. The belt should not rub against any timber, metallic surface or rock.
14. Face machinery like conveyors, coal cutting machines and drills should be operated by remote control arrangment.
15. Use conveyor belts made of P. V. C. which is fire resistant. The brattice cloth used for coursing the air should also be fire resistant.
16. Minimise frictional sparks arising from the use of coal cutters, shears and similar equipment by use of sharp cutter picks. In conjunction with wet cutting the sparks are rendered harmless.
17. Use only permitted explosives in underground coal mines. Proper care should be taken in shotfiring to ensure correct charge and burden so that blown out shots are avoided. All the shot-holes should be carefully checked for the presence of cracks. For this the shotfirer should be properly trained.
18. Do not work in a coal seam lying below a fire area or another seam on fire without permission from the D. G. M. S. Provide barriers of sufficient thickness against existing fire in the same mine or adjacent mine.
19. Check the underground workings, specially at the goaf edges, regularly for any unusual rise in temperature or other symptoms of spontaneous heating. Special care should be taken to see that inspections are not neglected on non-working days.
20. In abandoned coal inclines guard against illicit distillation of liquor as such practices have been noticed in a number of mines. The fires ignited for such illicit distillation are very often not extinuished by the persons indulging in such antisocial and illegal activities.
21. If some pillars of coal have developed cracks they are sources of leakage from the intake to return side and give rise to spontaneous heating. The trouble is accentuated with high w. g. of the ventilating air. Such cracked pillars catching fire due to spontaneous heating were noticed in Jambad seam. The remedy lies in having large sized pillars, guniting them, scaling up the cracks with cement mortar and circulating the air at low w. g.

Now a days in highly productive coal mines abroad, at strategic points in underground mines, CO sensors are installed and their values are automatically monitored in a room on the surface.

DEALING WITH UNDERGROUND FIRES :

When the foreman/overman, or Asst. Manager notices any indication of heating or fire in any part of mine he should withdraw workers from the ventilating district and the area likely to be affected (e.g. area on the return side of the ventilating district), cut off electric power to that area and inform the mine manager, Symptoms of heating or fire are more pronounced on the return side of the fire or place of heating. One method of informing all the workers of the emergency is to pour eucalyptus oil in the intake air stream. The smell which spreads in no time to all the parts of the mine with ventilating air current warns the workers. Such steps are taken in the mines of Kolar Gold Field.

The mine manager, on getting information of fire, has to inform the J. D. M. S. and also the nearby rescue station if he considers the presence of the resue brigade essential; in any case, he should advise the rescue station to keep the rescue brigade in reserve even if the fire is small and may not demand immediate arrival of the rescue team.

The following measures may be taken in dealing with the fire.

(A) Quenching the fire with water, sand or with the suitable fire extinguisher. This direct attempt is justified if the fire is small and accessible. Ventilation to the fire area is restricted by short circuiting the major quantity of ventilation. The fire should be tackled only from the intake side, and water should not be used to quench electrical or oil fires. The quenched material is dug up and sent to the surface in specially marked tubs/mine cars. The dug up area is drenched with water and allowed to cool down. The cavity is packed with sand or earth. During the process of the fire fighting, flame safety lamps and CO detecting apparatus, or alternatively small birds for CO detection, should be kept at the site. Digging out is possible if (1) the fire is easily accessible, (2) localised over a small area (3) there is no danger of firedamp explosion, and (4) the roof is good and has not collapsed due to fire. Accidential fires, if detected in time, are not extensive and can generally be dealt with by the direct method of attack. A small accessible fire can be covered with a blanketing material like earth or sand to starve it of oxygen. The blanketing has to be done not only on the burning material but also at the cracks or passages of air supply to the fire. In one case where flames were seen emerging from the roof of a roadway, air entry to the fire was found to be through cracks about 30 m away.

If the fire cannot be tackled by the above method one of the following measures has to be adopted.

(B) Sealing off the fire

(C) Drowning the fire

(D) Flooding the fire aea with inert gas

(B) Sealing off the fire :

If the fire is inaccessible, e.g. in a goaf area, or if it is extensive, or if there is danger of firedamp explosion when dealing with it, it has to be sealed off. The aim is to prevent access of air to the fire and starve it of oxygen. The sealing operations have to be conducted from places which will not be affected by the effects of fire like heat, smoke, CO, CO_2, etc. A large area has therefore to be sacrificed during sealing though it may be recovered later when the fire is extinguished and the area cooled down. Time factor is important in sealing. Delay will cause the fire to spread over a large area and the extensive fire may go out of control resulting in abandonment of the mine; it may spread to adjacent mines as well. In a gassy coal mine the fire may spread to pockets of explosive gas-air mixture and result in explosion.

The stoppings constructed to seal a fire area are of the following types depending upon the purpose they serve.

1. Preparatory stopping
2. Emergency stopping or temporary stopping
3. Permanent stopping

Preparatory stoppings are partially constructed in the main entries leading to a district, generally before depillaring is undertaken. After the depillaring operations are over such stoppings are made complete to isolate the depillared are then called isolation stoppings. Bricks and earth are always kept in readiness at such stopping sites. Since the preparatory stoppings serve ultimately the purpose of isolation they are often called isolation stoppings even when they are partially constructed and are not actually isolating an area. The preparatory stoppings have to be provided in all mines, whether depillaring is to be carried out with caving or with stowing. Where depillaring with caving is planned such stoppings should enclose an area and form a panel from which all the coal can be extracted during depillaring within the incubation period. If depillaring with stowing is planned, the area enclosed is permitted to be much larger (D. G. M. S. Circular No. 55 of 1962). A preparatory stopping should be provided with an iron door which is generally taken off the hinges and kept beside the stopping. In case of an emergency like fire in the panel the door is very handy in isolating the panel from the rest of the mine.

An isolation stopping in Cat. I & II gassy coal mine should be of a miniumum thickness of 1 m, brick in cement or lime. Regular air samples should be taken from behind the sealed area and whenever it is found on analysis that CH_4% has increased to 2 or more, the stopping should be strengthened so as to make it explosion proof. In Cat. III gassy coal mine, an isolation stopping should be explosion proof.

An explosion proof stopping should consist of a pair of two brick stoppings built in cement mortar having a minimum thickness of 1m and spaced at least 4.5 m apart. The intervening space between the two stoppings should be packed solid with incombustible materials. (There should be no coal pieces or other carbonaceous matter in the packing).

The isolation stoppings should be well keyed into the roof, floor and sides, and for this purpose, the minimum depth of locking should be as follows :

i. In coal 1.0 m

ii. In sand stone roof/floor, 15 cms.

iii. In shale stone roof/floor, 30 cms.

Emergency stoppings or temporary stoppings are constructed after heating or fire is detected. Such stoppings are required where preparatory or isolation stoppings had not been constructed *e.g.* in a development area where there is no necessity of their construction. The purpose of an emergency stopping is to seal on area immediately and prevent access of air to the fire which dies down in course of time. It also prevents the heat, smoke and fumes from reaching the places where permanent stoppings have to be constructed. The temporary stoppings have to be built at places which are free from ground movement and free from cracks as such cracks provide passage of air to the fire. If a place free from cracks is not available the cracks should be sealed :

1. By guniting, i.e. spraying the surface with liquid cement sand mixture (sand : cement in the ratio of 3 : 1), or
2. By spraying the surface with Latex (a trade name) with the help of compressed air operated gun.

Both these methods require compressed air which may not be available underground in most of the coal mines, as electric power is the common practice.

Pillars with cracks are common in the vicinity of depillaring areas, stopping areas and also in other areas which are subject to crushing due to small sized pillars or robbing of coal and heightening/widening of galleries.

Sealing a fire in a non gassy mine (deg. 1 gassy mine) :

The method of sealing a fire depends upon whether the mine is gassy or non-gassy. In a non-gassy mine or Cat. I mine the standard practice is to immediately stop air supply to the fire by construction of temporary stoppings in the intake and return roads of the district and in other roads through which air may have access to the fire. This is followed by construction of permanent stoppings. During such work all workers in the ventilating district affected by the fire should be withdrawn.

4th #

Construction of temporary stoppings :

These are constructed in one of the following ways :

1. Stopping made by nailing wooden planks to timber props.
2. Stopping made of C. G. I. sheets fastened to timber or rail props and having piles of sand or stone dust bags behind them.
3. If stone of ripping is available underground, a packwall of stone plastered with earth or cement.
4. If sufficient number of bricks are available underground, in a nearby place because of previous storage, a stopping of brick-in-mud.
5. A stopping consisting of a close knit wire mesh stretched across the roadway and sprayed with Latex sealant.
6. A pre-fabricated stopping of tongued and grooved wooden boards.
7. A sand bag stopping. This is common and easily constructed. Empty cement bags are filled to half their capacity with earth. Such half filled bags are easy to carry and they make a compact packwall when piled one over another. The surface of the packwall is plastered with earth or cement.
8. Gypsum stoppings. These are constructed of a quick setting gypsum. They have been tried successfully in foreign countries like Germany and U. K. This type of stopping consists of two tight shutterings between which is injected dry gypsum powder by compressed air driven guniting machine. As the gypsum powder is being injected, water is added up to form a slurry which soon sets and forms a hard mass. The recommended thickness of stopping is : 2m for roadway height upto 3m ; 2.5 m for roadway height upto 3.5 m.

For every m^3 of stopping volume about 1.3 te of gypsum and 1 m^3 of water is required.

The gaps between the shutterings and pillars of coal/mineral are packed with gunny bags to prevent the slurry from flowing out. A sampling pipe is to be provided through both the shutterings.

Such stopping is simple and easy to erect.

The construction of fire stoppings in an emergency calls for a good streamlined organisation. The following points should be noted.

1. Stores should have adequate stocks of bricks, cement, C. G. I. sheets, rails, sleepers, empty cement bags, stone dust, etc.
2. Haulage track should be extended to the stopping site for transport of heavy quantities of bricks, cement, sand, earth, etc. It may sometimes be necessary to install a small haulage also.
3. Work of stopping construction should be done by workers who should not work overtime. A plan should be drawn out to decide the number of workers required and their shifts. It may be necessary to take help from adjacent collieries, specially in respect of rescue trained workers and stores materials. An officer should be incharge of man power and allotment of the duties of different workers, the arrangement of their shifts, etc.
4. At the site of stoppings the following equipment should be provided : (i) Flame safety lamps for detection of CH_4 (ii) Birds for detection of CO, (iii) Sufficient stone dust and fire sealing materials, like bricks, cement, sand, earth and tools of masons, carpenters, miners, and other workers, (iv) First aid boxes and stretchers, (v) Extra electric cap lamps as some may become dim after 5 or 6 hours use, (vi) Smoke helmets and other apparatus required by rescue party, (vii) Drinking water and snacks, (viii) Mine plan and tracings of fire area on a large scale.

Telephones should be extended upto the site of construction. The work should be done under the supervision of experienced officers.

Imp **Permanent stopping** of brick in cement or lime are constructed outbye of the temporary stoppings after a lapse of nearly 48 hours. A permanent stopping is the last step to seal off an area and the work should be reliable. The sites of permanent stoppings are selected at places free from cracks and having the minimum cross section. The stopping should extend well into the floor, sides and roof intact up to the stone in the roof and for this purpose the cutting should be by miner's picks without the use of explosive which may cause cracks leading to leakage. The thickness of the stopping is nearly 1m near the roof and it increases by 15 cm for every 3m of height. In some mines like Jealgora stoppings had to be constructed in roadways 7m high and in such roadways bottom 3m section would be 1.30 m thick, the next 3m section. 1.15 m thick and the rest 1m near the roof, 1m thick, Special attention should be paid to leying of the stopping against the roof as it is a comparatively difficult taks, specially under restricted conditions of ventilation. The offset, if any, should be on the fire side so that the outbye side presents smooth surface which can be plastered and white washed for easy detection of cracks. In thick seam some time should be allowed for the brick construction to settle before the final jamming of the stopping against the roof, since the brick wall tends to shrink when the mortar is wet.

A fire stopping should have the following fittings :

i. Water gauge,

ii. Thermometer,

iii. A sampling pipe, 18 mm dia, with a valve for collecting samples of air from behind the stopping. The sampling pipe should pass through the temporary stopping and should extend about 3 m beyond it towards the fire side. One design of sampling pipe consists of a 5 cm dia. pipe 1.5 m long, grouted in the stopping and provided at its outbye end with a socket which takes a brass plug; to the brass plug is fitted a 6mm dia. copper tube in 2m lengths screwed together; the copper tube extends inbye the stopping to a minimum of 4.5 m and is supported on a brick pillar. (Fig. 4.2)

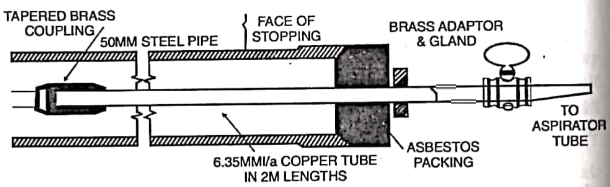


Fig. 4.2 Sampling pipe

iv. 50 mm dia. m. s. pipe with plug for recording temperature of the atmosphere behind the stopping.

v. Water seal at the floor level for drainage of water. A water seal consists of a piece of pipe built into the brickwork at floor level with a bend at its outbye end which dips into the water contained in a small reservoir. The reservoir may be a ditch cut in the floor or a small brickwork tank. The pipe is a ditch cut in the floor or a small brickwork tank. The pipe is sometimes provided with a spring loaded valve. When sufficient water

accumulates behind the stopping, the valve opens and allows the water to flow outbye but as soon as water is drained out, the valve closes under action of the spring. As the end of the pipe remains always under water, air cannot leak in.

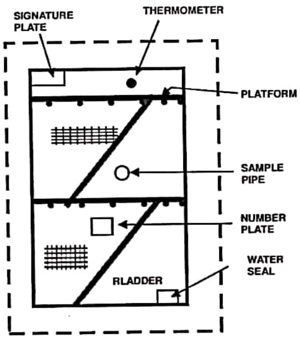


Fig. 4.3 Fire stopping in a thick seam

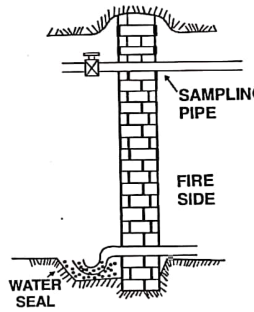


Fig. 4.4 Water seal

vi. A number plate giving the reference number of the stopping.

vii. A signature board for the overman to put his signature and date after inspection of the stopping. If the stopping is high such board should be at the top and ladderways or platforms should be arranged zigzag so that all parts of the stopping can be easily insepected by the overman as he travels to put his signature on the board. (Fig. 4.3)

Position of a sampling pipe in a stopping :

Atmosphere behind a stopping is still and the gases do not diffuse. The heavier gases like CO_2 settle to the floor level and the lighter gases like methane remain at the higher level. To get an air sample representative of the conditions behind the stopping, the sampling pipe should not be near the floor, nor near the roof but it should be preferably above the floor level at a height between ½ and ¾ th the stopping height. Widthwise, it should be at midwidth of the stopping. It should extend at least 3 m inside the stopping and to prevent its bending, should be supported on a brick pillar or a wooden prop.

If there is any leakage of air through the stopping, it is noticed by black streaks indicating deposit of soot or smoke at the cracks. Stoppings should always be kept whitewashed on the surface and at the roof, floor and sides for 1m outbye for easy detection of such cracks. Some indication of leakage may be available from smell. Increase of temperature behind the stopping indicates leakge of air into the sealed off area. In some cases there may be a hissing sound at the cracks if the leakage is heavy.

In a gassy mine, the construction of temporary stoppings stops air supply to the fire site but this results in gradual diminution of oxygen percentage from the air trapped behind the temporary stopping as the fire continues to burn. At the same time, the percentage of CH_4 issuing from the strata goes on increasing. These conditions create gas air mixture which is potentially explosive and there is possibility of gas explosion within 24-48 hours after sealing the area by temporary stoppings which are not explosion proof and are blown out by the explosion when one takes place. The construction of permanent stoppings to follow temporary stoppings is therefore, not recommended by some authorities who consider it advisable to construct temporary stoppings in gassy mines of robust construction and avoid the need for two sets of stoppings, temporary and permanent. The single set of stoppings is designed to combine the advantages of speedy construction and robustness of permanent stoppings.

Sealing off a fire in a gassy coal mine (Cat. II & III) :

The work of sealing a fire in gassy mines should be done by workers trained in the use of rescue apparatus and a rescue team with self-contained breathing apparatus should be kept in reserve.

At the stopping sites, selected after due consideration of the points stated earlier, robust stoppings, 4m thick, of sand bags are constructed. During the construction the affected area is heavily stone dusted. Sufficiently large area near the fire is enclosed by the stoppings so that a large volume of

air is trapped behind them. This ensures that when the stoppings are being constructed, there is enough air near the fire to prevent formation of explosive gas-air mixture and normal quantity of ventilating air should be allowed to reach the district by keeping the fan at the usual speed. During the construction of sand bag stoppings air supply to the fire area is maintained by providing two steel pipes, nearly 400 mm dia. in them. At the time of final sealing of the stoppings the two steel pipes (air tubes) are closed by pulling a previously prepared sand bag plug with the aid of thin wire rope. To prevent the sand bags from slipping and falling over C. G. I. sheets backed by steel rails are erected on the fire side of the stopping. Two holes, 400 mm dia. are left in the C. G. I. sheets for air tubes; in addition a hole 50 mm dia. is left for the sampling pipe. The sand bag stopping is reinforced on the outbye side by C. G. I. sheets propped up by vertical rails or props.

A sampling pipe with valve has to be provided on the robust sand bag stopping. Final sealing of the stoppings, one on the intake side and the other on the return side is effected simultaneously with the help of synchronised wrist watches. (Fig. 4.5)

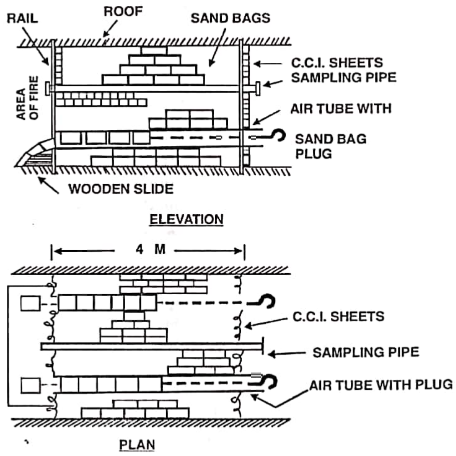


Fig. 4.5 Sand bag stopping

During the process of sealing, the atmosphere near the stopping should be tested frequently by CO detectors and flame safety lamps.

After the sealing all the workers engaged on the job should be withdrawn out of the mine. There is possibility of gas explosion within 24-48 hours of sealing and the mine conditions should be watched during the period. After the expiry of 48 hours air samples from behind the stoppings should be drawn and analysed on the surface. In deg. III gassy mines the stoppings should be reinforced on the outbye side with brick stoppings of explosion proof design equipped with the fittings of permanent stoppings.

Samples of atmosphere from behind the stoppings should be collected at the following intervals :

i. 48 hours after sealing ..... 8 hr. interval.

ii. When O_2 falls to 10% or low and methane rises to 16% or more ..... 24 hr. interval

iii. When O_2 falls to 5% or less and methane rises to 30% or more ...... 48 hr. interval or longer.

Air samples from behind the stoppings should be drawn during periods of low barrometric pressure, such as between the hours 12 noon and 2 P.M. (DGMS cir. 15 of 1961). The samples are collected in sampling bottles which are vacuum sealed or in other bottles by water displacement.

Breathing at fire stoppings :

In a sealed off fire area as the oxygen of the trapped mine air is consumed its pressure becomes less than the atmospheric air in the mine and fresh air leaks into the fire area. This is called "breathing". If the pressure inside the sealed off fire area is positive due to emission of firedamp from the strata it will cause foul gasses to leak out of the stoppings into the rest of the mine. Fluctuations of barometric pressure may also result in breathing. The leakage is usually small and cannot be prevented and it need not cause concern. If the stopping is whitewashed and the leakage is through small cracks in it, hair thin black lines of fine coal dust deposited on the cracks during leakage of air indicate the patches to be repaired.

In the case of a shallow mine the underground sealing is not effective if the fire area is connected to the surface through cracks. Depillared areas of the shallow mines are invariably so connected through surface cracks which have to be sealed by blanketing with inert material like earth or sand. The blanketing layer should be nearly 3m thick. Earth blanketing is not as effective as sand since the earth develops cracks. Sand blanketing of surface over the goaves of shallow coal mines is noticeable in many parts of Jharia coalfield where the inadequate attempts at sealing have not completely quenched the fires that have revived through surface breathing and have gone beyond control.

A novel method of sealing an underground fire from the surface was adopted successfully at Kurasia Colliery (M. P.) in 1961. The Colliery had underground workings in practically flat seam and the entrance was through adits. Part of the mine was worked by open cast mining with the help of heavy earth moving machinery. When a fire broke out in the underground mines (depth 15-60 m), it was decided to quench it from the surface. For this purpose the underground roadways were correctly correlated on the surface and 200 mm dia. boreholes, 600 mm apart, were drilled from the surface with the help of well hole drills to join with the underground galleries. Each gallery which was selected for construction of fire stopping, received nearly 21 bore holes 7 in each row across the width of the gallery (4.8 m). The number of holes depended upon the height of the gallery. Stone chips of + 10 mm – 60 size were dropped throught the holes to fill up the gallery upto the roof. Liquid cement mortar was then injected down the hole at a high pressure. The stone chips and the liquid cement mortar, when consolidated, resulted into a good seal against the fire.

Sealing the shaft :

If the shaft mouth has to be sealed to prevent entry of air into the mine on fire, the seal has to be built below the connection of the shaft with the surface fan drift. Holes are made in the shaft wall for placing steel rails. Over the steel rails are placed C. G. I. sheets and a layer of concrete about 1 m thick.

Flooding the mine with water or inter gas :

An underground fire may be quenched by pumping water into the mine through shaft or bore holes if a river or water tank is nearby and has sufficient water. The gas explosion at Chinkauri Colliery in 1958 had resulted into a fire which was quenched by pumping Damodar River water into the mine and not by underground methods of sealing. The method is very effective if attempt to seal the fire by underground stoppings is fraught with risk of gas explosion. If the explosion damages the headgear, the winding pulley and the winding arrangements, thereby making it impossible to go down the mine, the fire has to be tackled by flooding from the surface or by sealing the mouth of the shaft. Water cools down the underground strata faster than in the case of a fire quenched by fire stoppings and reopening of the sealed area need not wait for long. The method can however be adopted if the water can be retained in the mine and is resorted to only in the following cases :

1. Sufficiency of water near the mine.
2. Outbreak of fire on the dip side of the mine.
3. Non-existence of goaves on the dip side connected to other mines.

4. Non-existence of connecting galleries/drifts to other seams, or other mines.
5. Sufficient thickness of barrier against the mine on the dip side to withstand water pressure.
6. No possibility of dangers from land slides. In shallow mines, there is danger from the land slides if the mine is flooded.

Quenching a fire by flushing with nitrogen gas was practised in some mines in India, e.g. at Ramagundam, Kenda group, etc. For small requirements, the gas can be obtained in gas cylinders. A large demand requires use of tankers for transport of liquid nitrogen just like the familiar liquid oxygen tankers which are well insulated to reduce evaporation of gas during transport. In Jharia field such liquid nitrogen can be obtained from Sindri Fertiliser Plant. If the demand for liquid nitrogen is excessive, it is advisable to install nitrogen evaporating plant at the mine site. For example, at Fernhill colliery in U. K. where an underground fire sealed by fire stoppings could not be effectively controlled, a nitrogen evaporating plant was erected underground and the inert nitrogen gas was fed to the fire zone. That extinguished the fire. This was done nearly 3 decades ago.

In Jharia field fire near Londna and other fire areas was prevented from spreading further by installing near the fire area nitrogen producing machine. Such machine separated gaseous nitrogen from gaseous oxygen of the atmospheric air (in the gaseous form) and the nitrogen was fed to the fire area. As fire is near the surface, breathing of air to the fire area was hindering fire quenching operations and the machine could be used only to contain the fire.

In recent years (in 1989) fire in a goaf of a coal mine in north eastern France was quenched by a novel method which involved sealing of a roadway from a remote location through a specially drilled 85-m long borehole which was used for injection of expanding Mariflex foam resin. The method was adopted after injection of nitrogen gas proved ineffective. Mariflex has a swell factor of more than × 20.

In some mines pipelines for the injection of nitrogen are installed as a preparatory measure and they are linked to a surface supply system of nitrogen gas. Such was the case for the workings of Henri seam in Marienau field in North Eastern France where fire took place in 1989.

Reversal of ventilation after fire :

If a fire takes place in or near the DC shaft, smoke and fumes are carried by the ventilating current inbye to the working faces and endanger the men working at the faces. The smoke and fumes will soon travel to the main return and make the return airway unsafe as an escape route.

Reversal of ventilation in Degree-II and Degree-III gassy mines, is a major decision which only the mines manager should take after the occurrence of a fire or explosion. The reversal of ventilation may be advantageous if a fire is in the DC shaft or in its vicinity in the main intake airway. The reversal prevents the smoke and fumes from reaching the working faces where majority of the workers may be present. The reversal results in throwing open the separation doors between the DC and UC shafts and also other ventilation doors between the intake and return airways. This causes short circuiting of ventilation and steps should be taken to see that the separation doors and ventilation doors are kept closed. The nearest ventilation stopping inbye of the fire side should be broken to short circuit air after reversal. If the mine is ventilated by a forcing fan, the reversal results into exhaust ventilation and in Degree-II and Degree-III gassy mines, reversal may cause movement of large bodies of gas from goaves and old workings into the main intake airway and from there, over the fire with consequent risk of explosion. In a deep mine ventilated by an axial flow fan the latter may not be able to develop sufficient water gauge after reversal to counteract the N. V. P. in winter and reversal would not prove effective.

QUESTIONS

1. What are the different types of fire extinguishers ?
   Describe the fire extinguishers that can be used for Class 'C' fires.
2. What is meant by spontaneous heating ? What are the factors that contribute towards spontaneous heating in a coal mine ?
3. What preventive steps should be taken to guard against spontaenous heating in (a) overburden dumps in coal quarries, (b) coal stock on the surface ?
4. How is spontaneous heating detected in an underground coal mine ?
   What is the significance of CO/oxygen deficiency ratio in the air sample of a coal mine ?
5. What are the different types of temporary stoppings that can be constructed in a mine to deal with a fire ?

O O O

CHAPTER 5

EXPLOSIONS IN MINES

An explosion is a sudden combustion process of great intensity accompanied by release of large quantities of heat energy and in which the original gas or solid substance like coal dust is converted instantaneously into gaseous products. An explosion is invariably accompanied by violence on a large scale. Explosions in coal mines are of :

1. Firedamp,
2. Coal dust, and
3. Water gas

Firedamp has been the cause of most of the explosions in coal mines and in every mine steps are taken to prevent a firedamp explosion. Explosions of coal dust alone (in the absence of fire damp) have been comparatively rare though they have been initiated by firedamp explosions in a number of cases. Water gas explosions are rare and there is only one case of such explosion in our country.

- Presence of firedamp in air between 5.4 and 14.8% forms an explosive mixture. If a suitable source of ignition is available the mixture results in an explosion. An explosive mixture of gases is one which, when once ignited, will allow the flame to be self-propagated throughout the mixture, independent of and away from, the source of ignition. Such mixture is sometimes referred to as inflammable mixture but strictly speaking all the low mixtures of methane burn in air producing a bluish flame and they can therefore be termed inflammable. The maximum explosive violence is produced when the explosive mixture contains about 9% of firedamp. Laboratory experiments have proved that by suddenly compressing the gas-air mixture, mixtures containing as low as 2% and as high as 75% firedamp, can be ignited and made explosive as both the temperature and pressure are raised simultaneously. The lower limit of explosibility with which a miner is primarily concerned, remains practically unaffected by presence of black damp as may normally be found in mines. If the percentage of black damp exceeds about 35, the atmosphere becomes non-explosive irrespective of any percentage of fire damp.

The lower limit (x%) of a mixture of methane and other hydrocarbons liberated underground with normal air, for usual mine temeratures, and pressures can be calculated from the formula of Le Chatelier :

$$x = \left[\frac{100}{\frac{P_1}{N_1}+\frac{P_2}{N_2}+\frac{P_3}{N_3}+\ldots\ldots\ldots\ldots}\right] \text{percent}$$

Where P_1, P_2, P_3.. = contents, in per cent by volume, of each combustible component of the mixture

$(P_1+P_2+P_3+\ldots\ldots\ldots = 100\%)$

N_1, N_2, N_3 -- = lower explosion limits of each component.

Thus, for example, if in the mine the gas emitted is a mixture of 80 % methane, 10% hydrogen, 10% ethane, then the lower limit of explosibility of this gas mixture will be :

$$x = \frac{100}{\frac{80}{5}+\frac{10}{4.1}+\frac{10}{3.2}} = 4.6\% \text{ approx}$$

The presence in the air of fine dry coal dust lowers the lower limit of explosibility of the methane-air mixture to well below the usual 5%.

Methane explosions are always accompained by two shock waves, the *forward shock wave* caused by the heated gaseous products of the explotion under high pressure creating an air wave of a considerable force travelling rapidly away from the point of explosion, and the second or *reverse wave* which results from the pressure drop at the point of explosion because of the cooling of gases and the condensation of the water vapour (occupying a considerable part of the volume of the explosion products). The reverse wave is of a rather smaller force than the primary wave, but it follows the course just traversed by the first wave. The forward shock wave precedes the flames of explosion which consumes all the oxygen of the air.

Coward's diagram :

If fire damp present in an area is below 5.4%, it will burn away when flame is applied to it. The gas is then **combustible.** If, however, gas is present in air above 5.4% application of flame or source of heat of sufficient intensity will cause the mixture to explode. Fig. 5.1 shows the limits of explosibility with diffferent percentages of firdamp and oxygen. It has significance when

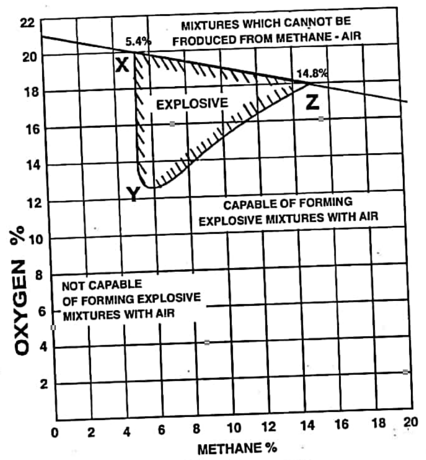


Fig. 5.1 Coward's diagram

sealing off a fire in a gassy mine and also when reopening a sealed off a fire in a gassy mine and also when reopening a sealed off area. The important points to note are :

1. All mixtures lying within the traingular area **XYZ**, are in themselves explosive.
2. All mixtures lying to the right of **PYZ** contain too much methane to explode but they will form explosive mixtures when mixed with the right amount of air.
3. All mixtures lying to the left of **PYX** are neither explosive nor capable of forming explosive mixtures with air.
4. Lower limit of explosibility remains almost constant at about 5.4% for all percentages of oxygen down to about 12.5%.
5. The higher limit of explosibility gradually decreases from 14.8% to about 6 % with decreasing percentage of oxygen.
6. No percentage of fire damp is explosive when the percentage of oxygen is 12 or less.

7. A firedamp-air mixture may become explosive when diluted with an appropriate quantity of air which brings the new mixtures within the limits of the triangle **XYZ.**

Ignition Temprature :

The ignition temperature of methane is the lowest temperature to which methane-air mixture must be raised in order to set the mixture aflame. It is usually taken as 650 to 750° C. However, the most recent investigations have shown that it is not one fixed temperature and depends upon the intensity of heat of the igniter, the methane content of the air, the presence of various impurities in methane and the pressure under which the gas-mixture is confined.

The most characteristic property of methane is that when in contact with a hot source, it ignites, not immediately but with a delay, that is, after an interval of time which depends on the temperature of the hot source; at 650° C the delay is ten seconds, at 1000°C it falls to one second . It is considered that methane begins to burn only after it has absorbed an appropriate amount of heat (22.1 kcal /mole), and this is the reason for lag on ignition of methane. the presence of hydrogen and other combustible gases in methane reduces the delay on ignition, *e.g.,* the presence of 30% hydrogen in methane eliminates the delay completely.

The lag on ignition to firdamp and the actual moment of ignition is the time interval that clapeses between application of a source of ignition to firdamp and the actual moment of ignition of gas. It is usually a few seconds and in some cases, a small fraction of a second. The lag on ignition has an Important bearing on the design of safety appliances in gassy mines as will be explained later.

A Firedamp explosion can be prevented by :

1. Avoiding dangerous accumulation of firedamp , much below the lower limit of explosibility.
2. Avoiding sources of ignition which may cause the firedamp accumulation to explode.

Proper ventilation of the mine is the correct method to prevent dangerous build-up of firedamp. Regular inspection of places where firedamp may accumulate is essential in addition to proper ventliation. A place is not considered to be adequately ventilated if the firedamp percentage exceeds 1.25 and a mine is considered to be inadequately ventilated if the firedamp percentage in the general body of the return air exceeds 0.75. In a mine.

(deg. II. and deg. III) where electricity is used underground, if the firedamp percentage in the return air of a ventilating district and steps taken daily in the ventilating district exceeds 0.8, air samples have to be taken daily in the ventilating district and steps taken to bring down the gas percentage till it falls to less than 0.8.

If in the ventilating district of deg. II and deg. IIImines, percentage of CH_4 exceeds 1.25 at any time electric supply to that district should be cut off and steps taken to bring down the percentage of gas.

When approaching a dyke, fault or other geological disturbance, care should be taken to guard against gas and the Mining Regulation requires that "no working or gallery in a gassy seam of second of third degree; or in a gassy seam of first degree which has approached within 30 m of known dyke , fault or other geological disturbance, shall be extended to a distance of 4.5 m from the nearest ventilation connection unless the air is coursed upto a point within 4.5 m from the face".

The second step to prevent firedamp explosion is to avoid sources of ignition of firedamp. The sources which can possibly result in ignition of gas in an underground mine have already been stated in the earlier chapter. If electricity is used the motors, switchgears and transformers should be provided with flamproof enclosures.

A flamproof enclosure is used for electrical apparatus at higher voltage exceeding 25 volts. A flamproof enclosure is not gas tight, but it is so designed and constructed that during its normal operation, even if a gas explosion takes place inside the apparatus, the cover withstands such explosion and the hot gases coming out from the rough-machined flanges of the cover are sufficiently cooled so that they do not ignite gas-air mixture outside the apparatus because of lag on ignition. An intrinsically safe apparatus is one which is so constructed that during its use, the spark produced by it is not of such high temperature as to cause ignition of gas. This construction is possible with apparatus operated by voltage upto 25 volts. Signalling bells, te-lephones, exploders and relays are of intrinsically safe construction.

A flamproof enclosure ceases to retain its flameproof character if not assembled correctly after repaires or inspections. Sometimes some of the bolt-holes are not fitted with the bolts and this is a common omission.

If the gas emission is very high, as may sometimes occur in very deep mines, it is a safe practice to substitute electric power by compressed air, or to arrange for methane drainage. This has already been elaborated in the earlier chapters.

COAL DUST EXPLOSIONS AND THEIR PREVENTION :

It has been established by experiments and on the basis of studies of a number of explosions that coal dust, when suspended in the air as a cloud, is capable of bursting into an explosion and propagate it, even in the absence of firedamp . It should be noted that for a coal dust to start an explotion the dust should be in the form of cloud in the air, so dense that one cannot see through it. The quantity amounts to 30 to 40 g/m³ of space . The cloud may not be of a large size but its density is important and a source of ignition of sufficient intensity , as one can well understand, should be present. Once the coal dust explosion starts, its propagation needs very small quantity of the dust , only I gram per c. c. of space which is equal to the thickness of a layer of ordinary thick paper over the periphery of 2.4 m × 2.7m roadway. This quantity is so small that no part of a coal mine free from when one occurs as it is impossible to keep a mine free from when one occurs as it is impossible to keep a mine free from when one occurs as it is impossible to keep a mine free from such small accumulations. For this reason when a coal dust explosion takes place it travels to practically all parts of the mine.

For a coal dust explosion to take place the dust must first be raised into air in the form of a cloud, as already stated, and then ignited by a source of heat of sufficient intensity. Such circumstances generally exist after a firedamp explosion. This however , does not imply that all gas explosions are followed by coal dust explosions. The two gas explosions in recent years in India, one at Jeetpur in 1973 and the other at Sudamdih in 1976, were gas explosions.

The lowest temperature at which a fine dry coal dust cloud can be ignited and can cause the flame to travel throughout the dust-air mixture is 700 to 800 ° C. It has been observed from the research conducted in the experimental galleries abroad and also at C.M.R. S . ,Dhanbad that when coal dust propagates an explosion, flame front , as in the case of gas explosion and this important observation is the basis of such precautions as stone dusting , stone dust barries and water barries against coal dust explosions. (In the case of fire damp explosions also the blast or pressure wave travels ahead of the explosion flame front.)

The *inflammability of a coal dust may* be defined as its ability to cause a flame to spread away from the source of ignition. Some coal dusts are more inflammable than others . The lower limit of inflammability of coal dust is 1 gram per c . c . but the higher limit is quite high and for bituminous coals it is above 2000 g / m³ of space. This higher limit represents a very thick cloud which is difficult to exist in a mine under normal mining conditions. The inflammability of a coal dust is dependent upon the following factors :

1. *Percentage of volatile matter,*
2. *Fineness of particles,*
3. *Percentage of inert or incombustible matter,*
4. *Peresence of moisture,*
5. *Presesnce of fire damp,*
6. *Nature and intensity of ignition source,*
7. *Age of the dust,*
8. *Condition of dust distribution.*

1. *Percentage of volatile matter :* The inflammability increases with the V. M. content. It is observed that coal containing less than 13% of V. M. calculated on dry ash-free basis, is not likely to propagate flame.
2. Fineness of coal dust particles : Finer the coal dust, greater is its inflammability . The finest dust is most dangerous. The fineness of mine dust is defined by its content of - 80 fraction, particles below 1/10 to 1/15 mm in size.
3. *Percentage of incombustible matter :* The incombustible matter in coal dust is moisture and the inherent ash.In a mixture of a coal and rock dust in a mine the incombusitible matter absorbs some of the heat of the igniting source so that the temerature of the combustible portion does not reach the igniting temperature and flame cannot be propagated. this is the principle behind "stone dusting" in coal mines.
4. *Presence of moisture :* External moisture added to coal dust reduces its inflammability and if the moisture is in sufficient quantity it binds the dust particles together therby preventing them from rising in the air as a cloud. The quantity of moisture that would ensure non propagation of flame by coal dust is however high, atleast 1/30th of the total mixture.The dust in contact with so much moisture would be in the form of a thin paste which is rarely present in a mine, except at a few local places.
5. *Presence of fire damp :* The inflammability of coal dust increases almost directly in proportion to the percentage of firedamp present in the atmosphere. For every 1 % firedamp 10 to 14 % additional stone dust is required for efficient stone dusting.

6. *Nature and intensity of ignition source :* These factors influence the inflammability of coal dust. Explosions initiated by high temperature sources develop the former faster and cause more damage.
7. *Age of coal dust :* Weathered coal dusts are more inflammable as they contain oxygen loosely combined with the coal substance.
8. The varying conditions of dust distribution and propagation of the explosion also affect the course of dust explosions. The least dangerous condition of the dust generally is on the floor and sides; the most dangerous position is on the roof and on the bars of the timber props.

Prevetion of coal dust explosions :

A coal dust explosion can be prevented by :

1. Reducing the formation of coal dust at the working faces, haulage roads and elsewhere.
2. Preventing its spread.
3. Rendering coal dust harmless by wetting it with water or mixing with inert stone dust.
4. Provision of stone dust barries or water barries.

Formation of coal dust can be reduced in the following ways.

1. At the face :

i. By water infusion : At the longwall face holes are drilled at an angle of about 45° to the face and water under high pressure is injected into them till a thin film of water is visible at the coal face. Water infusion has the further advantage of rendering the coal easier for ploughing and sharing. Pulsed infusion shot-firing at longwall faces and also in board and pillar working has the advantage of reducing dust formation.
ii. By use of water sprays on the coal cutting machine picks and shearer picks.
iii. By the use of sharp picks on the coal cutting machines and the shearers. Blunt picks produce more coal dust. Use of gummer on the coal cutting machine helps in collecting the dust at one place and prevents its dissemination in air at the face.
iv. Giving cut by coal cutting machine in a soft stone band or shale band if one is present at the coal face.
v. By selecting right type of explosives and by proper control of shot–firing so that the blasting operations produce less dust. Alternatives to expiosives like Armstrong air breaker, Cardox , Hydrox , ets. which produce more lumpy coal can also be tried.

2. During transport of coal :

i. Coal tubs and mine cars should be spillage- proof.

ii. Haulage track should be well laid to prevent derailments.

iii. Belt conveyors should be properly aligned and so installed as to avoid spillage.The fall of coal from the conveyor to the mine cars or tubs should be minimum. Much fine dust is deposited below the conveyor rollers and in particular below the driving and return drums where the dust generally collects in the form of heaps. Such accumulations of coal dust should be removed at intervals.

Water spraying at loading points, transfer points and over the loaded coal tubs helps in reducing the dissemination of coal dust. Low air velocities of the ventilating air current at the face and on the haulage roads is a further contributory factor for reducig coal dust in the air.

Dust at transfer points may be collected by the use of dust extractors. A dust extractor essentially consists of (a) a hood or canopy over the loading point (b) air tube to carry the dust laden air to a collecting vessel (c) compressed air jets or fans to produce suction effect.

The dust which is produced should be collected at intervals and removed out of the mine in totally closed containers .

iv. Loading of skips should be arranged in the upcast shaft.

v. Surface screening plant and tipplers should be atleast 50 m away from the intake shaft. Use of water sprays at the tipplers is recommended.

The dust remaining in the mine even after the above precautions are taken is sufficient to propagate a coal dust explosion unless the mine wet. A mine may be considered to be naturally wet if the dust from the mine, when squeezed in the first of the hand, exudes slight quantity of water.

The steps taken to render the coal dust harmless are :

1. Wetting of the coal dust with water,
2. Spraying or sprinkling stone dust,
3. Provision of stone dust barries or water barries.

For wetting the coal dust sprays of water on the roof, sides and floor are sometimes used. But the dust is difficult to wet, tends to floor are sometimes used. But the dust is difficult to wet, air current. In some mines, Sufficient water may not be available. Moreover, introduction of water in the mine has the disadvantage of increasing the humidity. For these reasons water may be used only for local application, e. g. at transfer points, or on loaded coal tubs. Addition of some chemicals keeps the dust wet for longer periods. The chemicals recommended are : C_2Cl_2 , "Calso-lene Oil " manufactured by M/s Imperial Chemical Industries (India) , "Shell non-ioic detergent $_p$–47-C "

"Coalest X 5" and "Coalset " manufactured by M/s Daiichi Karkuria Pvt. Ltd. The optimum concentration for "Calsolene Oil " in wetting dust of low moisture as well as high moisture Jharia and Raniganj field is o.16 % by volume in aqueous solution of Calsolene Oil = 242 kg. i. e. 151.25 kg. dust per litre of calsolen oil supplied. (DGMS Circular No. 31 of 1966).

The salient features of Coal Mines Regulations for measures against coal dust are as follows :

Measures against dust (CMR 123) :

i. Chain of a coal cutting machine will always be equipped with complete set of picks.

ii. Mechanical coal cutter chain will be sprayed with water jets during cutting.

iii. For power operated drills dust trap should be used or wet cutting should be adopted.

iv. At every working face and roadway within 60 m of the face water should be stone dusted at intervals so that dust on the floor , roof and sides has at least 30 % water at all times ; or

v. Floor, roof and sides of workings, if not naturally wet, should be stone dusted at intervals so that the dust samples at those places have at least 75% incombustible matter in them (if coal has less than 30 % V . M. on dmmf basis) or 85 % incombustible matter in them (if coal has over 30 % V. M. on dmmf basis) or , the places should be sprayed with water at intervals as stated in (iv).

vi. Stone dust used for dusting should not contain more than 5 % of free silica.

vii. Specifications of stone dust should be tested once in 3 months.

viii. Where dust cannot be suppressed to safe limits the D. G. M. S. may require that every worker exposed to such dust shall be provided with a suitable respirator.

CMR. 123 (A) :

i. A dust plan on a scale of 1 : 2400 or larger scale should be maintained showing areas which are naturally wet, water pipe line for spraying as a dust control measure, and areas to be stone dusted at intervals of 24 hrs, 7 days, 30 days or 3 months. The areas shall be marked by separate colours or codes.

ii. Underground, the areas to be treated as required in (i) should be demarcated by notice boards.

iii. The dust in charge should maintain a daily record of steps taken for dust control. He will also see that before shot firing the area in the veicinity is sprayed with water.

CMR. 123 (B)

i. Every return airway lying within 200 m of last working face and every haulage, tramming or conveyor roadway (if not naturally wet) should be divided into dust sampling zones of 150 m length (or smaller). Each zone should be divided into 3 sections a, b c. The zones and sections should be marked on a sampling plan (1 : 2400 r.f. or larger scale) and they should also be marked underground by notice boards.

ii. Representative samples should be collected from every zone and section once a month.

iii. In a water sprayed zone samples should be collected by method of "strip sampling" (10 cm wide strips , 5 m apart).

iv. In a stone dusted zone samples should be collected by method of spot sampling.

v. Sampling in charge should be mining diploma holder (minimum qualification)

Application of stone dust :

The stone dust mixed with coal dust, as already stated , has the effect of absorbing the heat that would otherwise ignite the coal dust cloud and therefore the stone dust prevents coals dust from reaching the ignition point. The most generally accepted *index of explosibility* of a coal dust at present is the amount of stone dust which must be added to it to make it non-explosive. The index can be expressed in two ways : either directly by the weight of stone dust added, in kg per kg of dust being tested, or by the percentage of ash content in the mixture after the stone dust has been added, including the natural ash content of the dust being tested. The desirable qualities of stone dust are :

1 Easily dispersible to form a cloud in the air when disturbed.
2. Easily available in large quantities and easily grindable.
3. Not injurious to health. Siliceous dust should be avoided.
4. Preferably white in colour.

Dust of shale, limestone or gypsum is generally recommended for use. Sometimes boiler ash, finely ground, is also used but it contains siliceous matter and moreover it is not available in large quantities. It was believed at one time that limestone dust is about twice as efficient as shale dust, but this

belief has been found to be incorrect. The percentage of incombustible dust in the coal/ stone dust sample in a mine should be 50 to 60 % for the stone dust to be effective in preventing propagation of flame. The stone dust should be fine, generally 100 mesh and finer. In wet or damp conditions, water proofed dust must be used. Stone dusting of underground roadways should be carried out at regular intervals so that the top layer is of stone dust. Mechanical distributors of stone dusts are recommended, e. g. Mist Sprayer (MISTER) manufactured by Mine Machinery & Spares, Dhanbad (D.G.M.S. Circular No. 68 of 1970)

Stone dusting and wetting of coal dust by water cannot go together. Stone dusting is effective if the dust is readily raised in air in the form of a cloud but wetting of coal dust by water aims at achieving the opposite result, viz., consolidating the dust so that it is difficult to rise in the air.

Samples of coal dust in the mine treated with stone dust should be taken at regular intervals, once a week, and analysed to know the content of stone dust.

The D.G.M.S. Circular has prescribed the following *maximum* velocities of ventilating air in a coal mine with the object of reducing the drying effect of air current and preventing escape of moisture from the coal dust.

Locality	*Maximum Velocity*
Man-hoisting shafts and haulage roads (other than conveyor roads)	8m /sec
Other roadways.	6m /sec
Conveyor roads, loading points and transfer point.	4m /sec
Working faces in developing in depillaring/stoping	4m/sec.

Stone dust barriers :

A stone dust barrier consists of shelves placed side by side and each shelf consists of planks placed on above the other and loaded with stone dust. These are placed on supports in the main roadways of an underground mine in such a manner that the planks collapse with the shock of an explosion, thereby causing the stone dust to disperse in air and form a thick stone dust cloud in the path of the oncoming explosion flame. The stone dust cloud smothers the flame and prevents ignition and explosion of the coal dust. Stone dust barries have to be provided in addition to the normal measures like coal dust wetting or stone dusting adopted in a coal mine to counter the danger of coal dust.

A stone dust barrier may fail to arrest an explosion if there is a layer of accumulation of fire damp in the roof of the roadways over the barrier or at a short distance inbye of it since such a layer can provide a bypass for the same. It may also fail if located within 40–60m from the face or other potential source of ignition.

There are a few design of stone dust barries adopted in European countries. The D.G.M.S. Circulars recommed adoption of Polish type of stone dust barries. In this type of barrier the stone dust rests on planks which run longitudinally in the road-way and whose length equals the width of the shelves. These planks rest on a rigid frame, the two main members of which are at least 150mm in depth and rest on their edges on two fixed rigid brackets. Neither the frame, nor the planks are fastened either to each other or to the fixed brackets (Fig. 5.2).

According to the loading of the shelves and the total quantity of stone dust on the shelves, the barries may either be as :

i. Light, also called Primary or First Barrier; or

ii. Heavy, also called Secondary or Second Barrier.

Light type of barriers are intended for use nearest to a possible point of ignition. They consist of light loaded shelves not more than 35 cms in width. Heavy type of barries are intended for use further from the possible site of explosion. They, therefore, contain more dust because the greater distance will give the explosion the opportunity to develop greater violence which will then be difficult to stop. The amount of dust in a heavy barrier should, therefore, be adequate to stop such an explosion.

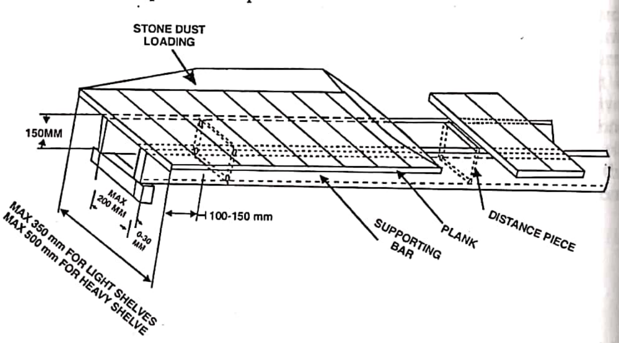


Fig. 5.2 Shelf of a stone dust barrier of Polish type.

To ensure its successful operation a barrier must be provided with shelves which collapse in the event of an explosion.

For a light barrier, a roadway 3m × 3m cross-section would need minimum 11 shelves, each with 100 kg of stone dust but a roadway of the same size, if equipped with a heavy barrier, needs 16 shelves, each with 200kg dust, plus 8 lightly loaded shelves, each with 100 kg of dust. The lightly loaded shelves are placed adjacent to one another and are placed at the inbye end of the barrier. 1/3rd of the shelves comprising a heavy barrier should be lightly loaded shelves similar to those for the light barriers and the remainder, heavily loaded shelves.

Location of stone dust barries :

The stone dust barries should be sited as near as possible to the potential source of ignition. In the bord and pillar workings it is sufficient to provide a single barrier of the heavy type sited at a sufficient distance from the face to ensure that there is no likehood of its being passed by the flame from a fire-damp explosion originating at the face. Such a barrier should be provided at a distance of not less than 150m from the nearest working face and at not more than 400m from the farthest face. The barriers shall be kept advanced to their new positions and when necessary. To be effective, the heavy barrier will have to be provided in all the entries to the district. Since the bord and pillar working there are likely to be a number of entires to a district, the places where the barriers are to be sited may be reduced by building explosion proof stoppings consisting of 1.8m thick brick in cement in the entries other than those essentially required for ventilation and haulage. etc. The shelves of the barrier would ordinarily be included in about one pillar length and as far as possible, the shelves should not be positioned at a junction.

In the longwall workings a barrier of light type should be installed in all longwall gate conveyor roads within the range of 50-120 m from the nearest point of the face. A second barrier (heavy type) should be placed further outbye at 200-350 m from the face.

Near Shaft Insets :

Where more than one scam are worked from the same shafts, heavy type of barries should be sited in the roads adjacent to the shaft landings at a distance of 100m to 150 m from the landing. These barriers should, as far as possible, be so arranged that they are in the middle of a straight stretch of road at least 200 m in length.

Maintenance of Barriers :

The proper maintenance of the barries is of first importance. It should be the responsibility of a specially appointed component person like Ventilation officer to examine and maintain the barries in satisfactory condition For this purpose, the duties of the competent person should include :

i. examining once in every week all stone dust barriers ;

ii. as a part of this examination , testing the dispersibility of the dust by taking some inhand and blowing on it. If this shows a tendancy to cake or consolidate, the dust in the barrier should be removed ;

iii. arranging for the repair of any damage to the shelves or other parts of th barrier ;

iv. supervising the erection of new barriers as required;

v. to write a report on such inspections and on any action taken or required. The reports should be countersigned by the manager or under manager in the book maintained for such reports, there shall also be recorded all data concerning position, quantity of stone dust, cross-section of the road in which the stone dust barrier is situated, date of inspection and renewal of stone dust, etc.

Any defect in the stone dust barrier should be removed forthwith. If this is not possible, shotfiring should be stopped in the district.

It is important that the barriers are moved at the necessary intervals to ensure that they are maintained constantly with the recommended ranges of distances from the face .

A board should be provided near each barrier on which the following information is recorded :

1. Cross-section of the roadway.
2. Total dust loading of the barrier.
3. Number and loading of shelves.
4. Date of last removal of the stone dust.
5. Reference number of the barrier.
6. Date of last inspection.
7. Signature of the competent person appointed specially for the maintenance of the barriers.

Plan :

On the rescue plan, and the stone dusting plan should also be shown the :

i. position and type of barriers;

ii. frequency and rate of its advance;

iii. Serial number of the barrier.

These plans should be brought upto date not less than once in three month.

Copies of the stone dusting plan should be provided to the official (s) responsible for examining and maintaining the barriers.

Action by underground supervisory staff after an explosion :

The action to be taken by the senior supervisory staff like over man, assistant manager and others after an explotion may well be illustrated by the following question and its answer.

Question : You are with a number of workmen in a district of a mine and hear a very loud sound uncommon in the normal working of the mine. Assuming that an explotion has taken place, what steps will you take for the safety of your workers?

Answer : An exact procedure to be followed in such a situation is difficult to state as it will depend on the possible site of the explosion, the distance of my district from that site, the lay- out of the mine. the nature of ventilation system and the extent of explosion. Telephonic communications are invaribly disrupted after an explotin and I cannot expect guidance from others.

Normally a gas explosion is sometimes followed by a second one and keeping this in mind my line of action will be as follows :

1. Send word to all workers in by to come to my district.
2. Count all the workers to see if any person is missing or still in
3. Explain to the workers, the gravity of the situation and the need to keep cool without losing heart. To keep up their morale, I would quote examples of previous gas explosions in other mines where men had survived.
4. Assuming that my district is ventilated by an independent split which is not affected by the products of explosions such as smoke, fumes, afterdamp, coal dust-laden air current, I shall decide to escap by the intake air route. A level headed strong man should form the rear guard and all the men will travel together.
5. If enroute, there are signs of violence resulting in roof fall, jamming of the route with tubs, broken steel arches etc., I shall take another route to reach an unaffected seam or another district away from the possible site of explosion. The main idea is to avoid any route where afterdamp may be present and for this reason a return airway will be avoided.
6. If my district happens to be on the return side of the explosion smoke, fumes, coal dust-laden air will enter my district with the ventilating air and I shall then have to escap via the return airway. Before escaping, the ventilation will be stopped by erecting brattice cloth to arrest travel of on coming air mixed with afterdamp, smoke, fumes, dust,etc.

7. If the workers are provided with self rescuers, I shall ask them to d the self rescuers before passing through a zone containing afterdam If self rescuers are not provided and the return route is also affect by afterdamp and roof falls, the only course for me will be to retr inbye and stay in the air which is still unpolluted by afterdam The air will be conserved by erection of brattice stoppings and l short-circuting the ventilation. If compressed air pipes are availab in the district, I would turn on the compressed air assuming the surfa compressors to be still running. To conserve the cap-lamp curren practically all the lamps are provided they will be dimmed. The a trapped in the district behind brattic stoppings will last a long tim and perhaps for a few days.

   The party then has to wait for the arrival of the rescue teams.

Steps to be taken by the mines manager after an explosion :

An explosion is a situation which unnerves many an experience Manager, much less to speak of young inexperienced Managers. The Mi Manager should therfore take major decisions in consulation with senior offic of the mine and other senior officers of adjecent mines who may be availab at the site soon after the explosion. The Manager should first find out th districts or mines that are affected, the condition of the winding arrangeme and the surface fan. He should get first hand information from workers w might have come to the surface from the affected mine and also from oth present at the surface at the time of explosion. Electric supply to th underground part of the affected mine should be immediately cut off and activities directed to rescue of the possible survivors in the mine. That shou be the first object. The Mine Manager should set in motion all the pre-plann organisation to deal with emergencies so that responsible workers and offic are on their jobs. Telephonic information should be sent to the followi authorities.

1. Joint Director of Mines Safety.
2. Director-General of Mines safety.
3. Rescue station of the area.
4. Police Station.
5. District Magistrate and sub Divisonal Officer of the Distr administration.
6. Administrative headquarter of the company owning the mine.
7. Adjacent mines for assistance of men and materials.
8. Regional Coal Mines Welfare Hospital and other hospitals of the ar
9. Trade Union leaders.

The Manager should arrange to get together all rescue trained workers of his mine and other mines in the area. The attendance clerk and lamp cabin attendant should be directed to keep a strict count of the persons going down the mine or leaving it. All the orders given by the Manager and others should be recorded as far as practicable. A control room should be set up at the surface and placed in charge of a responsible officer. During the time taken for all these activities it is likely that the Mines Safety Directors, rescue brigades, senior officers of adjacent mines and senior officers from company headquarters arrive at the spot. The Manager should then form an advisory body consisting of senior officers who might have dealt with situations of similar nature. The execution of the work will rest with Mine Manager. Assistant Manager and other officers of the mine. The actual steps that the Mine Manager has to take will vary from situation to situation.

QUESTIONS

1. What steps should be taken in a mine to prevent explosion of fire damp ?
2. Discuss the factors that affect the explosibility of coal dust in a mine.
3. What are the descriable qualities of a stone dust ? A coal dust explosion has taken place in a mine in spit of regular stone dusting. Discuss the possible causes.
4. What is a flame-proof apparatus and an intrinsically safe appratus ? How does a flame-proof apparatus cease to retain its flame-proof character ?
5. What are stone dust barriers ? Give the specifications of a light barrier and heavy barrier as recommended by the D.G.M.S. in his circulars .
6. What procedure should an overman adopt when he is on a routin inspection of his district and comes to know of an explosion in some other district ?

O O O

CHAPTER 6

RESCUE APPARATUS AND RESCUE OPERATIONS

Mines rescue apparatus primarily provides protection to human breathing system against poisonus and irrespirable gases and the term also includes reviving apparatus., escap apparatus, self rescuers, gas masks and the self contained breathing apparatus. with compressed oxygen or liquid oxygen.

Before dealing with rescue apparatus it is essential to understand the human respiration system.

When a person inhales air or oxygen, the air or oxygen is drawn into the lungs down the windpipe or trachea whish divides into a number of bronchial tubes. These tubes end in multitudes of minute air-sacs or alveoli which form the ultimate structure of the lungs. The alveoli are lined with a thin membarance over which the circulating blood is spread in a microscopic network of a myriad capillaries. The circulating blood requires a continuous intake of oxygen which is provided by human breathing. In addition the lungs require an intermittent intake of oxygen, but in regular way and this is also supplied during inhalation. If the supply of oxygen to the blood stream or to the lungs is stopped for a few seconds it results in death. A person inhales the atmospheric air (21 % O_2 79 %, N_2, nearly), and only 3 to 5% of oxygen is party used up and partly converted to CO_2 but the nitrogen remains unchanged so that exhaled air has the composition :

Oxygen .... 18 to 16 %

CO_2 .... 3 to 5 %

It is proportional to the O_2 used.

Nitrogen 79%

A person doing normal work (not heavy work) breathes at the rate of nearly 15 breaths per minute and the number increases to 20 or more during heavy work. During each inhalation he inspires about 2 litres of air of which he consumes about 100 c. c. of oxygen so that with 15 breaths per min. he

consumes 1.5 liters of oxygen per minute. The following table gives the consumption of air and oxygen per minute. The following table gives the consumption of air and oxygen per minute for different degrees of exertion.

Activity	air inhaled lit./min	O_2 consumption lit./min.
Resting	8–10	0.3 — 0.4
Walking with load	15-20	0.6 — 0.9
Normal work	30—40	1.3 — 1.8
Hard work with pauses ...	40–50	1.8 – 2.3
Very hard work for short duration	60— 90	2.7 —4.0

Self contained breathing apparatus :

Principle of action : Consider a volume of 20 litres of exhaled air containing 17 % oxygen. The exhaled air would comprise 3.4 litres of O_2, 15.8 litres of N_2 and 0.8 litres of Co_2 can be removed and an equal quantity of oxygen supplied instead to the residual air, the resultant gaseous mixture would again have the normal composition of the atmosphere. A self contained breathing apparatus functions on these lines and it so constructed that the wearer carries with him all the means for respiration in an irrespirable atmosphere, on any other person. The wearer using the apparatus can move and work in an atmosphere consisting of poisonous and noxious gases, whatever their percentage. The normal period for which the apparatus provides protection varies from one hour to four hours, depending upon size.

A self contained breathing apparatus is of the closed circuit type in that oxygen supplied to the wearer from a cylinder is not lost to the atmosphere during exhalation but is reused by him after the exhaled air is freed of the CO_2 by chemical absorbents. As the unused oxygen of the exhaled breath is not sufficient for the wearer it is replenished from a cylinder containing oxygen in the compressed or liquified form. A closed circuit type apparatus in which the oxygen of the exhaled breath is reused is also called *regenerative* aparatus. The self contained breathing apparatus used in our mines are two well-known makes : (a) Proto self contained compressed oxygen apparatus, and (b) Drager self-contained compressed oxygen apparatus.

Proto Mark IV and Mark V apparatus are the major equipments in most of our rescue stations though they are no longer manufactured. Both the apparatus look alike and work on the same principle.

Proto Mark IV apparatus :

It consists of the following :

(a) A light alloy cylinder of 2 litre (empty) capicity, containg 300 lit. of oxygen compressed to 150 kg /cm². It is fitted with main valve, a pressure gauge valve, a by--pass valve, a reducing valve. The main valve is the cylinder closing/opening valve which is kept open by a locking device when the apparatus is in use. The reducing valve reduces the pressure of oxygen supplied to the wearer and ensures 2 lit. of oxygen per minute. The by-pass valve is manually operated by the wearer if the reducing valve fails or when the wearer needs more oxygen than that supplied by the reducing valve (for example, when doing heavy work). The pressure gauge valve admits high pressure oxygen to the pressure gauge.

(b) Breathing bag made of vulcanised rubber and divided into two compartments by a partion except at the bottom end. The bag contains 2 kg of CO_2 absorbent known as *protosorb*. It is a mixture of calcium hydroxide and coustic soda and it keeps the percentage of CO_2 in the breathing circuit below 2 %. The wearer breathes from and into the bag which serves as an air reservoir.

(c) A cooling chamber of copper containing sodium phosphate which is in crystal form at ordinary temperatures but liquefies at 35°C absorbing much heat in the process.

(d) Inhalation valve, exhalation valve and relief valve.The relief valve allows the escape of any oxygen in excess of the wearer's requirement.

(e) Nose clip, mouthpiece, inhalation and exhalation tubes. Weight of sodium phosphate is 170g.

The apparatus is designed for a 2-hours use. Breathing is through the mouth and the nasal passages are closed by a special nose clip. After donning the apparatus the wearer has to take a few breaths of pure oxygen and flush out the nitrogen from his respiratory system. In actual use the exhaled air passes over the protosorb which removes the CO_2. On inhalation tube to the lungs. On exhalation the air passes through the exhalation valve to the breathing bag for further regeneration.

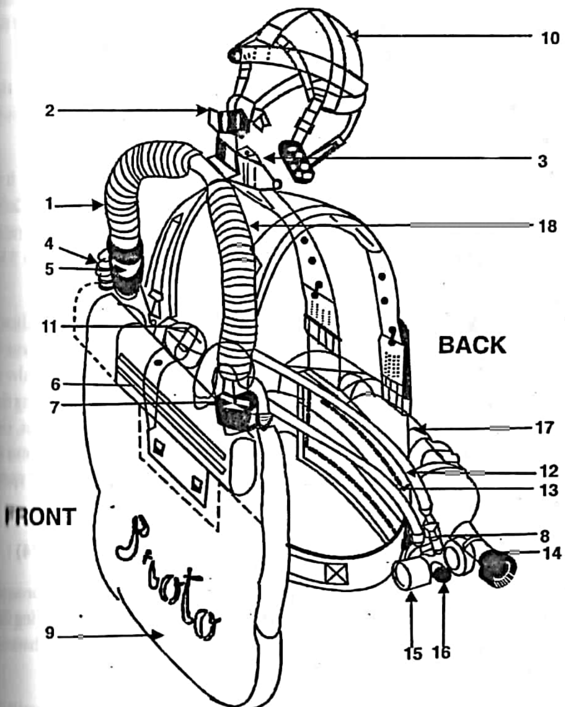


Fig 6.1 Proto mark V self-cntained compressed-oxygen breathing apparatus.

1-exhalation tube, 2-nose clip, 3-mouthpiece, 4-relief valve, 5-exhalation valve, 6-cooler, 7-inhalation valve, 8-pressure gauge valve, 9-breathing bag with inhaling and exhaling compartments, 10-head harness, 11-pressure gauge, 12-pressure gauge tube, 13-oxygen supply tube, 14-cylinder valve, 15-press button bypass valve, 16-reducing valve, 17-oxygen cylinder, 18-inhalation tube.

Courtesy of Siebe German Co. Ltd.

The apparatus is so designed that when worn, the oxygen cylind remains on the back and the breathing bag and cooling unit on the front of th wearer. Weight of the apparatus when fully charged is 15.6 kg.

Wearer of Proto apparatus Mark IV cannot speak and th communication among brigade members is by pre-decided audible sign of horns.

Proto Mark V apparatus is an improvement over Mark IV. It available as a 1-hr or 2-hr apparatus. In 1-hr apparatus the coolant is Ca but in 2-hr apparatus it is soda phosphate as in Mark IV Wt. of 1 -hr appara is 14.5 kg. and that of 2-hr apparatus is 17.2 kg. oxygen flow rate in 2 apparatus is 2.0 lit/min. in 1-hr apparatus, 2.5 lit./min.

2-hr Mark V apparatus is fitted with a warning signal to indic approaching cylinder exhalation, and it can be supplied with a wide visi face piece (Vistaram mask) and a speaking diaphragm. The by-pass valv of push button type instead of the hand- wheel type of Mark IV. Excepting t differences stated above between the Mark IV and Mark V apparatus, th specifications of Mark V are the same as those of Mark IV. Latest apparatus the market is Mark IV model 80 which is practically like Mark IV, the sp parts of both being interchangeable.

Drager self contained breathing apparatus (model BG 174)

Like the Proto apparatus the Drager apparatus is also compres oxygen type, with closed circuit for the inhaled and exhaled air, rendering t wearer self dependent in an atmosphere of toxic and poisonous gases, whate their percentage.

Compared to Proto apparatus of 2-hr duration the Drager appara (model BG 174) is light, weighing only 12.8 kg when fully charged w compressed air, and it has duration of 4 hrs. The compressed O_2 is held in alloy cylinder at 200 kg/cm². Capacity of the oxygen cylinder is 2 litres a the oxygen available is 400 litres. Capacity of the breathing bag is appro mately 6 litres. The constant feed rate from the apparatus is 1.5 Lit/min. a a lung-governed valve automatically adjusts the oxygen feed to the wear requirement over and above the 1.5 lit/min.

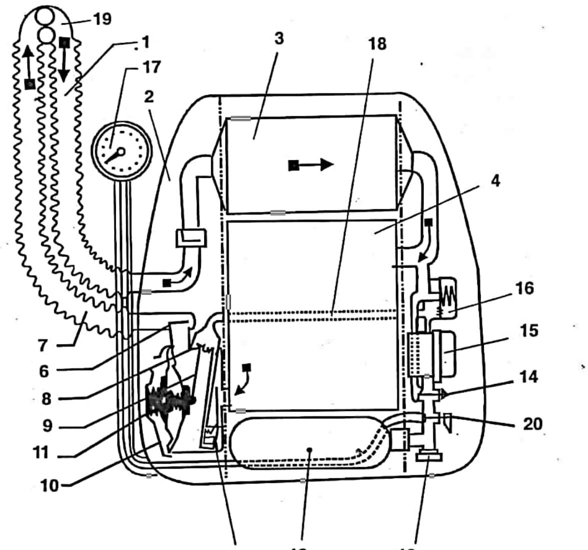


Fig. 6.2 Drager self-contained compressed-oxygen breathing apparatus, model BG 174.

1-exhalation tube; 2-exhalation valve; 3-regenerator, 4-breathing bag; 5-warning signal; 6-inhalation valve; 7-inhalation tube; 8-lung governed valve; 9-stem valve; 10-control memberance; 11-excess pressure valve; 12-oxygen cylinder;13-cylinder valve; 14-by pass valve; 15-reducing valve; 16-pre-flushing device; 17-pressure gauge; 18-oxygen supply tube; 19-breathing connection piece; 20-pressure gauge valve.

Courtesy of Joseph Leslie Agencies, Pvt. Ltd.

The apparatus is therefore completly automatic and the breathing air is controlled by the respiratory valves.

The exhaled air is regenerated by absorption of CO_2 as it passes over an air cooled regenerating alkali cartridge.

The main specifications of Drager Model BG 174 are :

Safe working period	4 hr hard work
Cylinder capacity, empty	two lit
Cylinder capacity with O_2 at 200 kg cm²	400 lit
Breathing big capicity, approx.	6 lit
Wt. of apparatus, fully charged	12.8 kg.
Oxygen flow rate	1.5 lit/min
Oxygen feed by lung governed valve	as required by wearer

The special features of the apparatus are :

1. When the cylinder valve is opened the breathing bag is automatically flushed with an inrush of 7 lit. of Oxygen. Thus it is not necessary to evacuate any nitrogen from the apparatus by breathing from it.
2. A warning whistle is fitted in the inhalation passage below the inhalation valve. It is controlled by the pressure in the oxygen line which leads from the pressure reducer to the lung demand valve. It gives a warning if the apparatus is used with the cylinder valve closed or if the cylinder is empty.
3. In addition to the lung demand valve which operates automatically in case of higher oxygen demand (in excess of 1.5 lit /min) there is a manually operated by-pass valve for by-passing the pressure reducer and admit more oxygen to the circuit.
4. A rechargeable soda-lime carridge can be used for training purposes.
5. The pressure reducer reduces the oxygen pressure to a tolerable working pressure 4 kg/cm.
6. The breathing bag is protected on all sides.
7. A face mask for wide vision (Panorama Nova mask) can be used instead of the gas tight protective goggles with triplex safty glasses.
8. Any excess pressure arising in the circuit in case of low oxygen consumption is eliminated through the automatic relief valve.

AUER /SAR-30 :

A chemical oxygen breathing apparatus for work and rescue marketed by M.S.A. Ltd. is **AUER/SAR 30.** It is manufactured by the firm of AUER in West Germany.

The AUER SAR 30 is a chemical oxygen breathing apparatus independent from the ambient atmosphere. It protects the user from toxic and oxygen deficient atmospheres for approx. 30 minutes. The condensed dimensions and a wide non-slipping harness permit use of the SAP 30 even under very difficult conditions as those found in shafts, manholes, small confined areas and sewerage systems.

The AUER SAR 30 consists of the full face mask with speech diaphragm, the breathing hose, the breathing bag with vent valve, the KO_2 canister with quick start system, the cansister cover with neck strap and waist belt and the electronic warning signal system. The ready -to -use SAR 30 is contained in robust non-corrosive steel case which is sealed.

The oxygen needed for breathing is produced by potassium superoxide, which is contained in the canister. It consists of coarse granules. Oxygen is liberated by the chemical reaction of moisture (water vapour contained in the exhaled air) with the KO_2. In addition to this, a second reaction takes place between pottasium hydroxide which is the product of the pottasium superoxide and water, and the carbon dioxide of the exhaled breath to combine and retain CO_2.

The equations governing the working of a self rescue breathing apparatus where KO_2 is used are as follows :

(a) Oxygen generator

$$2\,KO_2 + H_2O \rightarrow 2KOH + \frac{3}{2}O_2$$

$$2\,KO_2 + CO_2 \rightarrow K_2CO_3 + \frac{3}{2}O_2$$

(b) Carbon dioxide remover

$$2KOH + CO_2 \rightarrow K_2CO_3 + H_2O$$

$$KOH + CO_2 \rightarrow KHCO_3$$

The chemical reactions are self regulating; they respond to the quantity of CO_2 or humidity available and thus make the device demand-sensitive. The harder the user works, the more oxygen is generated and the more CO_2 is removed. The KO_2 liberates oxygen at a rate slightly faster than it can be consumed by the wearer; thus adequate oxygen supply regardless of work rate is ensured. The oxygen produced by the chemical is inhaled through the breathing bag which also serves as breathing air reservoir. The quick-start system covers the immediate oxygen requirements of the user until the chemical of the canister becomes activated by the exhaled air.

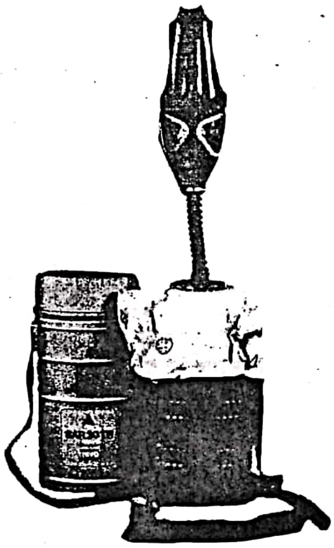


Fig. 6.3 Chemical oxygen breathing rescue apparatus AUER/SAR-30

Donning the face mask activities automatically the warning signal system, which releases *after approx.20 min.* a visual and audible retreat warning signal. This signal lasts for approx. 10 min. and reminds the wearer to leave the contaminated area. The chemical calnister is easy to exchange without special tools. The calnister cover serves as carrying appliance, heat insulating cover and shock protector. Arranged between breathing bag and breathing tube is the heat exchanger which cools the air heated up by the chemical reaction. In addition a paricle filter prevents the inhalation of dust particles which may come from the chemical.

The breathing bag is protected by a cover of aluminized glass fibers.

Technical Data : SAR 30 ready for use (without case) :

Dimensions. (without face mask) :

Height approx. 410 mm; *width* approx. 390 mm; depth approx. 160mm; Weight : approx. 4.7kg ;

Operating life : Approx. 30min.

Shelf life : Shelf life upto five years if stored properly ; if in readiness like on vehicles, upto three years.

Self Rescue Breathing Apparatus - Model Spiral 2C

The firm of Fenzy in France is manufacturing a self rescue breathing apparatus model Spiral 2C. It operates in a way similar to the conventional closed circuit breathing apparatus using potassium superoxide (KO_2).

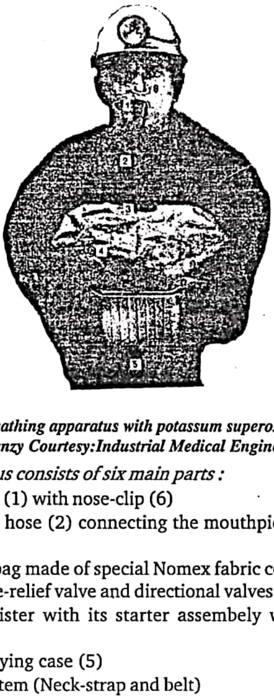


Fig 6.4 self - rescue breathing apparatus with potassum superoxide: Model spiral 2-C of Fenzy Courtesy:Industrial Medical Engineers

The apparatus consists of six main parts :

- a mouthiiece (1) with nose-clip (6)
- a corrugated hose (2) connecting the mouthpiece to the breathing bag (4)
- a breathing bag made of special Nomex fabric coated polyurethane, with pressure-relief valve and directional valves (3)
- the KO_2 canister with its starter assembely which gives oxygen immediately.
- a plastic carrying case (5)
- a harness system (Neck-strap and belt)

Its working is as follows :

(a) *Operation :* Suspend it round your neck, pull the red lever and take the cover away from the case. Then put the mouthpiece into your mouth,place the noseclip and pull the red plate of the starting system in front of you.

(b) *Breathing circuit :* The exhaled gases go by the corrugated tube through the KO_2 canister to be stored, regenerated, in the breathing bag (4) When inhaling, the gases stored in the breathing bag go to the directional valves by the couraged tube to reach the mouthpiece.

Technical data : Weight in its carrying case : 3.3 kg in use: 3 kg; Dimensions - breadth : 190 mm, height: 290 mm, depth: 120 mm; Duration- 1 hour to 4 hours (depending on the work performed); *Breathing resistance–* when exhaling : 20 to 50 mm column of water at the end of operation, when inhaling: 5 mm column of water; *Starting of apparatus* - Automatic by pulling the red plate; *concentration of* CO_2 - Constantly remains below 1 % *temperatures of the gases-* Constantly remains below 30 °C, even at the end of operation; KO_2 *Canister (interchangeable)* - 900 g with incorporated starter assembly.

Gas mask :

In gas masks, self rescuers and some other rescue apparatus the chemical **hopcalite** consisting of a mixture of manganese dioxide and copper oxide, is used as a **catalyst** which changes CO to CO_2. The conversion is an exothermic action by generation of heat.

This is an apparatus used by the wearer to protect himself from poisonous atmosphere containing mainly CO and also other toxic gases in small percentages. It is essential that the atomosphere should contain sufficient oxygen to support life and should not contain more than 2% of CO. If an atmosphere contains more than this percentage of CO, it is likely that it will not cotain sufficient oxygen to support life or combustion of a flame safety lamp. It should also he remembered that when the CO percentage is nearely approaching 2, it is an indication that the fire in an advanced stage, making the place unbearably hot for workers gas mask is not dependentt on others for respiration or air supply.

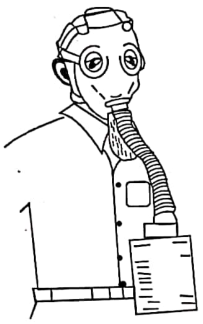


Fig. 6.5 Gas mask

A gas mask consists of :

(a) a metal containing layers of the following filters and granular absorbents.

i. Anhydrous calcium chloride as a drier to remove water vapour. This is usually the top-most layer.

ii. Hopcalite : It acts as a catalyst which changes CO to CO_2 . It also absorbs organic vapours

iii. Cotton wool to remove dust and smoke.

iv. Silica gel to remove ammonia and water vapour.

v. Caustic (caustic soda and pumice) to remove sulphuretted hydrogen.

vi. Impregneted activated charcoal to remove organic vapours and the acidic gases. This is the bottom-most layer.

The metal canister is fitted with an inlet valve and the atmospheric air, as it is inhaled, passes through an opening at the bottom of the canister through different layers mentioned above. The air must be well dried before it comes into contact with the hopcalite so that the latter is not affected by moisture and remains in granular form. Hopcalite ceases to be an effective catalyst when it becomes powder or undergoes hydration.

(b) A face piece fitted with eyepiece and an exhalation valve and connected to the canister by a corrugated hose pipe. Wide vision face mask (Vista mask) is also available inplace of the facepiece with twin eyepieces.

A gas mask weighs only 4 kg and it can be easily worn by an untrained person after a few minutes instructions. The wearer is free to use his both hands and his movements are not obstructed by the size or shape of the apparatus. Sealed canisters have a shelf life of 2-4 years but a canister should be discarded within one year after the seal is broken(if the apparatus has not been used). A wearer can use a canister for a service period of nearly 1-2 hrs in an irrespirable atmosphere provided it satisfies the conditions stated above.

In a mine of a team of rescue workers has to use gas mask and work in a poisonous atmosphere, it should always carry a lighted flame safety lamp or other device to detect deficiency of oxygen. A rescue team equipped with self contained breathing apparatus should be in reserve at the fresh air base if a gas mask team is working inbye of the base.

In a gas mask, and also in a self rescuaer (described next), heat is generated as CO is converted to CO_2 by the catyalytic action of hopcalite and the heat is imparted not only to the apparatus but also to the inhaled air. Though the hopcalite used is effective for 2 % CO or more in the inhaled air, the heat generated makes the inhaled air unbearly hot causing burns in the mouth and throat and therefore a gas mask or self rescuer should not be used in an atmosphere containing more than 2 % CO.

Self rescuer :

A self rescuer is essentially a gas mask in a simplified form without the corrugated hose tubing and the mouthpiece is attached directly to the canister. Its weight is low, nearly 1 kg. The chemicals are the same as those used in the gas mask and the wearer is protected against CO if it does not exceed 2 % by volume in the air to be inhaled, if the oxygen is sufficient to support life and if the air does not contain other toxic gases and vapour. The self rescuer does not supply oxygen but functions to convert carbon monoxide , a highly toxic gas, into carbon dioxide, a non toxic gas. The main purpose of a self rescuer is to enable the wearer to escape through an atmosphere resulting after a fire or after an explosions in a mine. In foreign countries a miner going underground has to carry a self rescuer with him, according to the mining laws, but such provision does not exist in our mining laws at present. The self rescuers are stored on the surface in the lamp room or in a self rescuer room adjacent to the lamp room thereby enabling the worker to pick up his lamp and the self rescuer at the same time when going down the mine.

When must a Filter Self Rescuer be used? A self rescuer should be used immediately at the first sign of fire or explosion-even if no smoke is visible. Waiting until smoke is visible may prove fatal because the area could be filled with a poisonous concentrated odourless, colourless CO in advance of the smoke. The miner should therefore don the rescuer immediately on seeing any one of the following signs : clouds of smoke, smell of combustion gases, headache and giddiness, unexpected dust eddies, sudden pressure surge. Any of these signs may be indicative of fire or an explosion.

Fig. 6.6 shows the constructional features of one self rescuer, model IW-65. manufactured by M. S. A. Ltd. The drying agent is charcoal impergnated with a mixture of calcium bromide and lithium chloride

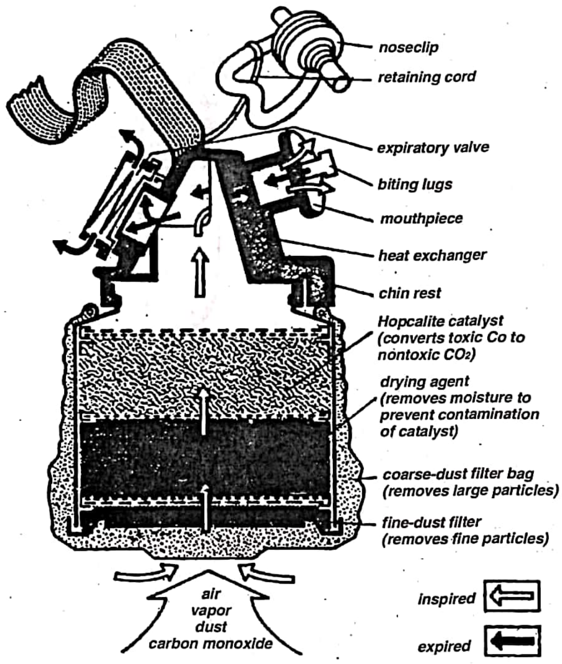
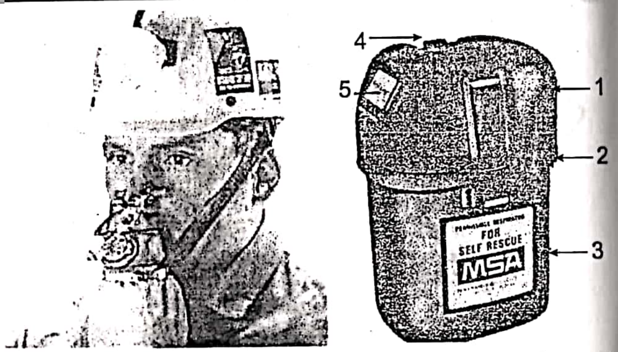


Fig. 6.6 Auer self rescuer I.W.65 Sectional View

(approx. 53 % $CaBr_2$ and 5% LiCl) . The CO catalyst is hopcalite, as stated earlier, effects the conversion of CO to CO_2 which is accompanied by generation of heat of reaction. The filter section has an outer coarse-dust filter and an inner fine-dust filter to remove dust particles. The filter materials are seperated by screens and baffles. The self rescuer is placed in a container is one kg. The self rescuer can be used for one hour after its seal is broken in atmosphere of 1% CO and the actual filter canister has to be replaced before re-use, but the manufacturers recommend that the self rescuer should be discarded after one use only.



1. Cover 2. Retaining band
3. Container. 4. Release lever.
5. Identification tag holder.

6.7 A miner with the rescuer on.

The model IW-65 incorporates a heat exchanger but does not have an inspiratory valve. Inspired air is cooled by the heat exchanger before inhalation. The exhaled air passes back through the heat exchanger and out through the spring-loaded expiration valve. Excess saliva is also expelled through the expiratory valve.

Tests at 1.5 % CO in atmospheric air showed that the heat exchanger will effectively reduce the temperature of inhaled air from approximately 150°C to 65° C. Though uncomfortable, one can tolerate even higher inhaled air temperature since the repiratory system itself is an effective heat exchanger.

Drager Self Rescuer, Model 810 :

A self rescuer model 810 is manufactured by the firm of Drager. The cotainer is closed by negative pressure (vacuum sealed) The difference between the normal external pressure produced on closing, forms the locking force (about 70 kg) which is impervious to water vapour. Weight of the self rescuer is nearely 1 kg and height 135mm. It gives protection for atleast 1½ hours in atmosphere containing 1% CO. Except for the vacuum sealing, other constructional main features of Drager 810 are like M. S. A. 's IW-65.

In an emergency to a on Drager 810 self rescuer, the opening lever is lifted, producing an aperture at a predetermined breaking point in the cover. The pressure of the atmosphere and the interior of the container is immediately balanced. The cover can now be removed easily. The self rescuer is taken out

and put on. After the self rescuer is donned the breathing is by mouth and not by nose which is closed by noseclip (for both M. S. A. and Drager).

The importance of training the user prior to an emergency in the use of the noseclip and of breathing through the self rescuer at all times until fresh air is reached, is emphasised by the fact that CO concentration of 0.5% (5000ppm) can cause rapid collapse, unconsciousness, and death within a few minutes. It is far better to be alive with a hot or even blistered mouth than to be overcome or killed by CO. Do not sneak a breath or two of relatively cool air into the mouth by opening the lips.

After putting on the self rescuer the worker should hurry up to reach the safe atmosphere. What course he should take and how he can reach the safe atmosphere will depend upon the circumstances of each situation and the worker should not become flustered or panicky He should take directions from his immediate superior officer, if the latter is available, and with cool thinking, he should leave the area quietly. The more quickly one runs, the higher is the breathing resistance. If exhausted, he should pause for a moment. After putting on the self rescuer, the worker should not speak when in an irrespirable zone (as CO is inhaled despite the filter during talk) but communicate by signs and nods.

A filter self rescuer is only an escape until and should in no case be used for reconnaisance or as a working unit, or for fire fighting.

Air pressure hose apparatus :

There are two types of rescue apparatus for use at a place of irrespirable air which is not far from an area of fresh air, upto about 50 m They are :

1. Smoke helmet.
2. Hose mask.

Both these apparatuses require the wearer to depend upon other man for supply of fresh air when the wearer himself is working in an irrespirable atmosphere. The **smoke helmet** is a helmet made of tough polyathene having mica eye pieces fitted in aluminium frame. It is provided with canvas strapping to cover the shoulder. The helmet totally covers the face and head of the wearer and is provided with a flexible hose pipe of 25 mm bore, 30m long. The wearer receives supply of fresh air through the hose pipe from a hand-driven bellow in fresh air operated by a second person. The bellow is operated sufficiently fast to supply requirement of air of the wearer. The exhaled air passes from the helmet via the loose joints of canvas strappings around the shoulders of the wearer.

The hose mask, one example of which is the **LA-IF** line respirator marketed by Industrial Medical Engineers, differs from rhe smoke helmet in that it has undirectional separate inhalation and exhalation valves, an air filter to prevent entry of dust and foreign matter, a corrugated tube between the mask and the air tube of rubber (30-100m). The mask permits more freedom of movement to the wearer's head as compared to the smoke helmet. The corrugated hose pipe connected to the mask forms an air reservoir or "equaliser".

In both the above apparatus the air hose pipe is under a continuous positive pressure and there is no danger to the wearer from the leakage of outside irrespirable atmosphere into the apparatus. The maximum length of air hose to be used with pressure hose apparatus is 50 m if manual bellow is used. When using air from a compressed air supply pipe line, an oil filter should be used for cleaning the air. At the place from where fresh air is supplied by the bellow to the wearer a lighted flame safety lamp and a canary should always be kept for indications of oxygen shortage, carbon monoxide and methane.

Reviving apparatus :

A reviving apparatus is used to administer pure oxygen (i) to an unconscious person, (ii) to one whose breathing is considerably feeble for some other cause. Accidental burial under a heap of landsliding earth or other material, drowning chocking, traffic accidents, collapse-these are other occasions demanding use of reviving apparatus. The apparatus is sometimes used in conjunction with some other method of artificial respiration and is then brought into operation after the patient's breathing is rapidly restored. In an emergency, a victim who is being carried from inbye irresrirable atmosphere of a mine to a fresh air base, can be given treatment from a reviving apparatus by the rescue team on the way as the team is still on the move.

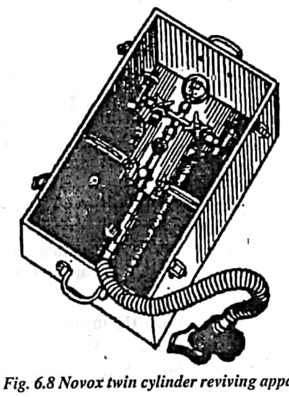


Fig. 6.8 Novox twin cylinder reviving apparatus.

The treatment has, however, to be given when the patient and rescue team are passing through a zone free noxious gases or at the fresh at the fresh air base.

Fig 6.8 The Novex Reviving apparatus consists of :

(a) Two cylinders, each containing 400 lit. of oxygen charged to 120 atmospherers.

(b) Reducing valve.

(c) Hand-control valve.

(d) Dial calibrated from zero to 30 litres per min.

(e) Flexible breathing bag.

(f) Corrugated tube, with inhalation valve, leading to a face mask fitted with exhalation valve, and also an inlet valve opening to the other atmosphere.

(g) Pressure gauge and main valve.

The complete apparatus weighs 16 kg. A somewhat lighter single-cylinder model is also available.

In using the apparatus, one of the main valves is opened and the hand control valve is opened and the hand control valve adjusted so that the breathing bag is not allowed to collapse, the rate of flow usually ranging from 10 to 30 litres per min. according to the rate and depth of the patient's breathing. The quantity of oxygen breathed (within the controlled limits) thus depends on the patient himself. If the patient requires more than 30 litres per min., he draws additional air through the inlet valve on the mask.

Because of its size and weight the rescue team can carry only one apparatus inbye the fresh air base, in addition to the other equipment which the team has to carry.

Pulmotor, reviving apparatus of Drager :

The reviving apparatus manufactured by the firm of Drager is known by the trade name **Pulmotor. (Fig. 6.9)**. It is used to give effective, immediate assistance in case of gas poisoning and where breathing has stopped temporarily or become feeble in situations stated earlier. The Pulmotor switching unit to which a face mask of suitable size is attached is hose-connected to the Pulmotor distributor manifold and fed with oxygen from the cylinder. Some models such as PT-60, have two oxygen cylinders, and in some (e. g. PT-61), only one. A cylinder is of 2.5 litre capacity holding oxygen under a pressure of 200 bars, i. e., 500 litres of oxygen at atmospheric pressure. The changeover valve of the Pulmotor is normally set to 50 % O_2, the balance bring fresh air entertained by an injector. The device can be changed over to 100 % O_2 in gas contaminated atmosphere. Provided that the respiratory tracts are clear and the mask fits tightly, the respiratory air flows into the lungs of the victim and produces a clearly visible expansion of the thorax.

As soon as a pressure of about + 20 mbar is reached in the lungs, and the inspiratory phase is thus completed, the appliance switches over automatically to expiration and begins to suck the air out of the lungs. Then when the required negative switch-over pressure of-10 mbar is reached, towards the end of the expiratory phase, a new inspiratory phase begins.

The switching rate depends on the size of the lungs and on the oxygen inflow. If switching is too fast, the oxygen supply must be reduced at the distributor control valve and increased if switching is too slow. The negative pressure at the end of expiration helps especially to assist the circulation; but in special cases (*e.g.* corrosive gas poisoning) pressurised respiration only can be advisable. A Drager special attachment, supplied for this purpose, is the respiratory valve for cutting out the negative phase, inserted between the Pulmotor switching unit and the face mask to ensure that negative pressure is not produced during the expiratory phase. Expiration takes place spontaneously through the collapse of the expanded throax.

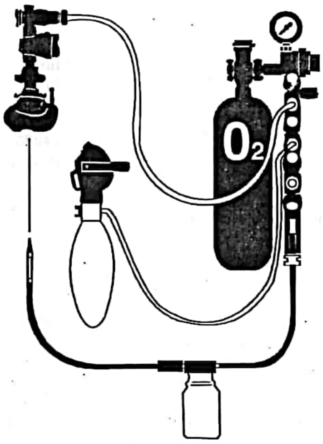


Fig. 6.9 Pulmotor

The Pulmotor switching unit indicates faults in use automatically: (1) Unless the mask fits tightly the pressure required for switching over cannot build up and the Pulmotor switching unit remains inoperative. In such cases it is usually sufficient to press the mask more firmly against the face to prevent the respiratory air escaping. (2) If the respiratory tracts are blocked, this is indicated by very rapid switching (" chattering"). Lifting the lower jaw and extending the neck is often sufficient to clear the respiratory tract; but if the respiratory tracts are blocked by secretions, water or blood, then removal by suction is necessary.

This ability of the switching unit to indicate whether or not the lungs are being supplied with adequate respiratory air is exclusive to the Pulmotor.

Blocked respiratory tracts frustrate any attempt at ventilation of the lungs. Pulmotors are equipped with oxygen powered injectors. Their powerful suction removes mucus, water and blood from the respiratory tract in seconds. The extracted secretions are collected in a receptacle and respiration can then begin or be continued forthwith.

The breathing bag in which inflowing O_2 accumulates during the expiratory phase helps to conserve oxygen. Its movement enables the breathing to be observed and the necessity of ventilation of lungs to be determined.

Resuscitation :

It is the process of inducing artificial repiration in a person who is unconsious and whose rate of breathing has become considerably feeble. Such occasions may arise if a person has received electric shock or has inhaled oxygen-deficient air or toxic gases. When the breathing is too feeble the lungs are not ventilated with adequate quantity of air or oxygen which the patient needs and he may die in course of time, sometimes within a few minutes. The resuscitation operation must be continued once it is started till natural breathing is restored or till it is considered that it will be of no avail. The processes of *manual* artificial respiration, commonly used, are :

1. The Schafer's method.
2. Sylverster's method.
3. Mouth to mouth resuscitation.

1. Schafer's method :

Adjust the patien's in a prone position (i. e. back upwards0. with arms extended above the head, and his head turned to one side, so as to keep his nose and mouth away from the ground. Do not waste time loosening clothing ; no pad is to be placed under the patient, nor need the toungue be drawn out, as it will fall naturally towards the lips. (Fig 6.10)

Fig. 6.10 Schafer's method of respiration. Top-Inspiration, Bottom- Expiration

To turn the patient to the prone position, stoop at his side, place his arms close to the body, cross his far leg over his near leg and , protecting his face with one hand, with the other, grasp the clothing at the hip on the opposite side of the body and pull smartly over.

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Imitate the movements of breathing. Induce expiration : Kneel at one side of the patient,,or keep his thighs between your thighs, facing his head and sitting on your heels. Place the palms of your hand on the patient's loins, their lower edges just clearing the top of the pelvis, the wrists nearly touching, the thumbs as near each other as possible without strain and the fingers passing over the loins on either side and pointing towards the ground, but not spread out. bending your body from the knees and somewhat straightning the hip-joints swing slowly forward keeping your arms quit straight and rigid so that the weight of your body is conveyed to your hands directly downwards. No exertion is required; the necessary pressure is imparted by the weight of your body. In this way the patient's abdomen is pressed against the ground; the abdominal organs are forced against the diaphragm; the diaphragm rises and air is driven out of the lungs together with any water or mucus which may be present in the air passages or in the mouth, producing expiration.

Induce inspiration : Swing your body slowly backwards to its first position thus removing the weight from the hands (which are kept in position) and relaxing the pressure on the abdomen.

The organs now assume their former position, the diaphragm descends, the throax is enlarged and air passes into the lungs, producing inspiration.

Alternate these movements by a rhythemic swaying forwards and backwards of your body from the knee joints, twelve times a minute. The rhythm is -pressure two seconds and relaxation three seconds.

When natural breathing begins regulate the movements of artificial respiration to correspond with it, and promote circulation by rubbing the limbs vigorously towards the heart and by applying warmth.

2. Sylvester's method :

This method is to be used only when it is impossible to turn the patient on to his face.

Adjust the patient's position; without wasting a moment, place the patient on his back on a flat surface, inclined if possible from the feet upwards. Undo all tight clothing. Raise and support the shoulders on a small firm cushion or folded article of dress placed under the shoulder blades.

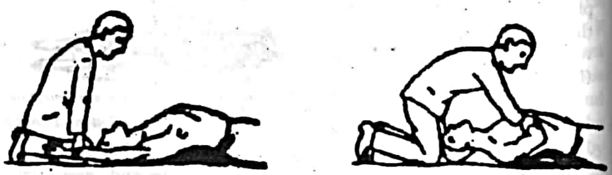

Fig. 6.11 Sylvester's method of artificial respiration.
Left -Inspiration; Right - Expiration.

Maintain a free entrance of air into the windpipe. An assistant must catch hold of the patient's tongue with a handkerchief, draw it forward as far as possible, and hold it in that position. If this is not done there is a great danger of obstruction of the windpipe by the tongue falling back over the top of it.

Imitate the movements of breathing. Induce inspiration :

Kneel at a convenient distance behind the patient's head and grasping his forearms just below the elbows, draw the arms upwards, outwards and towards you, with a weeping movement, making the elbows touch the ground. The cavity of the chest is thus enlarged and air is drawn into the lungs.

Induce expiration :

Bring the patient's flexed arms slowly forwards, downwards and inwards; press the arms and elbows firmly on the chest on each side of the breast-bone. By this means air expelled from the lungs.

Repeat these movements alternately, deliberately, and preseveringly about twelve times a minute. The rhythm is pressure two seconds and relaxation three seconds.

3. Mouth to mouth resuscitation :

This is the simplest and an effective method which can be used during emergency. In this method, the rescure or first-aider blows air forcibly into the patient's mouth or nose to inflate his lungs and stimulate the act of inspiration. No efforts are required for expiration of air as it occurs due to the recoil of the elastic chest walls. When applying the method the first aider kneels down behind the head patient, puts his face on the patient's face and blows his own breath hard into the patient's mouth. After the blowing the first-aider moves his face aside, takes a deep breath and again blows his breath into the mouth of the patient. This operation is continued till the patient shows signs of breathing. A rhythm of 12 to 15 operations/min should be maintained. The method may be considered unhygienic but where the patient's life is to be saved, considerations of hygene have to be brushed aside.

When adopting any of the methods of artificial respiration time factor is very important. Moreover, the first-aider should know what treatment the patient needs first; if a victim is having arterial haemorrhage artificial respiration should be given only after the uncontrolled arterial haemorrhage has been stopped by proper treatment.

Rescue Stations :

Rescue stations for helping the mining industry have been established by the Central Govrnment at central places like Dhansar (near Dhanbad), Sitarampur (near Asansol), ramgarh, Parasia and few other places. These are under the management of Coal India Ltd's subsidiary companies like BCCL, ECL, WCL,ets. functioning in the area. A rescue station is under the control of Superitendent of Rescue Station and has in its employment one or more rescue brigades which consist of workers trained in rescue operations with different types of rescue equipment and having experience of underground mining. The members of the rescue brigade are able to perform miscellaneous jobs like that of a mason, timber setter and others. The rescue station is equipped with equipments of different types such as self contained compressed oxygen apparatus, reviving apparatus, gas mask, self rescuer, smoke helmet, pressure hose mask, etc. and has facilities for regular inspection, repairs and maintenance of the equipment.A rescue brigade consists of 5-6 members , including a leader all of whom stay near the rescue station and undergo regular training and exercise daily at the rescue station, and sometimes at the mines. The brigade members have to be in constant readiness to attend any emergency call demanding their services and when a mine requires help of a rescue team, the mine authorities have to telephonically inform the rescue station and brigade is available at the spot within the shortest possible time.

Each rescue station trains selected mine workers of nearby mines for rescue operations and keeps a list of such trained workers for summoning them in case of emergency at any mine within the area covered by the concerend rescue station.

Some mines have their own rescue stations, *e. g.* at Mosabani Copper mines, Kolar Gold Field mines, etc.

Rescue operations at the mine ;

The rescue team has to establish a fresh air base from where the team proceeds into the affected part of a mine. Such fresh air base is as close to the affected part as possible and, as the name suggests, has to be at a place where fresh respirable air, unpolutted by fire or explosion is available. If the entire mine is affected by a fire or explosion the fresh air base is at the surface at the start of rescue operation; otherwise it is underground at a suitable place close to the irrespirable zone of the mine. The fresh air base is equipped with the following persons and materials.

1. Two men of whom one is rescue trained.
2. A rescue team consisting of 5-6 persons including a leader, fully equipped with self contained breathing apparatus and ready for service. Such team is in addition to one that goes inbye.

3. Rescue equipment like reviving apparatus, smoke helmet, gas mask, etc.
4. First aid equipment, stretchers, tools and mining supplies of all kinds likely to be required during rescue and recovery work, bleaching powder as disinfectant when removing dead bodies.
5. Flame safety lamps, *munia* birds and other means of detecting carbon monoxide to ascertain the atmospheric conditions at the base itself. Additional equipment of similar type should also be kept at the base as it has to be carried inbye by the rescue teams.
6. Hygrometer.
7. Drinking water.
8. Mine plane and rescue tracing on a large scale.
9. Fire extinguishers.

The rescue team has to proceed inbye of the fresh air base as a complete unit and no member is allowed to stray away. The leader leads the team and each member occupies a definite place as the team is on the move. If any member experiences trouble due to physical causes or due to any defect in the apparatus; the entire team has to return to the fresh air base. With most of the apparatus used for rescue operations, the wearer has to communicate with others only by nods, hand signals and coded sounds of horns since speaking would endanger the wearer's life in toxic atmosphere.

The mine manager or responsible authorities should advise the team leader on what the team has to do where it has to proceed. The instructions should be in writing accompanied by a plan of the district. If possible, a rescue trained worker familiar with the district should be included in the team. The rescue team has to complete its assignment within the period limited by the duration of the self-contained breathing apparatus and it should return to the fresh air base within that period. No member of the rescue team should be asked to do a second spell of work, unless medically examined and certified fit.

The route followed by the team must be marked in chalk and if the atmosphere is smoky and the visibility poor, a lifeline must be exteneded from the base as the team proceeds inbye. A lifeline is a thin white polythene rope. The leader should not engage himself in manual work but should supervise and direct the work of his team members. The team should not proceed inbye unless the roof is safe or made secure by the team.

The distance which a rescue brigade wearing self contained breathing apparatus has to cover in an irrespirable atmosphere depends on the following factors :

1. Capacity of the apparatus. If the apparatus is provided with oxygen to last two hours, the rescue team must return to the fresh air base within one hour and 45-50 minutes. If the apparatus of any rescue member developes any defect the whole team must return to the base.
2. The temperature of the underground atmospher, the gradient of the roadways and the load carried by the team influence the team's capacity to work and the distance they can cover.
3. Obstructions in the roadway like roof fall, twisted rails, tubes jammed upto the roof, etc. A roomy well supported roadway free from roof falls will enable the team to cover a longer distance.
4. The nature of duties expected to be performed by the team. If the team has to mark on the rescue tracings such details as cross-sections of the roadways for constructions of stoppings, position of machinery and other obstructions thrown away in the roads by the explosion, *etc.* the distance that can be covered is naturally less.
5. Physical condition of the least efficient member of the team as the whole brigade has to travel and work together.

The fresh air base is advanced from time to time as the team as the exploration proceeds.

Attempts to rescue the workers should be done only by rescue teams using self contained breathing apparatus. Sometimes the hysterical of survivors suffering from shock or due to loss of near realtives impedes the rescue operations.

The danger and difficulties in rescuing survivors after an explosions are :

1. The shaft, cages and headgear may be damaged and unsafe for travel.
2. Telephonic communication to underground may be disrupted.
3. Surface fan may be damaged and ventilation deranged by short circuit of the air current through broken stoppings. blown up ventilation doors, damaged air crossings roof falls, ets.
4. Underground roads are blocked by roof falls, tubs or mine cars jammed up upto the roof, machinery and materials thrown in the way, twisted rails, cables,etc.
5. Smoke and dust considerably brings down the visibility.
6. Fumes, noxious gases and afterdamp are fatal to the men attempting to rescue the survivors.

7. A gas explosion is sometimes followed by a series of explosions at short intervals.
8. Fires may be caused by the explosions. Their smoke fills the roadways rendering them impassable, and also dangerous due to presence of carbon monoxide.
9. Lack of ventilation and heat of explosion raises the temperature of underground atmosphere considerably high.
10. Delivery pipes of pumps may be broken and water filled in the depressions on the routes. Deep areas may be flooded.

QUESTIONS

1. State the different types of rescue equipments used in mines. Describe the principle of working of a self rescuer and its construction.
2. What is a self contained breathing apparatus? Describe one such apparatus used in Indian mines.
3. State the nature of duties of the leader of a rescue brigade from the time he reports to a mine manager after getting a duty call after an explosion till returns to the fresh air base after underground work by the rescue team.
4. Write short notes on :

   (i) Fresh air base (ii) care and maintenance of self rescuers (iii) smoke helmet (iv) gas mask (v) artificial respiration.
5. A worker has entered inadvertently in the goaf area of a coal mine and has fallen unconscious due to firedamp accumulation. You are the overman of the mine. State how you will proceed to rescue the worker after information of the incident.
6. State the difficulties that a rescue team may have to face underground when performing their duties after an explosion.

O O O

CHAPTER 7

RECOVERY WORK IN A MINE

An area in an underground part of a mine might have been sealed due to spontaneous heating, explosion, fire, irruption of noxious or inflammable gases or completion of depillaring operations. In some cases an area may be sealed by isolation stoppings after the completion of development if depillaring is planned to take place after a long time. Recovery of an area sealed due to fire or explosion is not free from risks and some preparatory work is essential before connecting the operations to recover the area.

Reopening a Sealed off fire Area :

Before taking a decision to reopen a sealed off fire area the mine authorities have to ascertain the conditions behind the stoppings in respect of temperature, state of fire, composition of gases and their pressure. The temperature of the stopping, as felt by hand, may to the regular supervisory staff some indication whethere the temperature of the sealed of fire area is increasing or decreasing. This is reliably observed from the thermometers inserted in the stoppings through built-in pipes. Other aspects of the conditions behind the stoppings can be known after interpretation of the analysis of the gas samples taken from the sealed area at regular intervals over a long period. Such samples should preferably be collected when the pressure behind the stoppings is positive as the sample is then likely to be truly representative of the atmosphere in the sealed off area.

It is recommended that samples should be taken when the barometric pressure is low (between 12 noon and 2 P.M.) Fig. 7.1 shows a method of collection samples with the help of a suction pump when the pressure behind the stopping is negative. Samples may also be collected by the method of water displacement. In all cases samples should be collected from sampling pipes which are at least 3min side the stopping and the air which was present in the samplng pipe should be allowed to escape before the proper sample is taken.

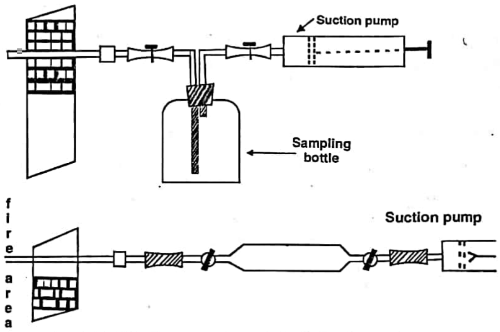

Fig. 7.1 Collecting samples with the help of a suction pump.

Such sample would be representative of the atmosphere behind the stopping and would be reliable. If a rescue team wearing self contained breathing apparatus has to collect spot samples after entering a sealed area, the samples are collected in the following manner.

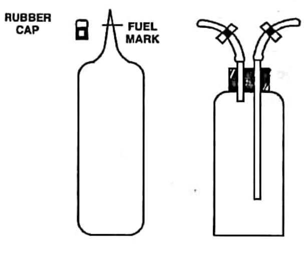

Fig. 7.2 Sampling bottles. Bottle on the left is vacuum filled.

1. By displacement of water : The bottle should be filled with clean surface water and not by mine water. It is preferable to fill the bottle with acidified water coloured with methyl orange.

   The moisture on the inside surface of the bottle may, however, dissolve small quantity of soluble gases like CO_2.

2. By inflating a rubber bladder with an air-filling pump at the spot of sample.

3. By breaking the neck of a vacuum sampling bottle at the spot where sample of air is to be collected. It is instantaneously filled with the air sample and is then sealed by a rubber cap. The vacuum bottles are usually 250–300cm³. (Fig. 7.2). The bottle should be labelled.

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How long the area should be kept sealed after an outbreak of fire is difficult to assess. In some cases it may he months; in others, a few years. In general an area sealed off on account of a conveyor or electrical fire can be opened earlier than an area sealed off an account of spontaneous heating.

Interpretation of analysis of gas samples :

1. The graph (Fig. 7.3) shows the trend of composition of gases behind the stoppings with the passage of time in a coal mine. If the fire is active three is gradual decrease in percentage of oxygen, gradual increase in percentage of CO_2 (say upto 5 or 6%) and the ratio of CO_2 formed/oxygen absorbed is high. CH_4% goes up rapidly in a gassy mine.

2. When the percentage of O_2 falls below nearly 12, actual flame ceases but the members continue to be hot and slow combustion continues when O_2% is as low as 4–5 resulting in formation of CO and CO_2 and evolution of heat. The percentage of CO_2 and CO gradually goes down and that of CH_4 gradually increases.

3. When the percentage of oxygen comes down to nearly 2 and remains at that level or below for some weeks the fire may be considered to be extinct. The CO/O_2 ratio at that stage is nearly the same as is normal to the coal seam when it was being worked. The CH_4 percentage gradually goes up and the atmosphere behind the stoppings is practically full of CH_4.

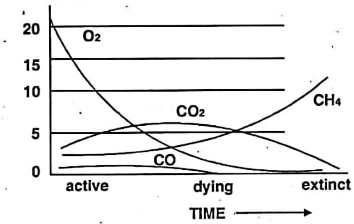


Fig. 7.3 Graph of analysies from sealed off area.
The figures 5, 10, 15, 20 represents percentages

The following analysis (Table) of air samples from a sealed off area is available. Its interpretation is given below.

TABLE 1

Date	O_2	N_2	CH_4	H_2	CO	CO_2	CO/O_2	CO_2/O_2
29-8-91	7.13	88.08	0.00	0.00	0.00	4.79	0.00	29.70
1-9-91	4.44	88.64	0.18	0.00	0.00	6.74	0.00	35.50
4-9-91	2.44	89.62	0.14	0.00	0.00	7.80	0.00	38.10

The analysis indicates that oxygen percentage is gradually falling and CO_2 percentage is steadily rising. The ratio CO/CO_2 is nil and CO_2/O_2 gradually rises. This is indicative that the fire is dying. The fact that the oxygen percentages is gradually falling indicates that the stoppings are reasonably airtight and mine air indicates that the stoppings are reasonably airtight and mine air not leaking into the fire area. Methane percentage indicates that the mine is only nominally gassy.

Table 2 gives results of analysis of air samples from behind the stoppings in another mine. Its interpretation is as follows :

(i) The first two samples show the normal trend of a dying fire and the percentage of CO_2 and oxygen are decreasing. The ratios CO/O_2 as also CO_2/O_2 are decreasing.

TABLE 2

Date	O_2	N_2	CH_4	H_2	CO	CO_2	CO/O_2	CO_2/O_2
22.12.91	1.83	89.58	0.02	0.00	0.08	8.49	0.37	38.7
24.12.91	1.67	89.96	0.00	0.04	0.04	8.27	0.18	37.3
4.1.92	4.86	87.55	0.00	0.02	0.02	7.55	0.11	41.2
12.1.92	1.80	89.41	0.00	0.00	0.00	8.79	0.00	40.2
18.1.92	2.22	89.32	0.00	0.00	0.00	8.46	0.00	38.6
24.192	1.91	89.51	0.00	0.00	0.06	8.52	0.28	39.1
29.1.92	1.84	89.32	0.00	0.00	0.06	8.78	0.28	38.0.

(ii) The third sample shows a sudden rise in percentage of oxygen from 1.67 to 4.86. This indicates that air is leaking into the fire area and the fire stopping is not leakproof. The dilution of the atmosphere is further evident from the *sudden* fall of CO_2 from 8.27% to 7.55%. Such dilution of gases behind the stopping may be due to small cracks in the stopping or because it is not made sufficiently airtight. The leakage may also be due to passage of air into the sealed area from cracks in the roof coal or strata above the stopping. Cracks in the floor strata below the stopping are not common though the possibility should be kept in mind. In a shallow mine the leakage may be from the cracks extending upto the surface.

If the leakages are stopped after studying the samples the oxygen percentage in the subsequent samples will show a downward trend. The ratio CO_2/O_2 and CO/O_2 will also gradually decline.

(iii) The mine is non-gassy.

(iv) The fire is nearly extinct.

Table 3 gives analysis of air samples from behind the fire stoppings in a mine.

The interpretation of the analysis is as follows :

The factor that immediately catches attention is the rapid and continuous rise in the percentage of methane, indicating a very gassy mine. the final atmosphere consists practically of CH_4 gas. To interpret the conditions behind the stoppings it is essential to calculate the CO/O_2 ratio and CO_2/O_2 ratio.

Third day (return stopping).

$$\text{Oxygen equivalent of 73.24\% Nitrogen} = \frac{20.93}{79.04} \times 73.24$$

$$= 19.39\%$$

Oxygen absorbed = 19.39 – 16.35 = 3.04%

CO_2 produced = 0.56 – 0.03 = 0.53%

$$CO_2/O_2 \text{ ratio } = \frac{0.53}{3.04} \times 100 = 17.4\%$$

$$CO/O_2 \text{ ratio } = \frac{0.400}{3.04} \times 100 = 13.16\%$$

Ninth day (return stopping).

$$\text{Oxygen equivalent of 34.8\% nitrogen} = \frac{20.93}{79.04} \times 34.8 = 9.21\%$$

Oxygen absorbed = 9.21 – 8.30 = 0.91%

CO_2 produced = 0.38 – 0.03 = 0.35%

TABLE : 3

The stopping in the Intake Airway

Samples taken	CO_2	CH_4	O_3	N_2	CO	Total
On completion of stopping	0.120	0.190	20.700	78.990	nil	100.000
3 days after completion	0.350	4.500	19.300	75.845	0.005	100.000
6 „ „ „	0.430	9.650	17.804	72.110	0.006	100.000
9 „ „ „	0.390	35.000	12.500	52.104	0.006	100.000
17 „ „ „	0.340	42.200	11.056	46.400	0.004	100.000
22 „ „ „	0.300	59.000	8.000	32.697	0.003	100.000
41 „ „ „	0.150	65.300	5.448	29.100	0.002	100.000
52 „ „ „	0.100	70.500	3.000	26.400	nil	100.000
58 „ „ „	nil	89.000	nil	11.000	nil	100.000

The stopping in the Return Airway

Samples taken	CO_2	CH_4	O_2	N_2	CO	Total
On completion of stopping	0.350	1.800	19.725	78.100	0.025	100.000
3 days after completion	0.560	9.450	16.350	73.240	0.400	100.000
6 „ „ „	0.500	26.390	14.550	58.505	0.055	100.000
9 „ „ „	0.380	56.511	8.300	34.800	0.009	100.000
17 „ „ „	0.200	78.505	nil	21.290	0.005	100.000
22 „ „ „	nil	90.000	nil	10.000	nil	100.000
52 „ „ „	nil	93.000	nil	7.000	nil	100.000
58 „ „ „	nil	95.000	nil	5.000	nil	100.000

Note – Figures give percentages

$$CO_2/O_2 \text{ ratio} = \frac{0.35}{0.91} \times 100 = 38.5\%$$

$$CO/O_2 \text{ ratio} = \frac{0.009}{0.91} \times 100 = 1.0\%$$

Of the two ratios the CO/O_2 ratio is more important and it indicates that the fire is burning actively on the 3rd day but has died down considerably by the 9th day. Even without calculating the CO/O_2 ratios, similar inference can be drawn by a look at the CO and CO_2 percentages but the method of interpretation based on CO/O_2 and CO_2/O_2 ratios is more scientific.

The various conclusions from the interpretations of the analysis figures can be summarised as follows :

1. Active combustion occurs chiefly during the first 6 or 8 days and during this period the percentages of CO and CO_2 increase but the percentage of oxygen goes down and the fire gradually dies down with the fall in oxygen percentage.
2. Slow heating continues throughout most of the period till the oxygen percentage falls down to 2% at the intake stopping.
3. At the end of 58 days the fire is completely extinguished. It is however prudent to wait for a few months more and watch the trend of temperature and gas analysis before contemplating to reopen the fire area.
4. The seals are reasonably airtight as there is no fluctuation in the percentages of oxygen and nitrogen and the CH_4, percentage is gradually rising without any dilution by leaking air.
5. There is danger of firedamp explosion between the 3rd and 6th or 7th days, i.e. when the percentage of CH_4 lies within 5% and 15% and the percentage of oxygen exceeds about 13%. Thereafter no such danger exists partly because the percentage of CH_4 is too high and partly because the percentage of oxygen is too low.

It should be noticed that the figures of analysis at the return stopping are more important for purposes of interpretation.

If the anaylsis of air samples points towards fire extinction and the temperature of the area has come down to the normal stata temperature and these conditions remain constant for a few months, the stage is ripe for reopening the fire area.

Before the fire area is to be reopened the mine authorities have to inform the DGMS of their intention to reopen the sealed workings. The area is then inspected by a rescue team wearing self contained breathing apparatus after breaking open only one of the fire stoppings just sufficient for entry. Before the rescue team is allowed to enter the area the following precautions should be taken :

(a) The roadways leading to the affected fire area should be thoroughly stone dusted.

(b) The return airway by which gas under pressure from behind the stoppings may travel to the upcast shaft should be free from any possible source of ignition and electric current should be switched off from all the equipment in the return air route.

(c) Water mains and telephones should be extended upto the fire stoppings.

(d) Materials and men for sealing off the fire area and construction of stoppings should be kept ready at hand. If possible haulage track should be extended upto the stoppings or a convenient place near it.

The rescue team marks on the large-scale tracing of the district the information it gathers during inspection in respect of temperature, water logging, roof falls, road blockade, sites of gas samples, position of machinery, condition of ventilation stoppings and doors, etc. Such information and the interpretation of gas samples help the management to take a decision on reopening the sealed off fire area.

If the sealed off area is small the whole area is ventilated by breaking one stopping on the intake air route and one on the return air route, thereby circulating the air current through the district. It is preferable to construct a regulator in the return air stopping of the district to be reopened before circulation of air. The sealed off area should be thickly stone dusted. It is to be noted that host coals or other material may remain buried under ashes and roof falls and on admission of fresh air the fire may be revived. The area has therefore to be kept under observation for indication of smoke or rise of temperature. Samples of air from the return airway of the district should be taken at half hourly intervals and speedily analysed for proper interpretation.

Reopening a sealed off area in stages :

When the sealed off area is large and inspection by the rescue team in one stretch is impossible due to the extent of the area, numerous roof falls, water looging or some other reasons, it should be reopened in stages. (Fig. 7.4).

The procedure to be adopted is as follows :

1. Build an airtight airlock comprising of two steel doors on the outby side of the stopping of proposed intake airway.
2. Establish fresh air base on the outby side of the airlock.
3. Breach only on stopping inbye of the airlock just sufficient to allow the rescue team to enter.

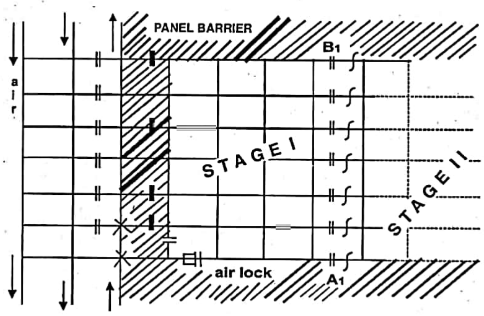


Fig. 7.4 Reopening a sealed off area in stages

4. Permit the rescue team to inspect the area only upto a specified distance, generally 50 to 100 m. The rescue team, in addition to its normal duties, should also note the cross-sections of the roadway where stopping may have to be constructed and mark the cross sections on the rescue tracing.
5. If report of the rescue team indicates condition favourable for reopening part of the area, another rescue team should be sent inbye to inspect a further area, say upto 200m inbye and on receiving a report from the second rescue team a decision should be taken on the total area to be reopened. Construct temporary stoppings to separate the area to be recovered from the area which will not be recovered in the first stage.
6. Break stopping of the proposed return air route after constructing a temporary stopping on its inbye side.
7. Open the door of the airlock to allow only small quantity of air about 20 m³/min, to circulate past the temporary stoppings till the gas percentage falls to about 2 and keep the area under observation for rise of temperature.

8. Construct permanent brick stoppings in front of the temporary stoppings and provide doors in then at A_1 and B_1 Transport of material, etc. can be done by mine workers but rescue teams equipped with self contained breathing apparatus should be kept in readiness at a nearby convenient point.
9. For recovery of the area inbye of the stoppings A_1 and B_1 in stages construct an air lock at A_1 and repeat the same procedure as outlined above.

   An auxiliary forcing fan may be used for ventilation of the recovered area if the return air route is blocked by roof fall machinery or other causes. Use of suction fan is not recommended as the air of the sealed area may be charged with fire damp.

QUESTIONS

1. Describe in brief the various methods of collecting samples of atmosphere from behind a sealed area.
2. From a mine where regular samples of air are taken and analysed the following figures of gas percentage are available.

No. of Sample	Oxygen	Nitrogen	Methane	Carbon dioxide	Carbon monoxide	Location
1.	29.93	79.04	–	0.00	–	Intake
2.	19.89	79.01	0.69	0.40	0.008	Return
3.	19.95	79.04	0.59	0.41	0.013	"
4.	20.10	78.74	0.72	0.42	0.020	"

3. Give the likely results of analysis of mine air samples drawn from behind the stoppings sealing off a free area at the following intervals (i) Soon after sealing, (ii) After a week, (iii) After a month (iv) After six months.
4. (a) In reopening a sealed off fire area in a coal mine what preliminary investigations and arrangements would you make before calling Rescue brigades ?

   (b) What dangers and difficulties are the Rescue teams likely to encounter in the fire area ?
5. A large area in an underground coal mine has been sealed off due to an explosion. State the procedure that should be adopted in recovering the area.

O O O

CHAPTER 8

MINE LIGHTING

A miner working under conditions of insufficient light over long periods not only impairs his efficiency but also developes an eye disease known as nystagmus. Before the invention of miner's flame safety lamp naked lights were used in coal and metal mines. Nowadays electric cap lamps are used by every underground worker in coal mines and in a number of metal mines though naked lights continue to be used in some small metal mines. Flame safety lamps are used by supervisory staff in underground coal mines for detection of methane gas.

A flame safety lamp provides light in a mine without danger of ignitiing inflammable gas and, in the hands of trained workers, it is also a handy and a very convenient device of detecting the presence as well as percentage of firedamp.

Technical terms in lighting and photometry :

Some of the technical terms used in lighting & photometry are given below :

Intensity of light is the relative amount of luminous energy given by any source and is measured in candles or candle power or in candels in (C.G.S. units) (cd).

A light source generally gives different intensities in different directions. Hence candle power or candela does not convey the correct picture unless direction is specified.

Mean horizontal candle power (m.h.c.p) : It is the average candle power of lamp in all directions in a horizontal plane passing through the centre of the source and is usually obtained by rotating, the lamp about a vertical axis.

Mean spherical candle power (m.s.c.p) : It is the average candle power of a lamp in all directions, or the candle power of a uniform source given the same total flux of light. It is directly proportional to the total light given by the lamp and is measured by taking intensity readings in all directions.

Illumination : The illumination, E, at a surface is measured in foot chandles or in meter candle (in the C.G.S. units). One meter candle is the intensity of illumination on a surface 1m distant from a source of one candela. Illumination at a surface is inversely proportional to the square of the distance of the surface from the source of light, and directly proportional to cos θ where θ is the angle between the normal to the surface and the direction of the light rays.

Illumination of a surface (meter candle)

$$= \frac{(\text{Candela of source})}{(\text{distance in m})^2} \times \cos\theta$$

At 2 m distance the illumination would be

$\frac{1}{2^2} = 0.25$ meter candle. A meter candle is also termed a *Lux*

The statement that the illumination at a surface is 4 meter candle implies that it is the same as if it were illuminated by a point source of four international candles placed at a distance of 1m from it. Light is the means; illumination the end effect.

Lumen (lm) : This is the unit of light (luminous flux) emitted by a light source.

Lumens : emitted by a lamp = mean spherical c.p. × 4π

Lux : It is the unit of illumination in S.I. Units. A lux is an illumination of 1 lumen/m².

The minimum amount of light required for reading, writing etc. is 10 lumen/m², *i.e.* the light given off by 10 international candles at a distance of one m from the work. Much more light than this is required for reading without strain. In a factory in every part where persons are working or passing, illumination should be minimum 65 Lux.

Luminous efficiency is expressed in lumens per watt consumed and is from 10 to 20 in modern incandescent lamps, the higher values being for the larger lamps.

Reflection : When light falls upon a surace, part of it is reflected and part absorbed. In the case of a transparent body majority of the light passes through. Only that part of light which is reflected is useful for illumination. A white surface is a good reflector of light and in underground mines, to improve the lighting effect, the following places have to be white-washed.

(a) every shaft inset and shaft bottom or siding and every bypass which is in regular use :

(b) The top and bottom of every haulage plane, every regular stopping place, siding, landing, passbye and junction, except within 100 meters of the face ;

(c) every travelling roadway ;

(d) every room and place containing any engine, motor or other apparatus; and

(c) every first aid station below ground.

General lighting in mines :

General lighting arrangements have to be provided in a mine at the followng places and if electricity is available the lights should be electric.

(a) On the *surface* at the pit top/incline top and in every engine room if natural light is insufficient.

(b) Below ground :

i. at every shaft inset and shaft bottom or siding which is in regular use;

ii. in every travelling roadway normally used by 50 or more persons during any shift.

iii. at the top and bottom of every self-acting incline in regular use.

iv. at every place on a haulage roadway, at which tubs are regularly coupled or uncoupled or attached to or detached from a haulage rope;

v. at every place at which tubs are regularly filled mechanically.

vi. at every room and place containing any engine, motor or other apparatus;

vii. at every place where any pillar is underey traction.

viii. at every first aid station below ground.

Every lighting fitting in underground coal mines has to be of flame proof design.

The standard of lighting laid down in Circular No. 14 of 1964 by the D.G.M.S are as follows :

	Minimum average lumens/sq. ft
Pit bottom	1.5. to 3.0
Main junctions	1.25
Roadways	0.4
Haulage engines and control gear rooms	1.5

Flood lighting of depillaring areas (Deg. 1 gassy mines) 1.5 at floor level.

For depillaring areas Cir. 36 of 1969 by the D.G.M.S. recommends the following arrangements of lighting :

Deg. 1 gassy mines : Four or more 250-watt bulbs cluster if height of working in over 3m.

Deg. 2 or 3 gassy mines or mines having fire : Cluster of 15 to 20 cap lamps placed on a suitable stand, in addition to cap lamps for individual workmen.

FLAME SAFETY LAMPS :

The main safety feature in a flame safety lamp is the wire gaugze and its principle of action is illustrated in Fig. 8.1.

(i) Hold an ordinary iron wire gauze about 30 mm over a laboratory bunsen burner, turn on the gas and ignite it below the gauze. The flame burns only *below* the gauze but does not pass through it. If the gauze is brought gradually to the mouth of the burner, the flame is extinguished but, instead, if held slightly higher for some time, the gauze becomes red hot and the flame passes through it and appears below as well as above the gauze.

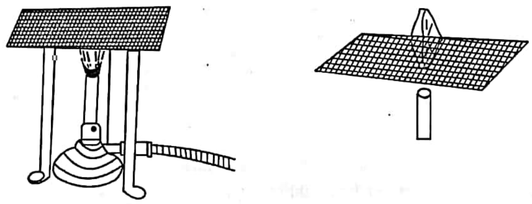


Fig. 8.1 Illustrating the principle of wire gauze.

(ii) Now hold the hot gauze is the same position, about 30 mm above the burner, turn off the gas, and allow the gauze to cool for a few minutes. Turn on the gas and apply a flame *over* the gauze. The gas burns over the gauze as a flame but the letter does not pass below the gauze; if the gauze is lifted upwards carefully high enough, the flame is extinguished.

The explanation for this behaviour of the gas is that iron wire gauze is a good conductor of heat and allows the gas to pass through it but conducts the heat of the flame away so quickly that the gas which is not burning on one side of the gauze fails to reach ignition temperature even though some gas is burning on the other side of the gauze. The gauze, thus, allows the gas to pass through but not the flame. However, if the flame continues to heat the gauze for a few minutes, all the heat of the red hot gauze is not conducted away, and the gauze allows the flame to pass through. This is the principle behind the safety provided by the wire gauze in a flame safety lamp and it is apparent that such lamp is safe as long as it is not allowed to get unduly hot. A copper gauze is a better conductor of heat than a wiregauze of iron but copper wire gauze is costlier and burns away comparatively early.

The principle of wire gauze explained so far is made use of in the construction of flame safety lamps which are better known as gas testing flame sâfety lamps. These are manufactured in the country by Mine Safety Appliances Ltd., J.K. Dey and Sons and one more company. They are used for accumulation test and percentage test of methane by the supervisory staff in coal mines. The flame safety lamps manufactured by J.K. Dey and Sons are sold under the trade name **Velox.**

The company, J.K. Dey and Sons, manufactures three types of gas testing flame safety lamps.

1. GL – 5
2. GL – 50
3. GL – 60

Of these GL – 5 is used for accumulation test and the other two for percentage test. The three lamps differ from one another in some constructional details, described later.

GL-50 lamp consists of three separable sections which can be screwed together for complete assembly and use.

1. Lower Section : This comprises a fuel vessel fitted with a burner, a round wick passing through the burner, side oil-filling arrangement, and a screw spindle flame adjustment device. The oil-vessel has retchet teeth at the top and provision for magnetic locking.

2. Middle section with partly upper section : This consists of a composite lower flange which is screwed on the fuel vessel and a composite middle ring assembled with five steel rods to provide bonnet and chimney at top. The steelrods connecting the lower flange and the middle ring also protect the glass. Separate air inlet and outlet ports are provided to avoid mixing of fresh inlet air with gases of combusion. This improves air circulation and results in sharp reactions of methane gas with the flame.There is only one cylindrical thick toughened glass which forms part of the middle section. During assembly the glass is provided with asbestos gaskets at the lower end, and also at the upper end below the outer wire gauze. The asbestos gaskets make the glass assembly air tight at the top and at the bottom and should never be omitted during assembly.

3. The upper section : This consists of a bonnet and chimney with hood provison of top feed device, i.e. for enabling mine air near roof to enter the lamp from the top. It also has two wire gauzes, each of 20 mesh.

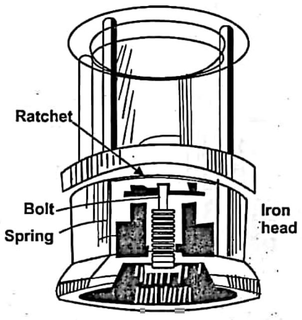


Fig. 8.2 One design fo magnetic lock on a flame safety lamp (not of the type used on Velox lamps).

The bonnet protects the wire gauzes and is provided with a handle for holding the lamp. The hot gases of the flame rise by convection to the top of the gauzes and through them and the oulet holes of the bonnet, to the atmsophere. The fuel used is a solvent spirit or motor spirit, *i.e.*, petrol. The solvent spirit is SBP 55/110 or ESSO solvent No. 1425 (in GL-5 lamp the fuel used is colourless K.oil. That lamp is used only for accumulation test).

During assembly the upper section is assembled with the middle section. The combined assembly is then screwed onto the lower section and magnetically locked. When the lamp is properly assembled, there will be no sound of any loose components if it is shaken by hand.

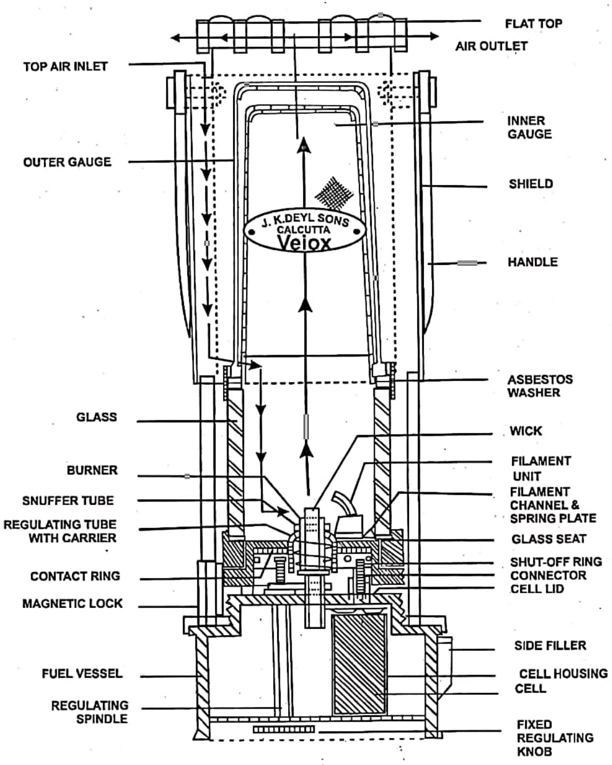


Fig. 8.3 Velox GL-60 flame safety lamp. Courtesy of J.K. Dey & Sons.

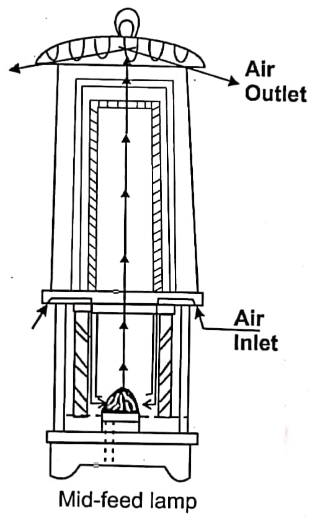


Fig. 8.4 Mid-feed lamp

The principle of magnetic locking is illustrated in Fig. 8.2 (in Velox lamps the magnetic locking arrangment is not exactly as shown in the figure but differs slightly). A spring loaded steel bolt is housed in a tubular body, fitted and soldered with the bottom flange of the midle section. The lock bolt passes through the collar into notches on the oil vessel and when the middle and top sections are fitted on the oil vessel by screwing, the lock bolt prevents their unscrewing by the ratchet construction at the top end of the oil vessel.

To unlock, the top of magnetic locking device is placed below the top pole of magnet unlocker in the lamp cabin. The lock bolt is pulled by the magnet and the base of the lamp can then be unscrewed. The magnetic locking arrangement is so designed that ordinary magnet cannot unlock the lamp. The magnetic locking is an important safety feature of flame safety lamps.

Weight of GL-50 lamp with fuel is 1.6 kg.

Sequence of components in Velox lamp, GL-50

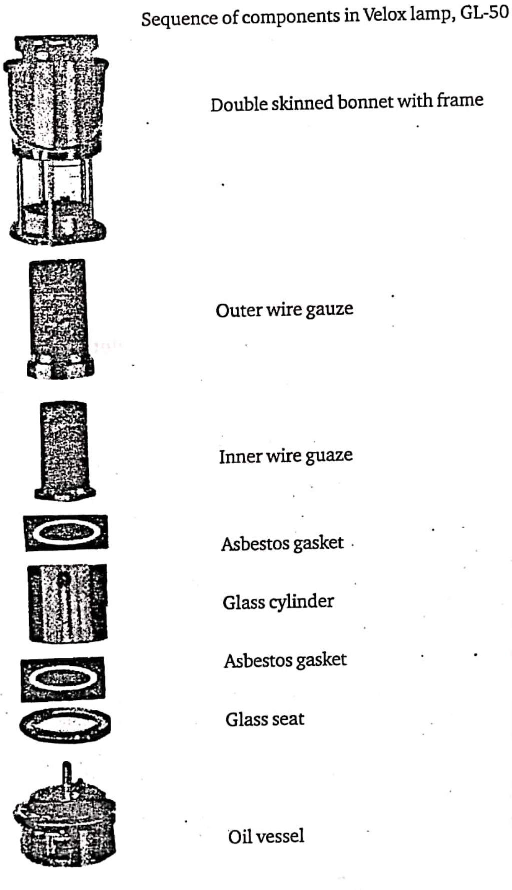


Fig. 8.5

The purpose of the gauze is to prevent the passage of flame from the interior to the exterior of the lamp so that even if the gas is ignited within the gauze, no external ignition will occur. This holds good as long as the gauze itself does not become red hot. The wire gauzes loose their effectiveness if covered with soot or if they are broken on distorted or wire is displaced or becomes thinner due to rust. They should be cleaned with fibre brush and checked daily in the lamp cabin. Defective or repaired components of a safety lamp should not be used and only the genuine spare parts supplied by the manufacturers should replace the damaged ones keeping the important safety aspect of the lamp in mind.

A flame safety lamp is used mainly for accumulation and percentage test of firedamp. It is an essential equipment of a mining sardar in coal mines.The lamp is to be lighted at the lamp cabin only if it is not provided with relighting arrangements. After lighting proper assembly, and magnetic locking, it is held before a compressed air jet of 6 m/sec velocity. If the lamp is properly assembled the flame should not unduly flicker or be afffected in any way. At the pit/incline top, before the lamp is allowed to be taken underground, a person deputed to check safety lamps and contraband, tests it for visual defects, possibility of unlocking and unscrewing and blows air by mouth to test for leakages. Modern flame safety lamps can withstand an air velocity of 15 m/s.

GL – 5 differs from GL – 50 in following respects.

1. It (GL-5) has got 2 cylindrical glasses surroundings the flame.
2. It is used for accumulation test.
3. It uses colourless K Oil.
4. It has mid-feed arrangement, i.e. inlet air enters the lamp at midheight and hot at the top.

GL-60 lamp differs from GL-50 in some constructional features through external appearnce of both is alike. Both use double wire gauzes, only single glass and have top feed arrangement. But GL-60 is more sophisticated. The differences are :

1. GL-60 has additional bottom feed device. This is incorporated in the lower section of the lamp of allow free air at the base of the flame. This makes relighting easy and also results in sharp reaction of methane gas in general body of air, as well as detects CO_2 nearest to the floor of a working place. When the bottom feed is closed, only the top-feed arrangement; works.

If bottom-feed is open, both the top-feed and bottom feed admit air. Movement of the level of glass seat for top-feed and bottom-feed is indicated by an arrow on circular brass ring supporting the glass. The bottom air feed with shut off device should be opened to ensure an instant relighting yielding steady even burning light. During gas testing, however, the lèver should be placed at opposition.

2. Automatic flame extinguisher is incorportated as a quite effective device to get the flame automatically extinguished if the fuel vessel of the lamp is accidentally removed during use due to failure of magnetic locking system.

3. The fuel vessel is full of cotton absorbent, eliminating possibility of fuel spilling.

4. It has a self-contained relighting mechanism which enables the user to relight the lamp himself.However, it cannot be relighted if only colourless K. oil is used as fuel.

5. Wt. with fuel is 1.7 kg.

The relighting arrangement in GL-60 lamp is as follows :

In the oil vessel there is a housing for a dry battery consisting of two cells, each 1.5V in series. On the glass seat are fitted a filament unit, a filament channel and a spring plate, a snuffer tube and a contact ring. A regulating spindle and knob regulates the wick and the height of the flame. For relighting spindle and knob regulate the wick and the height to the flame. For relighting, hold the lamp at chest level and turn the regulating knob clockwise. This causes the snuffer tube to go down thereby exposing more length of the wick and at the same time, glowing the filament to red hot by completing the electric circuit of the battery. The red hot filament ignites the vapour of fuel in the wick into a flame. As soon as the flame is produced the regulating knob should be turned anticlockwise quickly and the filament ceases to glow.

The fuel recommended to carry out the percentage test is ESSO Solvent No. 1425 or its equivalent, it is manufactured in our oil refineries but its scarcity has resulted in use of petrol as a fuel for flame safety lamps with the approval of DGMS. The battery cells used are Eveready 935 or its equivalent. They are not rechargeable and they must be kept out of the cell housing when the lamp is not in use. If the battery voltage is below 2.3 it should be discarded. The battery can relight the lamp 600 times in the hot lamp by the use of a graded filament. To achieve smooth ignition the wick should be kept burning for a minute or two in the lamp cabin with the help of a match or lighter as the cold wick takes unnecessary long time to light when the lamp is intended for use. For best result pour a few drops of fuel above the wick and then screw the fuel vessel to the top part of the lamp.

Relighting of the lamp becomes troublesome in the following cases.

1. Abnormal dropping of battery voltage to 2.3 or less.
2. Accumulation of coal dust on old burnt wick and sludge thereon from the oil vessel.
3. Damp wick.
4. Use of odd size wick other than the prescribed one.
5. Excess or short flowing of fuel on wick top.

Relighting of internal ignition lamp burning with solvent spirit or motor spirit has always been troblesome due mainly to the continued vapourising and excessive accumulation of fuel vapour in the lamp which sometimes defers ignition even in full glowing filament to dispel efficiency this surplus vapour, get fresh vapour by swinging the lamp side to side or blowing at the air inlet ports at the top of the lamp for smooth relighting and steady even burning light.

The plate shows flame safety lamp model Velox GL-50 of J.K. Dey & Sons and the gas caps at different percentage of methane in mine air with ESSO Solvent No. 1425 and petorl as fuel. The GL-50 is permitted by DGMS for gas testing in Deg. 1 and Deg. 2 gassy coal mines but Velox GL-60 is permitted for use in coal mines of all degrees of gassiness. GL-60 and GL-50 lamps have been exported to foreign countries like W.Germany and others.

The construction of flame safety lamps is robust and they have to withstand strength tests before approval by DGMS. Gas testing flame safety lamps have to conform to 'ISI' specifications NO. I.S. 7577 of 1975 which are framed for detection of combustible gases (gas content 0.5–4%) mainly composed of methane in a mine or other places. Some of these specifications.

i. Lead rivet locking arrangement shall not be used.

ii. The guaze shall have a total heat radiation area of not less than 155 cm^2.

iii. Glass cylinder shall be 4–5 mm thick.

iv. The pillars of M.S. rounds must be at such distances that a straight edge, when touching the two pillars, will not touch the glass.

v. Lamp shall be capable of burning normally and without extinction or undue flickering in air current of 15 m/s or less velocity.

vi. The lamp in normal positions shall be dropped 5 times from a height of1m onto a hard wooden board 30mm thick laid on a concrete floor. (Dropping height measured from the bottom of the lamp) Glass should not break and the lamp or any of its components should not be damaged.

vii. Performance test. The length and condition of the flame produced shall be as shown in Table 1.

TABLE 1

gas %	Minim. length of blue flame mm	Condition of Blue Flame
0.0	–	A slight cobalt-blue lined orange-yellow flame seen near the top of standard flame (flame length of about 2.5mm above upper edge of mouthpiece) A faint light is seen along cobalt-blue line.
1.0	7.0	Scarcely any formation of blue flame is seen as the flame colour is light and hence it is difficult to measure the length.
1.5	8.0	Blue flame becomes a little more distinct, especially the lower part turns somewhat clear.
2.0	9.0	Standard flame grows larger, and blue flame becomes distinguishable, but the top is invisible.
2.5	10.0	Colour of blue flame becomes clearer, but the top is still indistinct.
3.0	11.5	Top of flame becomes barely visible and blue flame seen clearer.
3.5	14.5	Blue flame clearly visible.
4.0	20.0	Blue flame becomes extremely clear and highly sensitive to a slight change of gas content.

Constitution of normal flame of lamp :

When the normal flame of an oil-lamp is examined, it is found to consist of three principal and fairly well defined zones, although these merge gradually on into another. The zones are : .

(A) *Inner zone of non combustion,* blue in colour and consisting of unburnt vapour given off by the fuel and unmixed with air. This is the coldest zone and is non-luminous.

(B) Intermediate zone of incomplete combustion, white or yellow in colour and consisting of partially burnt vapour and solid incandescent (white hot) particles of carbon. This is the zone that gives the maximum light and is also very hot, especially near the apex, above B in the sketch. (Fig. 8.7).

(C) Outer zone of complete combustion, forming a transparent fringe or mantle surrounding the flame and consisting of burning carbon monoxide, hydrogen, and carbon particles which have passed out of the yellow zone into the outer zone. Here combustion is completed and carbon dioxide and water vapour are formed. The mantle generates great heat but gives little light, being almost non-luminous.

Accumulation test and percentage test :

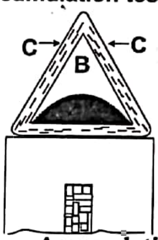


Fig.8.7

In a coal mine where methane gas is likely to be present, it is tested with the help of a flame safety lamp in two ways.

1. Accumulation test.
2. Percentage test.

Accumulation test :

The purpose of this test is to ascertain if there is any accumulation of gas in places where it may be suspected or is likely to accumulate. In a mine, if the mining sadar finds accumulation of gas at any place, he has to report the matter to the overman who should take steps for determining its percentage and its removal. To test for accumulation, switch off the cap lamp, raise the flame safety lamp cautiously with normal size of flame, or a flame only slightly reduced, and watch its behaviour; if it elongates i.e. if it spires or jumps, the percentage of gas can be taken as nerly 3% or more. No efforts should be made to raise the flame safety lamp higher than is necessary to test for accumulation because this results in keeping the flame in richer mixture of methane and air which may explode inside the lamp and extinguish the flame. Even if the mixture is not explosive the gas will burn inside the lamp and it may produce CO_2 which will extinguish the flame. It is unnecessary to conduct the percentage test when the flame spires up in a safety lamp as it is clear that the gas percentage is not less than 3.

If by mischance, firedamp begins to burn within the gauze, it should in no circumstances be allowed to continue to burn. The examiner should shelter his lamp from the air-current, hold it near the floor, and retreat carefully to fresh air. If this is not possible, he should smother out the flame, *e.g.* by covering the air-inlet holes with a handkerchief.

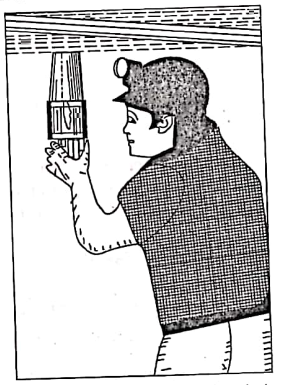


Fig. 8.8 Conducting percentage test

Percentage test :

To conduct percentage test for methane with flame safety lamp remove all bright light in the vicinity and switch off the cap lamp. Lower in flame of the safety lamp with the regulating knob till there is a continuous blue line (actually, curved line) across the top of the flame just above a speck of white (or yellow) light as shown in the plate. This should be done, not at the place where gas percentage is to be detected, but at a place nearest to it and free from gas. When firedamp present in the air at the spot of a non-luminous flame (bluish) which varies in height depending on the percentage of the gas. The size and height of the non-luminous flame produced by the burning of the gas (gas cap) also depends on the size of the wick and the quality of fuel used. Hence for determining the gas percentage, the lamp to be used, the size of wick, and fuel have to be standardised.

An oil flame safety lamp, through a convenient, handy and inexpensive device for detection of firedamp has certain limitations.

1. It can be used only by persons trained for the purpose.
2. Even in the most experienced hands, it can detect fire damp percentage not below 2%.
3. It can be used for detection only in accessible process within the reach of the person resting for gas.

If there is shortage of oxygen, flame of the oil safety lamp will reduce in size and will be completely extinguished if the oxygen percentage is 17 or less. Minimum 14% oxygen is required in air for supporting human life.

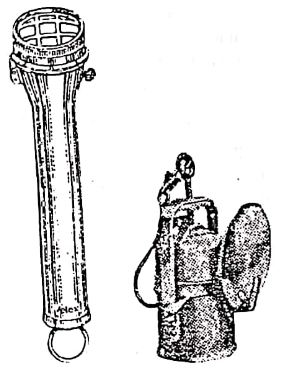
If blackdamp or CO_2 is present in air, the flame of oil flame safety lamp diminishes in size and will be extinguished if CO_2 percentage is 3 or more.

Acetylene portable hand lamp :

A commonly used portable lamp in metal mines is the acetylene hand lamp. The lower container contains calcium carbide and is provided with a burner. The upper container contains water which flows to the lower container, drop by drop, and is regulated by a valve operated by hand. The water acts on the calcium carbide to generate acetylene gas which can be ignited at the burner to give a white flame of good intensity. The lamp requires less oxygen for combustion than oil lamps. It does not provide an early indication of the presence of black-damp as compared to the oil lamps. It weights 1 kg without water and carbide. Carbide lamps can withstand an air velocity of 5 m/s but for safe working the air velocity should not exceed 2.5m/s where carbide lamps have to be used.

Flameproof safety torch :

Safety torches intrinsically safe in inflammable atmosphere, and using 2-3 torchs are used by senior supervisory staff in coal mines. One such torch is shown in Fig. 8.10



Valox flame proof safety torch
Acetylene portable hand lamp.
Fig. 8.10

1. The torch is locked in the lamp cabin and can be opened only by special tools. Unlocking and opening in a mine is impossible.
2. The glass at the mouthpiece is toughened Acrylic disc (unbreakable type).
3. The glass is protected by a grill of 2mm wide brassplates in the mouthpiece.
4. The bulb is protected by a spring placed over. If the glass breaks, the spring is thrown out and the bulb loses its contact with the battery terminal thereby ceasing to burn. Exposure of the hot filament of the bulb in case of breakage of the latter is out of question.

5. There is a spring cushion, a sort of shock absorber, between the bulb and dry cell which prevents crushing of the bulb against the weight of the dry cells in case of accidental dropping of the torch.

ELECTRIC CAP LAMPS

The electric cap lamps used in our mines are the popularly known Oldham cap lamps and also the cap lamps manufactured by Mine Safety Appliances Co., Ltd. In both types of cap lamps the entire cap lamp unit consists of a 4-V lead-acid battery (re-chargeable type), a lamp which can be hooked to the helmet and a connecting cable. The lead acid battery consists of two cells.

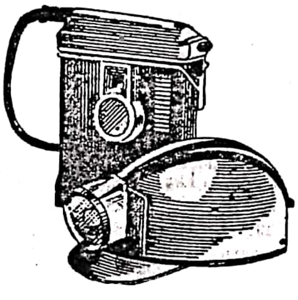


Fig. 8.11 An electric cap lamp.

(Courtesy of M.S. A. Co. Ltd.).

In the Oldham cap lamp unit each cell of the lead acid battery consists of a number of composite lead-antimony tubes or plates carrying the active materials and immersed in a 30% solution of sulphuric acid (H_2SO_4) and distilled water. The positive plate in each cell is of tubular construction, the negative plate is of pasted flat type, and the insulating separators are of Sponac (a highly absorbent type of wood) which absorbs about 85% of the total acid in the cell, so rendering the battery virtually unspillable.

In the fully charged condition, the active material in the positive plates in brown lead peroxide (PbO_2) and in the negative plates, it is grey spongy lead (Pb). During discharge, both positive and negative plates change partly into lead sulphate (Pb). During discharge, negative plates, it is grey spongy lead (Pb). During discharge, both positive and negative plates change partly into lead sulphate ($PbSO_4$) with the liberation of water. The reaction may be set out as follows, (read from left to right for discharge and vice-versa for charging) :

$$\underset{\text{Positive}}{PbO_2} + \underset{\text{electrolyte}}{2H_2SO_4} + \underset{\text{negative}}{Pb} = \underset{\text{positive}}{PbSO_4} + \underset{\text{electrolyte}}{2H_2O} + \underset{\text{negative}}{PbSO_4}$$

In the charged condition, the sp. gr. of the acid is about 1.260 but this falls during discharge, to about 1.180. During charge, the reverse occurs and the sp. gr. rises again. The end of charge is marked by the liberation of oxygen and hydrogen from the electrolyte, a condition known as "gassing". The two vent-holes in the front of the battery allow the gases to exit. These should always be kept free from obstruction to enable the battery to function correctly. Periodically; once in 7-10 days, the cells must be topped up with distilled water to replace the loss; acid should not be used for topping up. The normal voltage of a single lead-acid cell is 2 volts so that two cells must be connected in series to give a 4-volt lamp. Actually, at the beginning of discharge, the lamp voltage is about 4.4 volts but this falls progressively during use to about 3.6 volts (i.e. 1.8 volts per cell) below which it must never be allowed to fall in order to avoid permanent sulphating of the plates.

The headpiece of the Oldham lamp has a shell of moulded plastic fitted with a bulb, a knob-type switch, a reflector which may have either a polished or a matt surface, a cap hook, a charging contact and a thick armour-plate glass. The bulb is krypton filled for high efficiency and is rated at 4-volt, 0.67 amp. giving light output of 30 lumens. The life of the bulb in the pit is about 500 hours. The headpiece is virtually water tight, if well maintained. As an important safety feature, a cartidge type fuse is incorporated under the battery cover and is situated in the circuit between the cable lead and the negative battery terminal as a safety measure to guard against excessive current flow in the event of short circuit. It is rated to blow at 4 amps. This has adequate margin of safety as it has been established that at 4 volts, a current of 26 amps is required to provide a spark of sufficient energy to ignite an explosive mixture of methane and air. Silver fuse wire is employed.

In the lamp charging room the charging rack accommodates 100 lamps on a 102-type charger. All the lamps are connected in parallel and are charged on a constant potential system, the low voltage D.C. power required (at about 5 to 6 volts) being obtained from a transformer-rectifier unit. The correct voltage for charging with Oldham constant potential charger is 4.8 as indicated on the panel voltmeter. On the charging rack above each battery position are charging contacts to which the head-piece is applied when the lamp is placed on charge and also a meter which indicates whether or not charge current is flowing.

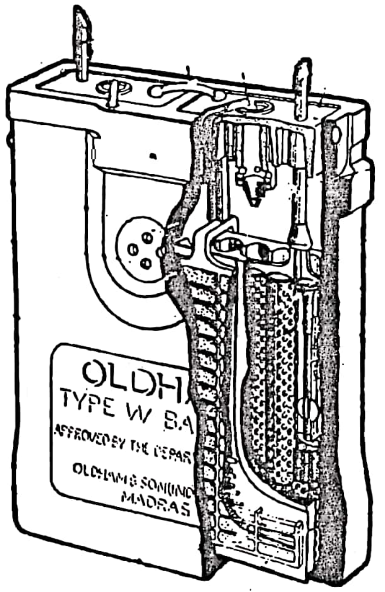


Fig. 8.12 Sectional view of the battery of a cap lamp.

The negative charging contact is a key mounted on the headpiece board. To place a lamp on charge the headpiece is fitted over this key and turned clockwise through 180 degress. The positive charging stud on the lamp then makes contact with a spring clip assembly, also mounted on the lamp then makes contact with a spring clip assembly, also mounted on the headpiece board. The rotation of the headpiece at the same time brings the key into contact with the negtive contact, within the headpiece, and the charging circuit is complete. The current taken varies according to the state of charge of the battery gradually becoming less as the battery becomes more fully charged. Each lamp takes only the current necessary for a correct charge and the low voltage used eliminates any danger of shock to the man placing his lamp on charge. When the lamp is fully charged, the charging curent to the individual lamp is automatically cut off, though other lamps continue to get charged on the same charging rack (Fig. 8.13).

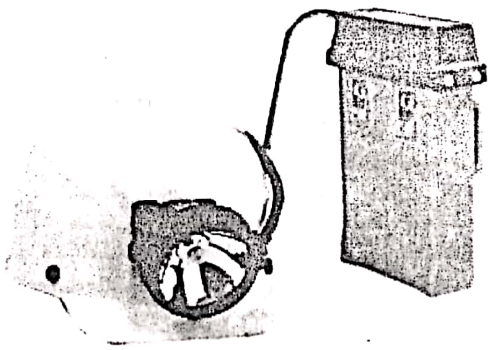
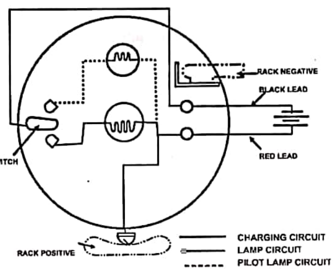


Fig. 8.13 Placing a cap lamp for charging in a self servicing system. After placing the lamp on the key as shown, turn the headpiece clockwise through 180°.

The Oldham lamp (and also the M.S.A. cap lamp) is designed for a "selfservice" system in which the miner himself puts high lamp on charge in the lamp room at the end of the shift and removes it himself after 12-16 hours. charging takes place through cotact on the headpiece, as described carlier, and it ceases automatically when the battery is fully charged.



Diagrammatic representation of headpiece connections.

Cap lamps are also powered by batteries having alkaline cells though such batteries are not manufactured in our country for the cap lamps used genrally by miners. An alkaline cell consists of a number of composite plates or tubes of thin perforated nickel steel containing the active material and immersed in a 20% solution of potassium hydrate (caustic potash, KOH) having a constant S.G. of about 1.2. The active material in the positive plates is nickel hydroxide, and that in the negative is iron or cadmium oxide or a mixture of the two. All plates are insualted from each other and from the steel container.

The chemical reactions are given by the equation.

$$2NiO_2+2KOH+Fe = Ni_2O_5+2KOH+FeO$$

+ Plate – Plate + Plate – Plate

(Read from left to right for discharge; from right to left for charge).

During charge, OH radicals pass from negative to positive, whilet the reverse takes place during discharge. A single alkaline cell gives an E.M.F. of about 1.25 volts so that three cells must be connected in series to give a 3.75-volt lamp. An example of the alkaline battery is the C.E.A.G. battery manufactured in Germany and England and used for some of the cap lamps abroad.

In appearance the battery with alkaline cells is similar to the lead-acid battery but it has some advantages over the latter; viz. (1) working life (5-6years), (2) lower maintenance cost and (3) it can withstand adverse treatment like overcharge, excessive discharging, short circuiting or remaining unused for long period. The lead acid battery, on the other hand, has a low nitial cost compared to the alkaline battery, higher electrical efficiency and lower cost for replating. The normal life of a lead acid battery is two years, depending on the care during its use, and as it gets old the light available gradually diminishes. When the battery reaches that stage it is time for sending it to the manufactures for re-plating. There can be other reasons for poor light from the battery and this should be investigated by the lamp room incharge before deciding to send the battery for replating.

Audio-visual alarm lamp :

Mine Safety Appliances Limited (M.S.A) is marketing an alarm lamp which is a portable audio-visual warning system. (Fig. 8.14). It is called Britelite Audio-Visual Alarm lamp and is developed by adding a blinkering red light and intermittent hooter to the basic hand-lamp manufactured and

marketed by M.S.A. The product can be used for normal working light and also, where required, to give warning. The blinkering light and hooter can also work individually. The make and break arrangements for the hooter and the flashing red light is through a sealed electronic module which has a very long life. The alarm lamp has been approved by D.G.M.S. Weight of the lamp is 1.5kg. The battery used in Exide Triclad F-2 which can be charged from the standard battery charging rack used for miner's cap lamps.

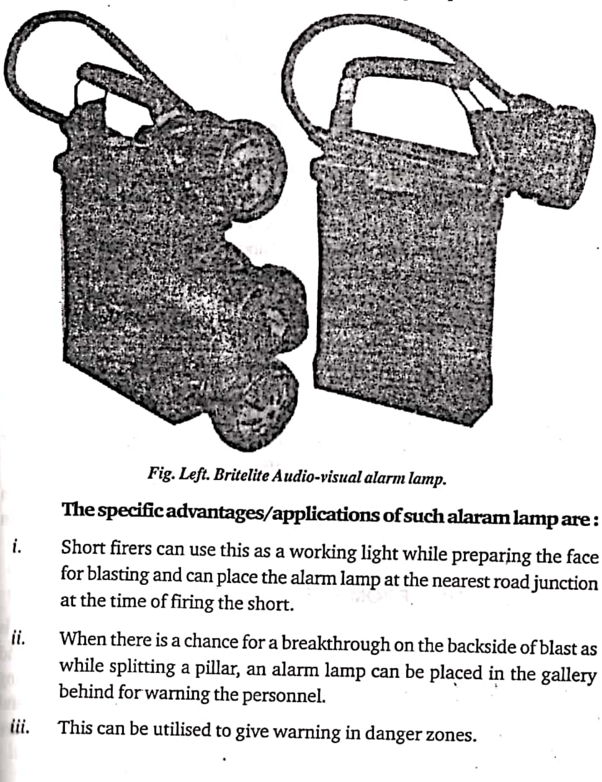


Fig. Left. Britelite Audio-visual alarm lamp.

The specific advantages/applications of such alaram lamp are :

i. Short firers can use this as a working light while preparing the face for blasting and can place the alarm lamp at the nearest road junction at the time of firing the short.

ii. When there is a chance for a breakthrough on the backside of blast as while splitting a pillar, an alarm lamp can be placed in the gallery behind for warning the personnel.

iii. This can be utilised to give warning in danger zones.

Oldham Methalarm :

METHALARM :

Fig.

By incorporating an alarm into the cap lamp headpiece, it is possible for miners to be automatically alerted to the presence of any unacceptable level of methane in the mine air. The alarm is given by a modulation of the main light beam from the lamp headpiece the alarm device, which can be pre-set to operate at any chosen threshold between 1 and 2.5% methane (with an accuracy of ± 10 percent) is an integral and additional weight is only 80g. The sensing unit consists of a matched pair of pellistors and an electronics module which together form a wheatstone bridge plus control circuitry. The presence of methane causes a change in temperature of one of the pellistors resulting in a change of resistance which imbalances the wheatstone bridge. At a pre-chosen methane concentration, this imbalance triggers the control circuitry and modulates the current to the main bulb. The current can be either switched off or reduced, depending on whether an on-off or a bright-dim modulation is required. In either case the frequency of modulation of four Hz. The detector circuit is intrinsically safe.

This Methalarm has been jointly developed by Oldham UK. Oldham France and Hawker Siddley Dynamics Engineering Limited, all comparines in the Hawker Siddley group.

POWER LIGHTING FROM ELECTRIC MAINS :

The light provided by flame safety lamps and electric cap lamps is inadequate for general lighting and proper illumnation of the underground mines. Electric lamps should, therefore, be used for purposes of general lighting and at places where more illumination is required. When providing for general lighting care should be taken to see that workers do not have to face the glare

of the lamps, deep shadows are not cast by the lights and there is sufficient illumination. An illumination of 20 to 40 lumens per m^2 at the pit bottom and 15 to 20 lumens per m^2 at haulage junctions is considered adequate. The desired results are obtained by (i) use of lamps of sufficiently high candle power but low intrinsic brightness. (ii) by closer spacing and (iii) by white washing the area. Also see "standard of lighting".

There are some restrictions on the use of power for electric lighting from the mains. The power can be used at voltage exceeding 110, *i.e.*, lights at 220/250 volts cannot be used. Lighting transformers should provide the voltage by stepping down from the usual 550/440 or 3300 volts and the neutral point of the secondary transformer should be earthed. All the lights and lighting fitting should be in flameproof enclosures and there are only 2 or 3 wellknown companies in India who manufacture such flame proof lighting enclosures. Though it is not clearly laid down in the Electricity Rules, it is advisable not to use electric lights from mains at or within a distance of 100m from a longwall coal face due to risk from emission of firedamp. Electric light can, however, be used in the main return of a gassy mine (Degree-II & III also) if the lighting fittings are flameproof, as already stated.

Electric discharge lamps and fluorescent tubes for mine lighting

At atmospheric and higher pressures the resistivity of gases is very high, and a breakdown of this resistivity by a sufficiently high voltage causes a spark or arc. If the pressure of gas in a closed space is reduced to a very low value, not only is the voltage required to break-down the insualtion of the gas reduced, but, when this breakdown occurs, the current passing through the gas does not form an arc, but forms what is called a discharge. The light produced by the current is not localised (unlike the light one sees in a filament or arclamp), but comes from the whole of the rarefied gas through which the current passes. The colour of the light given by the discharge depends upon the gas from which it takes place, and this colour can be varied by mixing different gases. A discharge in mercury vapour produces what are called ultra-violet rays.

An electric lamp for which this phenomenon is used is called a discharge lamp. Lamps of this kind containing mercury or sodium vapour are used to some extent for street lighting and for laterior decoration. The familiar lighting tube is a discharge lamp and is called *fluorescent discharge tube or* *phosphorescent discharge tube.* It consists of a long glass tube coated on the inside with a fluorescent material and having tungsten electrodes at each end. When the tube is switched on an electrical discharge takes place across the electrodes. The tube is filled with argon gas and mercury vapour, but it should be remembered that a mercury vapour lamp is a quite different type of a lamp of high wattage giving greenish light. The colour of light available from a fluoresent tube can be controlled by a suitable choice of the composition of the fluorescent or phosphorescent coating on the inside of the tube. The fluorescent tube has high efficiency, the lumen output being as much as $3\frac{1}{2}$ times that of the incandescent filament bulb. This is the reason for the popularity of the fluorescent tubes for lighting but the control equipment for these lamps is more elaborate and they need more maintenance. Moreover these tube lights are not yet manufactured in flame proof design in our country. For Deg. II and Deg. III gassy coal mines, they are not permitted to be used underground, but in Deg. I gassymines they may be installed with permission from Director of mines safety. Some Deg. II gassy coal mines have installed imported fluorescent tubes of flameproof design. For underground lighting, the tubes have an advantage of reduction of glare and they do not produce strong shadow effects but they suffer from the disadvantage of flicker.

They can be operated on 100 Volt or 220/250 Volts (on the surface only). The wattage of these tubes is low. The most common industrial fitting takes two tubes of 1.5m length giving a lamp wattage of only 160.

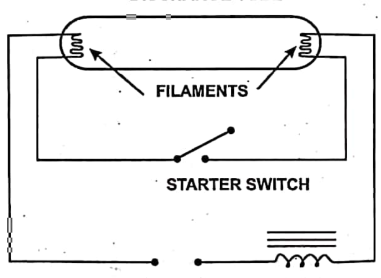


Fig. 8.15 Phosphorescent tube light.

QUESTIONS

1. Give definitions of the following terms :

   (i) *M.H.C.P,* (ii) *Candela,* (iii) *Lumen,* (iv) *Metre candle.*

2. Name the places where general lighting arrangements have to be provided in a mine at the surface and below ground.

3. State the standard of lighting at various places in a mine, as required by the Circular of D.G.M.S.

4. (i) Describe with illustrations the principle of safety resulting from a wire gauze in an oil flame safety lamp.

   (ii) What points should be kept in mind when assembling a flame safety lamp in the lamp cabin.

5. Describe the manner of conducting an accumulation test and percentage test for methane gas with the help of a flame safety lamp.

   What are the limitations of a flame safety lamp as gas detector ?

6. State the factors which make an electric safety torch safe in a gassy mine.

7. What care should be taken during the maintenace of a miner's cap lamp and its battery ? What precautions should the user take for the safety and adequate life of the cap lamp ?

O O O

CHAPTER 9

MINER'S DISEASES AND DUST HAZARDS

A worker working in mines many have to face some occupational health hazards mainly due to environmental and working conditions. The main diseases are :

1. Nystagmus,
2. Ankylostomiasis,
3. Pneumoconiosis,
4. Silicosis,
5. Asbestosis,
6. Siderosis

(3–6) resulting from dusts

Nystagmus :

This is peculiarly an underground miner's disease. The term nystagmus is applied to disease in which the muscles and nerves of the eyes are affected and there is an abnormal movement or oscillation of the eyeballs. This disease is caused by working over a number of years in places of insufficient light. Where naked lights are used, as in metal mines, the incidence of nystagmus among miners is low. Electric cap lamps have also brought down the incidence of nystagmus in coal and metal mines. Before the introduction of electric cap lampsin coal mines, the older type of flame safety lamp with less than 0.6 candle power had resulted in nystagmus among a large mumber of coal mines. It is stated that a miner suffering from nystagmus cannot see the gas cap in flame safety lamp. The supervisory staff in coal mines are therefore required to undergo periodical eye testing once in 5 years. The remedy to avoid the disease lies in proper illumination as per standards stated in earlier chapter.

Ankylostomiasis :

Ankylostomiasis or miner's anaemia is practically the same disease as "hookworm disease" and is caused by a thread-like blood sucking worm which enters the body through the skin. Miners working in insanitary conditions and cutting coal, standing in dirty water with bare feet over long hours, may be affected by this disease. The symptoms are pain in stomach, loss of apetite, constipation, followed by diarrhoea and dysentry. A person seriously affected looks anaemic; positive knowledge is obtained by examining the stools for hookworm eggs.

Dust hazards in mines :

Before appreciating typical diseases arising from inhalation of dust in mines, one should understand the dust hazards. The hazards of coal dust as a potential cause of explosion has been described in the chapter on explosions. Airborne dust of coal and other rocks in mines have harmful physiological effects. It is now well established that the incidence of pneumoconiosis, silicosis, etc. depends upon

1. The period of exposure to dusty surroundings, and
2. Nature and concentration of the dust.

The dustiness of the air, i.e. the quantity of dust contained in it is stated in two ways :

1. As the number of dust particles per cm^3 of air; this method is known as the dust count method.
2. as the number of mg of dust per m^3 of air; this is known as the weight or gravimetric method.

The idea of dustiness of a surrounding can be formed from the following figures obtained after a number of observations :

Dwellings	.....	about 1.5 mg/m^3
Stone crushings sites	.....	about 22 to 45 mg/m^3
Cement works and ore treatment plants	.....	about 130 to 200 mg/m^3
At chutes during coal loading by conveyors	.....	about 5 to 10 mg/m^3

For conversion of weight standars to dust count standard, it is accepted that 1 mg/m^3 corresponds to about 200 particles (upto 2 μ across) per cm^3 (μ = 1 Micron = 1/1000 mm).

According to a Russian text book the dustiness of a place may be considered as follows :

at dust contents less than 1 mg/m^3, the air is not dusty.

do	do	5 mg/m^3, moderately dusty.
do	do	10 mg/m^3, dusty.
do	do	20 mg/m^3, very dusty
do	do	100 mg/m^3, extremely dusty.

The persons worst subjected to dust hazards in coal mines are the operators of coal cutting machines, cutter-loader machines, the drillers, loaders, blasters and conveyor attendants at the loading chutes. In the metal mines the rock drillers are the persons worst affected.

Dust control by the respiratory system :

The first obstruction encountered by the dust particles in the inhaled air is from the hair and mucus in the nose where most of the particles greater than 10μ in dia. are arrested. The remaining coarser particles 10 μ and above in dia. are caught by mucus in the throat and respiratory passages. All such coarse particles are expelled by coughing.

As the air travels further down the respiratory tract most of the bigger sized particles are arrested in it by inertia and gravity settlement, allowing only the particles in the size range of 2 microns and below to pass into the alveoli. Since not more than 2/3rd to 3/4th of the inhaled air can reach the pulmonary air space, it follows that deposition here cannot exceed a similar fraction of inhaled dust even when the local efficiency of deposition is 100%. It has been calculated that about 40 percent of the inhaled dust (weight basis) is cut off by the upper respiratory tract and ony about 25 to 30 percent deposited in the alveoli, the rest being either arrested by the nose and throat or breathed out. The finer dust deposited in the alveoli can not be cleared out in the above manner and remains there to be dealt with by phagocytic cells whose actions are usually slow and incomplete. Some of the very fine dust particles reaching the alveoli are dissolved and go into the blood stream.

Pneumoconiosis :

Dust in mines and other dusty places of work in factories cause diseases of the lungs which are grouped under the general term *pneumoconiosis (Greek, Pneumon-lung and conis-dust)*. The term is applied to all conditions of the lungs resulting from the inhalation of dust over long periods, but in recent years, distinct terms are being used to denote the diseases caused by specific dusts, *e.g.* silicosis (by silica or quartz dust); siderosis (by iron oxide dust);

berylliosis (by beryllium either in the form of dust or fumes); asbestosis (by silicate of magnesium), etc. Dusts from lime stone, shale and some metallic ores are not harmful. Although it is generally agreed that anthracite and bituminous dusts do not produce lung disease, they do cause asthmatic conditions when breathed over a long period.

The lung diseases caused by dust may exist for considerable periods without producing symptoms or impairing physical efficiency.

Classificiation :

The classification of the pneumoconiosis may be made according to the physical nature of the dust or by the type of tissue response to the particular agent. Other consider the clinical and roentgen features of the particular dust disease. The inorganic dusts producing the disease are classified as :

A. *Fibrosis producing :* Silicosis and asbestosis are the most important in this group. These dusts are slightly soluble.

B. *Non-Fibrosis producing :* Anthracosis is the chief example in this group. These dust are inert and become encapsulated in the tissues or absorbed but do not produce fibrous tissue.

C. *Toxic and/or irritant :* This group comprises lime, dichromate compounds, lead, mercury and other heavy metals. These are corrosive and cause severe local reactions. If absorbed into the body in sufficient amount, generalised toxic effects appear. The dust and also the vapour fo these substances are harmful.

Silicosis :

This is the most disabling and worst of all the dust diseases. It results in fibrous tissues of the lungs and may ultimately lead to tuberculosis. Workers engaged on stone drifting, tunneling, rock drilling (at the surface in quarries, or underground), and stone crushing are more prone to silicosis; so also are the workers engaged in grinding and polishing industry and in iron and steel industries. It has been found that particles of silica measuring 0.5 to 2.5 microns in dia. are most apt to produce damage. Dust particles of silica are partly transformed in the alveoli into poisonous silicic acid (H_2SiO_3) which passes into the blood.

Asbestosis :

Asbestosis is a kind of pneumoconiosis which results from the inhalation of hydrated magnesium silicate. An important feature of this disease is the presence of asbestos bodies in the lung and sputum. Fibrosis of the lungs develops faster in asbestosis that in silicosis and in extreme cases a person may die of asbestosis within five years of the onset of symptoms.

Siderosis :

Siderosis is caused by inhalation of iron dust. The pathologic chang are due principally to the presence of silica in iron ore. Electric arc welde are also apart to develop this type of disease due to the fact that the electrod used in the process are composed of approximately 99% ferrous materia The fumes arising from the electrodes contain inorganic substances, particular finely divided iron oxide. Iron ore dust, through it produces pneumokonios usually does not show fibrosis and is generally non-progressive and non-disabli though the alveolar walls indicate their pigmenttation and consequent thickenin If, however, iron ore dust contains some free silica fibrosis will result.

Anthracosis :

Anthracosis occurs in coal miners and in city dwellers exposed dusty atmosphere containing large amounts of smoke. The term means bla lungs. The particles of carbon which find their way into the lung produce phagocytic reaction similar to that in silicosis. However, the carbon partic are inert chemically and do not cause local injury to the lung tissue. The forei material may be carried to the liver, spleen, bone, marrow lymph nodes an other organs. The affected organs are dark in colour but show no local reactio unless silica is also present. In mine workers subjected to the inhalation coal and silica dust mixtures, anthrocosilosis developes characterised extensive silicotic fibrosis and marked emphysema. These patients show high degree of tuberculosis. Miners in underground workings having po ventilation are prone to this disease and the symptoms may be observed o after 15 to 20 years of working. According to British Stanard, the concentrati of coal dust of particle size 1 to 5 μ should not exceed 850 particles per c

It is now well known that the capacity of the dust to damage t lungs depends upon the contents of free silica in it. According to Russi standards the permitted content of silica dust in the mine air should not exce 2 mg/m³. The following table gives the permissible concentrations :

Permissible concentrations of non-toxic mine dust

Type of dust	Permissible concentration; mg/m
Dust containing more than 70% free silica	1.0
Dust containing 10 to 70% free silica	2.0
Coal or coal-measure dust containing more than 10% free SiO_2	2.0
Coal dust containing less than 10% free silica	4.0
Coal dust containing no free silica	10.0

British Standards lay down that the concentration of airborne stone dust particle size 0.5 – 5μ must not exceed 450 particles per c.c. The same standard is accepted in India. Mouth breathers are more likely to acquire the disease than nasal breathers with normal condition of the nose and pharynx.

The symptoms of lung diseases in general and of silicosis in particular, are :

Stage 1 : The patient may look apparently healthy but has slight shortness of breath on very little exertion and has dry cough.

Stage 2 : Definite shortness of breath, pain in chest; dry cough in mornings, sluggish movements and a reduced capacity for work.

Stage 3 : Pronounced shortness of breath, frequent dry coughing, frequent tendency to spit, pulse rate increased, and capacity for work greatly impaired.

The above three stages can be detected by radiographs of the lung. The last stage in silicosis is emphasised by large shadows corresponding to areas of dense fibrosis.

Alkalies increase the effect of the dust while coal dust, aluminium and other minerals apparently retard the development of silicosis. The principle of aluminium therapy depends on the fact that particles of metallic aluminium of amorphous hydrated aluminia, when engulfed in the same phagocyte cells along with silica dust, neutralise the effect of silica and thus arrest further progression of fibrotic tissue reactions. Aluminium ordinarily has no harmful effect on lung tissue but heavy doses make the lung or susceptible to tuberculosis infection.

The remedy in preventing lung diseases lies in :

1. Reducing dust formation at places where workers are engaged.
2. Collecting the dust by suitable dust traps.
3. Examining the dust concentration at working places at regular intervals by suitable instruments and taking corrective steps.

The ways to reduce airborne dust in underground mines have been described in earlier pages. The instruments to assess the dust concentration in mines will therefore be described here.

Dust sampling methods and instruments :

In order to take suitable preventive and suppressive measures for allaying dust in mines, it is necessary to have a suitable device to estimate or sample airborne dust likely to be breathed by the underground workers.

The sample of the dust collected should be able to give a knowledge of the dust concentration in the dangerous size range. The process of estimating by suitable instruments/devices the dust concentration in the collected sample is known as sampling of dust.

Dust sampling methods, so far developed, can be classified as follows :

1. Filtration
2. Sedimentation
3. Thermal precipitation
4. Impingement
5. Electrical precipitation
6. Optical method
7. Impaction
8. Centrifuging.

Mass concentration of dust can be obtained by differential weighing of a suitable filter before and after the collection of sample.Dust collectd by filtration can also be spread on glass slids examined under the microscope for determining the particle concentration and size, but this gives unreliable results because of the dificulty of suitably dispersing the dust on a microscope slide. paper filters are usually made in the form of rigid thimbles, such as the Soxhlet thimble through which air is drawn either by a hand pump or by a compressed air injector. the latter being necessary for collecting large samples. paper thimbles have been claimed to have a collection efficiency of 98%.

The method of *thermal precipitation* utilise the principle that when a body surrounded by dusty air is heated, a dust free zone is produced around the hot body, the extent of the dust free zone depending on the temperature gradient between the hot boy and the surrounding air. If such a zone is intercepted by two glass cover slips and the current dusty air allowed to enter between them the dust in the air gets deposited on the cover slips at the points A and B where it remains attached by molecular atttraction (Fig. 9.1) After the sample has been collected on the cover slips, the latter are mounted on a 75mm × 25mm microscope slide and waxed to it so that the dust is enclosed between the two glass surfaces. The slide is then counted under a high power microscope. Counting is done with the help of a suitably calibrated graticule introduced in the eyepicce.

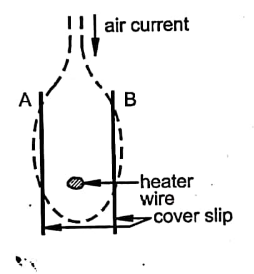


Fig. 9.1 Illustrating principle of thermal recipitator

The thermal precipitator (Fig. 9.2) is an accurate sampling instrument. It collects a represenive sample over a reasonably long duration and the low sampling velocity does not affect the aggregates, but it involves skilled operation and arduous microscopic counting.

The method of impingement utilises the principle of inertia precipitation of dust when a fast moving stream of air impinges on a surface. The mot commonly used instruments based on impingement are the Konimeter and the Midget impinger. The Konimeter (Fig. 9.3) is a very handy, compact and sturdy instrument suitable for routine dust sampling but it is a snap sampling instrument collecting only 5 cm³ of sample over a period of second thus providing a sampling velocity of the order of 100 m/s. Hence for accuracy in computing average dustiness at a particular place in the mine, it is necessary to take a large number of samples.

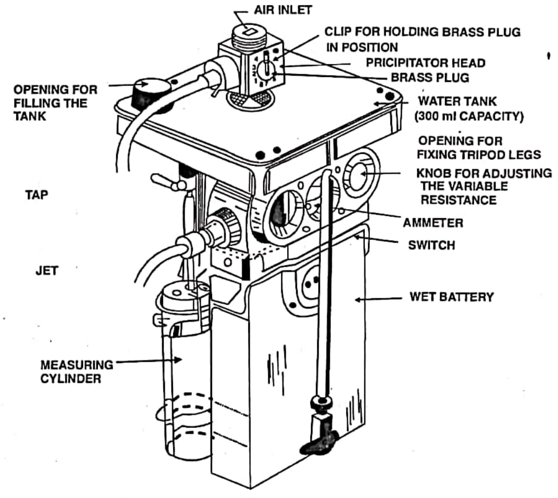


Fig. 9.2 Thermal precipitator

The electrical precipitator essentially consists of a charging wire maintained at a high negative potential of about 12000 volts and surrounded by an earthed concentric cylinder. Dust laden air is drawn through the cylinder by a fan at a constant rate. The dust particles when passing through the instrument get charged and are drawn to and precipitated on the inner surface of the earthed cylinder. This instrument, like the filtration device, has a large sampling capacity and is suitable for collecting large quantities of dust for chemical analysis the mass concentration can also be determined by noting the difference in weight of the cylinder before and after collection of dust. The instrument has a high collecting efficiency, but the high voltage used in it makes it unsuitable for use in coal mines where filters are preferred.

The *optical method* utilises the property of scattering of light by suspension of fine particles. The intensity of scattered light is proportional to the surface area of the particles, a fact well borne out in practice. The instrument based on this principle is the Tyndalloscope, very commonly used in German coal mines for routine dust sampling purposes, mainly because of its cheapness as well as quickness and case of operation.

The P. R. U. hand pump which is commonly used in British coal mines for routine dust sampling purposes partly utilises the optical property for the estimation of the dust concentratiion. In this a certain volume of dusty air is pumped through a suitable filter paper by means of a hand pump. The density of the stain on the filter paper is measured photoelectrically in densitometer where a beam of light is allowed to fall on a photocell connected to a galvanometer, after passing through the stained filter paper.

It was considered till not long ago that lung diseases depend, apart from period of exposure, on the concentration of dust particles in a given volume of inhaled air. This belief has led to the development of such instruments as soxhlet thimble midget impinger, sedimentation cell, thermal precipitator, P. R. U. hand pump, konimeter, etc. all of which are aimed at measuring the number of dust particles per c.c. of air. Most of the instruments work on a simple basic principle : A small quantity of the air is forced at high velocity upon a greased slide on which the size and number of adhering particles are estimated by a microscope. Of these instruments the konimeter is the only one in general use for taking snap samples and is very useful for examining the relative dust concentration at a point during different stages of a working operation. The cost and efforts involved in optical dust cunting precludes the use of the konimeter and the precipitator for the routine assessment of working places for dust concentration; as a large number of samples have to be optically

studied, the work is tedious. For these reasons, photo-electric examination of filter stains obtained by the P. R. U. hand pump has proved to be satisfactory.

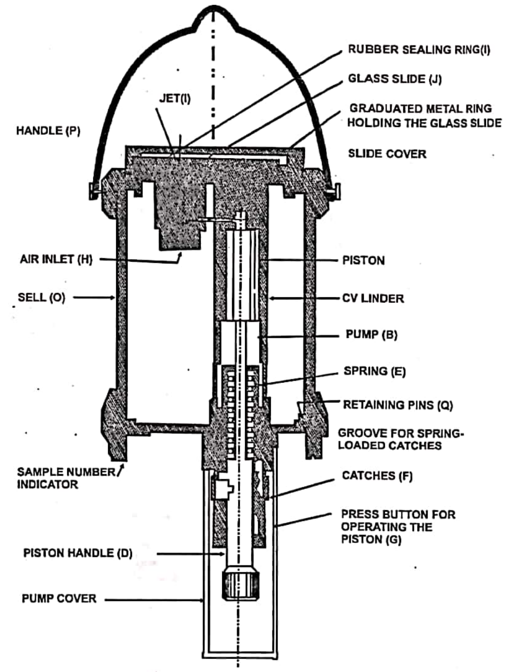


Fig. 9.3 Sartorius mining-type konimeter.

Ailments of the ear :

One of the ailments from which some categories of mine workers may suffer is the ailment of the ear. Such ailment often goes unnoticed. Workers exposed to loud noises of machines in mines and factories over long periods develop aliments of the ear and such machines in mines are coal cutting machines, shearers, jack hammers, crushers, and other machines producing unbearably large noise.

Sound, when it reaches an uncomfortable level, is called noise. Two units are used in the measurement of sound. The first is the unit of Hertz (Hz) which is a measure of the frequency of sound. Hz is an SI unit which is in common use in noise technology. When this number is small in value the sound will have a low note, and when large in value, the sound wil be high pitched. A human being will hear sounds between values of 20 and 15000Hz. Ultra-sonic flaw testing equipment, for example, operates at a frequency of the order of 3 MHz. A sound produced at this frequency is not audible to the human ear.

The second unit of sound is bel, a technical term named after the inventor of telelphone, Alexander Graham Bell. However, the more commonly used unit is decibel, (dB) which is 1/10th of a bel. Frequency measures the pitch of noise and decibel measures the intensity of noise. A whisper at 1m distance is assigned a noise rating of 30 dB while the noise level of an ordinary conversation at 1m distance is 60dB. This, of course, does not mean that ordinary conversation is only twice more noisy than a whisper, because the decibel scale is logarithmic where the difference mounts in power of ten. Thus a busy street with heavy city traffic (80 dB) is 100 times more noisy than an average conversation. To measure industrial noise, meters are available which give a direct reading in decibels.

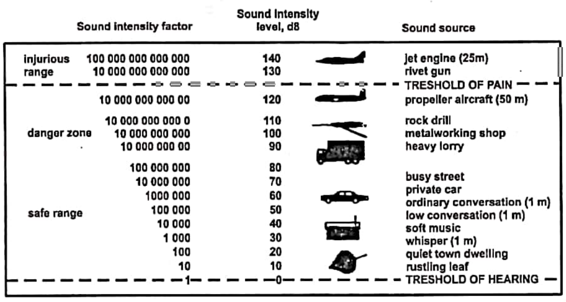


Fig.

What is the level of tolerance of noise ? Such level is dependent on the period of exposure, distance of the ear from the source and the way the enclosure reverberates the sound. However, in general, 70 to 85 dB may be taken as the level of tolerance. Prolonged exposure to noises greater than this may damage the ear temporarily or even permanently. Under persistant noise the vessels carrying blood to the brain dilate and thereby transport more blood there, causing headache. Besides there may be an aggregation of red blood corpuscles within the thin blood vessels and the vessels may contract in spasms. This may cause the blood to thicken leading to heart attack. The lesser evils are hearing loss, psychological-emotional distress, increased heart beats, indigestion, tension, increase in blood pressure and sleeplessness.

A newsreport released by the Society for Clean Environment in June 1980 on the occassion of World Environment Day states that Mumbai, Delhi and Kolkata are among the noisiest cities in the world where the average noise levels are over 90 dB.

QUESTIONS

1. Write short notes on :

   (a) pneumokoniosis, (b) nystagmus, (c) silicosis, (d) konimeter

2. (i) What are the dust standards adopted for reducing the incidence of silicosis ?

   (ii) How does the respiratory system control the dust inhaled by the human being ?

   Name the various dust sampling instruments and state the principle of their operation. How does the gravimetric dust sampler work and in what way is it superior to the conventional dust sampling instruments ?

3. How is the intensity of noise measured ? In what way is a miner affected by exposure to loud noises over long years ?

O O O

CHAPTER **10**

NUMERICAL EXAMPLES IN VENTILATION

Example 1 : The quantity of air going down a D.C. shaft is 900 m^3/min. The surface main ventilator developes water gauge of 50mm. When the ventilator is stopped the air going down the shaft is 300 m^3/min. What is the N.V.P. assisting the fan ?

Answer :

Supposed N mm is the w.g. by N.V.P.

From the equation, $P = RQ^2$, we get

$$R = \frac{N+50}{\left(\frac{900}{60}\right)^2} = \frac{N+50}{225}$$

Resistance of the mine is the same whether the fan is stopped or running.

$$\therefore \quad R = \frac{N}{\left(\frac{300}{60}\right)^2} = \frac{N}{25}$$

$$\frac{N+50}{225} = \frac{N}{25}$$

$\therefore$ $N = 6.25$ N.V.P. is 6.25 mm w.g.

Example 2 : The quantity of air going down a D.C. shaft is 7200m^3/ min at 80mm w.g. with the fan running at to normal speed. If the speed is reduced, the quantity falls to 4800m^3/min and the w.g.n. developed is 32mm. Calculate the N. V. P.

Answer :

Suppose N mm is the w.g. by N. V. P.

Total ventilating pressure at full speed of fan

$$= (80 + N) \text{ mm w. g.} \qquad \text{--- (i)}$$

Total ventilating pressure at reduced speed of fan

$$= (32 + N) \text{ mm} \qquad \text{--- (ii)}$$

From (i) $R = \frac{P}{Q^2} = \frac{80+N}{\left(\frac{7200}{60}\right)^2} = \frac{80+N}{(120)^2}$

From (ii) $R = \frac{P_1}{Q_1^2} = \frac{32+N}{\left(\frac{4800}{60}\right)^2} = \frac{32+N}{(80)^2}$

Mine resistance is the same in both cases

$$\therefore \quad \frac{80+N}{(120)^2} = \frac{32+N}{(80)^2}$$

i.e. $320 + 4N = 288 + 9N$

$32 = 5N$ or $N = 6.4$

$\therefore$ $N = 6.4$ mm w.g.

N. V. P. is 6.4 mm w. g.

Example 3 : A D. C. Shaft is 465 m deep and the average temperature of the downgoing air is 30°C. The U.C. shaft has equal depth but the average air temperature in that shaft is 37°C. What assistance, expressed in H.P. does this difference in air temperature render when the air passing the D.C. shafts 100m^3 sec. (Assume average barometric pressure in D.C. shaft to be 750 mm of Hg).

Answer :

$$\text{Motive column } h = D \times \frac{Tu - Td}{273 + Tu}$$

$$= 465 \times \frac{37 - 30}{273 + 37} = 10.5 \text{ m}$$

Density of air in D.C. shaft $W_d = \dfrac{0.4645\ B}{273 + T_d}$

$$= \frac{0.4645 \times 750}{273 + 30}$$

$$= 1.1498\ kg/m^3$$

N. V. P. = Motive column × density of air in D.C. shaft

$= 10.5 \times 1.1498 = 12.07\ kg/m^2$ or 12.07 mm of w.g.

H.P. assistance due to N. V. P. $= \dfrac{PQ}{75} = \dfrac{12.07 \times 100}{75}$

$$= 16.09$$

Example 4 : Calculate the w.g. a fan has to develope for ventilating a drift under the following conditions.

Cross-section of the drift – 2.4 m high × 3 m wide

Length – 200 m;

Quantity of air passing – 360 m^3/min.

Coeff. of resistance = 0.00161mm of w.g. per m^2 of rubbing surface per 1 m/sec. velocity of air.

Answer :

Surface area of the drift, S = Perimeter X length

$= 2\ (2.4 + 3) \times 200$

$= 2160\ m^2$

Area of drift, $A = 2.4 \times 3 = 7.2\ m^2$

Pressure to be developed, $p = \dfrac{KSQ^2}{A^3}$

$$= \frac{0.00161 \times 2160 \times (6)^2}{(7.2)^3}$$

$$\simeq 0.335\text{mm w.g.}$$

Example 5 : Three underground roadways in parallel spread out from a point near the bottom of a D.C. shaft and join at a point near the bottom of the U.C. shaft. The parallel roadways have resistance of 9, 16 and 25 kilomurg respectively. The resistance of the fan drift is 1 Kilomurg, and the D.C. shaft and U.C. shaft 2 Kilomurg and 3 Kilomurg respectively including the connecting trunk roads. Calculate the combined resistance of the whole mine.

If the w.g. developed by the fan is 153mm, calculate the quantity of air that passes through the whole mine and through each split.

Answer :

If R is the combined equivalent resistance of the 3 roaways in parallel and R_1, R_2, R_3 their individual resistance, then

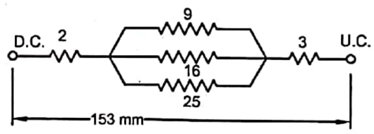


Fig.

$$\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}}$$

$$\therefore \quad \frac{1}{\sqrt{R}} = \frac{1}{\sqrt{9}} + \frac{1}{\sqrt{16}} + \frac{1}{\sqrt{25}} = \frac{47}{60}$$

$$\therefore \quad R = \left(\frac{60}{47}\right)^2 = 1.63\text{k}\mu$$

Total resistance of the mine = 1+ 2 +1.63 + 3

$= 7.63$kn

∴ Quantity of air passing through the mine is

$$Q = \sqrt{\frac{P}{R}} = \sqrt{\frac{153}{7.63}}$$

$= 4.478\ m^3$/sec.

$= 268.68\ m^3$/min

$$Q_1 = Q \times \sqrt{\frac{R}{R_1}} = 268.68 \times \sqrt{\frac{1.63}{9}}$$

$$= 114.46 \text{ m}^3/\text{min}$$

$$Q_2 = 268.68 \times \sqrt{\frac{1.63}{16}} = 85.76 \text{ m}^3/\text{min}$$

$$Q_3 = 268.68 \times \sqrt{\frac{1.63}{25}} = 68.61 \text{ m}^3/\text{min}$$

Example 6 : A district of a mine is ventilated by 30 m³/sec quantity of air and the water gauge across the district is 25mm. If the quantity has to be reduced to 20 m³/sec. by installing a regulator in the return of the district, calculate the size of the regulator.

Answer :

Resistance of the district

$$R = \frac{P}{Q^2} = \frac{25}{(30)^2} = \frac{1}{36}\ \text{k}\mu$$

Combined resistance of the district and the regulator should be

$$R_1 = \frac{P}{Q_1^2} = \frac{25}{(20)^2} = \frac{1}{16}\ \text{k}\mu$$

So, resistance of the regulator, $R_2 = \frac{1}{16} - \frac{1}{36} = \frac{20}{576}\ \text{k}\mu$

Area of the regulator to be installed, is given by the formula

$$A = \frac{0.385}{\sqrt{R_2}}$$

$$= \frac{0.385}{\sqrt{\frac{20}{576}}}$$

$$= 2.07 \text{ m}^2$$

Example 7 : The dry and wet bulb temperature recorded in a mine are as follows :

D.C. pit bottom		U.C. pit bottom	
Dry	Wet	Dry	Wet
33°C	30°C	35.2°C	34°C

The quantity of air circulating is 6,000 m³/min. The water content of saturated air at normal atmospheric pressure is given below :

at 33°C – 35 g/m³; at 35.2°C – 38.5 g/m³.

Calculate the amount of water carried out by the air from the mine per day.

Answer :

Difference between dry and wet bulb temperature at D.C. pit bottom = 33 – 30 = 3°C.

∴ Relative humidity of intake air = 100 – 3 × 7 = 79%

∴ Water content of intake air = 35 × 0.79 = 27.65 g/m³

Difference between dry and wet bulb temperatures at U.C. pit bottom = 35.2 – 34 = 1.2°C.

∴ Relative humidity of return air = 100 – 1.2 × 7 = 91.6%

∴ Water content of return air = 38.5 × 0.916

= 35.27 g/m³

∴ Amount of water carried out by ventilating air

= 35.27 – 27.65

= 7.62 g/m³.

The amount of water carried by ventilating air per day

$$= \frac{7.62 \times 6000 \times 60 \times 24}{1000 \times 1000}$$

$$= 65.836 \text{ tonnes.}$$

Example 8 : An airway 2.5 × 2m absorbs a ventilating pressure of 25mm of w.g. It is decided to enlarge the cross-section to 3m × 2.5m. What will be the saving of w.g. if the same quantity of air passes after enlargement, assuming the coefficient of friction to remain unchanged ? What will be the saving in ventilation cost per year if the quantity flowing is [illegible]000 m³/min and the energy cost is 15p per unit.

Answer :

Let suffix 1 and 2 refer to old and new airway respectively.

From the equation $P = \dfrac{KSQ^2}{A^3}$

we get $\dfrac{P_2}{P_1} = \dfrac{S_2}{A_2^3} \times \dfrac{A_1^3}{S_1}$ i.e. $\dfrac{S_2}{S_1} \times \left(\dfrac{A_1}{A_2}\right)^3$

The length of airway in both cases is the same so that the rubbing surface is proportional to the perimeter.

$\therefore$ $\dfrac{P_2}{25} = \dfrac{2(3+2.5)}{2(2.5+2)} \times \left(\dfrac{5}{7.5}\right)^3$ or $P_2 = 9$mm of w.g.

$\therefore$ Saving in w.g. on the airway = 25 – 9 = 16 mm of w.g.

Saving in air H.P. = $\dfrac{PQ}{75}$ H.P.

or $= \dfrac{PQ}{102}$ kW

$= \dfrac{16 \times (3000/60)}{102} = 7.84$ kW

Assuming overall efficiency to be 80%

Actual saving = $\dfrac{7.84}{0.80} = 9.8$ kW

$\therefore$ Annual saving in ventilation cost

= Rs. 9.8 × 24 × 365 × 0.15

= Rs. 12877.

Example 9 : A scale model has been made of a 6m diam. circular shaft which is to be sunk in a mine. In a wind tunnel, the scale model has been tested with the following results :

Scale of model – 1 to 10

Length of test section of model – 10m

Mean air velocity in test section – 250m/min

Mean air density in test section – 10 kg/m³

Static preasure drop across the test section – 250 mm w.g.

The actual shaft will be 100m deep and is expected to pass a quantity of 1000m³/min at an air density of 1.10kg/m³. Estimate the static pressure loss in the shaft when the required quantity of air is passing.

Answer :

Let suffixes 1 and 2 refer to the model and the shaft respectively. Then

$$P_1 = \frac{K_1S_1V_1^2}{A_1} \text{ and } P_2 = \frac{K_2S_2V_2^2}{A_2}$$

$$\therefore \frac{P_2}{P_1} = \frac{K_2S_2V_2^2}{K_1S_1V_1^2} \times \frac{A_1}{A_2} = \frac{S_2V_2^2}{S_1V_1^2} \times \frac{A_1}{A_2} \text{ assuming } K_1 = K_2$$

Now $P_1 = 250$ mm w.g. ; $S_1 = \pi \times 0.6 \times 10 = 18.86$ m²

$S_2 = \pi \times 6 \times 1000 = 18860$ m²

$V_1 = 250$ m/min

$V_2 = \dfrac{Q}{A} = \dfrac{1000}{\pi \times 3^2} = 35.35$m/min

$A_1 = \pi \times (0.3)^2 = 0.2828\text{m}^2$

$A_2 = \pi \times (3)^2 = 28.28\text{m}^2$.

$$\therefore \frac{P_2}{250} = \frac{1.10}{1.00} \times \frac{18860 \times (35.35)^2}{18.86 \times (250)^2} \times \frac{0.2828}{28.28}$$

This gives $P_2 = 55$ mm w.g.

Example 10 : A mine is ventilated by two splits, A and B, having resistances 0.15 kilomurg and 0.60 kilomurg respectively. The resistance of trunk airways and shafts is 0.13 kilomurg and a total quantity of 2000 m³/min of air flows through the mine.

If the ventilating pressure be 200 Kg/m², what will be the pressure requirement of a booster fan to be installed in split B to increase the air flow in that split to 1500m³/min.

Answer :

Pressure drop in shafts and trunk airways,

$P = RQ^2$

$= 0.13 \times (2000/60)^2$

$= 144.4$ mm w.g.

So, pressure across the splits = 200 – 144.4 = 55.6 mm w.g.

After installation of the booster

Let quantity in split A to be Q_1 m³/sec.

Pressure across the splits = $0.15\, Q_1^2$

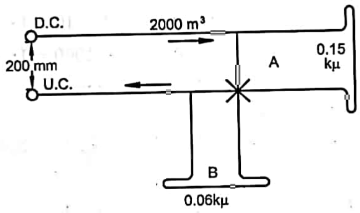


Fig.

Pressure spent in shafts and trunk airways

$$= 0.13\left(Q_1 + \frac{1500}{60}\right)^2 \qquad \text{---- (2)}$$

Since total ventilating pressure remains constant,

$$0.15\,Q_1^2 + 0.13\left(Q_1 + \frac{1500}{60}\right)^2 = 200$$

or $\quad 0.15\,Q_1^2 + 0.13\,(Q_1 + 25)^2 = 200$

So, $\quad Q_1 = 11.96$ m³/sec.

So, From (1)

Pressure across the splits = $0.15 \times (11.96)^2 = 21.47$ mm w.g.

i.e. 21.5 mm w.g.

New pressure requirement of split B,

$P = RQ^2$

$$= 0.60 \times \left(\frac{1500}{60}\right)^2 = 375 \text{ mm w.g.}$$

So, Pressure to be developed by the booster fan

$= 375 - 21.5$

$= 353.5$ mm w.g.

Example 11 : A mine fan having a capacity to circulate 25m³/s of air in a mine at a pressure of 50 mm w.g. is to be assisted by a booster. The mine consists of two splits, A and B, A circulating 15 m³/s and B, 10m³/s. The object of installing a booster in split B is to increase the quantity in it to 15m³/s. Calculate the size of the booster if the resistance of the shafts and trunk airways is 0.02kμ.

Answer :

Let $\quad R_A$ = resistance of split A

R_B = resistance of split B

R_T = resistance of trunk airways and shafts

P_B = pressure developed by a booster fan

Q_A = quantity in split A after booster installation

Q_B = quantity in split B after booster installation

Q_T = total quantity flowing through the mine after booster installation.

Let us assume that the main fan pressure after installation of booster remains unchanged. Then we have

$50 = R_T \times 25^2 + R_A \times 15^2 = R_A \times 25^2 + R_B \times 10^2$

This gives $R_A = 0.17$ kμ

and $R_B = 0.38$ kμ since $R_T = 0.02$ kμ

After installation of the booster

$50 = R_T\,Q_T^2 + R_A\,(Q_T - Q_B)^2$

$= 0.02\,Q_T^2 + 0.17\,(Q_T - 15)^2$ since $Q_B = 15$ m³/s

Solving the above equation, $Q_T = 29$ m³/s

Now considering flow in split B,

$$50 + P_B = R_B Q_B^2 + R_T Q_T^2$$
$$= 0.38 \times 225 + 0.02 \times 29^2$$

This gives P_B = 52.2 mm of w.g.

A study of this example will show that the total quantity the mine increases with an increase in the quantity of the boosted split. The result is greater pressure loss in the shafts and trunk. Airways thus reducing the pressure across the splits. Moreover, the main fan, when circulating a larger quantity, generates less pressure. These cause a reduction in the quantity circulating in the unboosted split. It can be shown by an example that two large a booster in one split can cause stoppage of air current, or even reversal of air current, inthe other split, particularly if the trunk resistance is high compared to the resistance of the splits. Selecting the size of a booster therefore needs careful consideration.

Example 12 : A main mine fan developes a w.g. of 120mm of which 80mm is consumed in the shafts and the trunk airways and 40mm is available to ventilate two splits A and B. The quantity of air passing split A is 15m³/s and that in split B is 10m³/s. To increase the quantity in split B, a booster, is to be installed in it. Calculate the size of the booster which will cause the stoppage of airflow in split A.

Answer :

Quantity passing through trunk airways $= 10 + 15$

$= 25$ m³/s

Resistance of trunk airways (and shafts) $= \dfrac{80}{25^2}$

$= 0.128$ kμ

No air flows through A when the booster is installed in B, which means the whole of the main fan pressure is consumed in overcoming the resistance of the shafts and trunk airways.

Let Q the quantity flowing through the trunk airways as well as through B after the installation of the booster.

Assuming the main fan pressure to remain the same before and after the booster installation of the booster.

Assuming the main fan pressure to remain the same before and after the booster installation.

$120 = 0.128\,Q^2$ or, $Q = 30.62$ m³/s.

Resistance of $B = \dfrac{40}{10^2} = 0.4$ kμ

Therefore pressure of the booster $= 0.4 \times (30.62)^2$

$= 375$ mm w.g.

Example 13 : A mine circulating a total air quantity of 1700 m³/min has two districts, A and B, connected in parallel. A quantity of 1133 m³/min flows in A and the balance in B. It is proposed to install a booster in split B so that its quantity may be increased to meet the demands of the concentration or workings. If the water-gauge across the outbye ends of the turnk airways is 10cm and that across the splits 2.5 cms, find the critical pressure of the booster fan and also the quantity of air that would then be circulating in B and the turnk airways. Assume pressure across trunks to remain unaltered.

Answer :

Definition of critical pressure : In theory, the booster has a critical pressure at which "the pressure difference at the point of splitting drops to zero. All the air passing underground then passes through the booster while the air in other splits stagnates".

In other words, the highest pressure that the booster fan can produce, without air reversal taking place, is called its critical pressure.

It can be shown that $P_c = \dfrac{P_f \times R_s}{R_t}$

where P_c = critical pressure of the booster

P_f = pressure of surface fan

R_t = resistance of trunk airways

R_s = resistance of split in which booster operates.

From the above definition we get

$$\left.\begin{array}{lll} \text{(i)} & \text{Pr. drop across splits} & = \text{Zero} \\ \text{(ii)} & \text{Quantity in A, } Q_A & = \text{Zero} \\ \text{(iii)} & \text{Quantity in B, } Q_B & = Q_T \end{array}\right\} \text{as shown in Fig. (ii)}$$

So, question now reduces to finding (i) P_c of booster

and (ii) Q_T (or Q_B)

From Fig. (i) Using $P = RQ^2$

where P = Pr. in mm of w.g. = kg/m²

R = Kilomurgs

Q = Quantity in m³/sec.

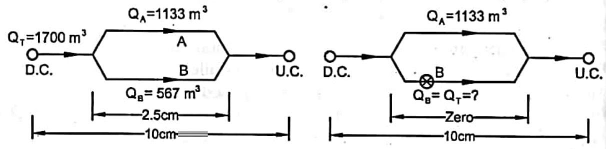


Fig.

(i) Before booster installation

(ii) After booster installation & critical pressure

(i) For trunks

P_T = (10 × 10 – 2.5 × 10) kg/m² = 100 – 25 = 75mm or 75 kg/m²

Q_T = 1700/60 m³/sec.

$$\text{So, } R_T = 75 \div \left(\frac{1700}{60}\right)^2 = \frac{75\times6\times6}{170\times170} = \frac{2700}{170\times170} = \frac{27}{289}\text{k}\mu \text{ --- (1)}$$

(ii) For district B.

$$P_B = 2.5\times10 = 25 \text{ kg/m}^2 \text{ and } Q_B = \left(\frac{567}{60}\right) \text{m}^3\text{/sec.}$$

$$\text{So, } \quad R_B = 25 \div \left(\frac{567}{60}\right)^2 = \frac{25\times60\times60}{567\times567} = \frac{90{,}000}{567\times567}\text{k}\mu$$

From the Fig. (i) P_T = 10 × 10 = 100mm or 100 kg/m²

$$R_T = \text{Just found in (1)} = \frac{27}{289}\text{k}\mu$$

So, using $P_T = R_T Q_T^2$

$$Q_T = \sqrt{\frac{100}{27/289}} = \sqrt{\frac{100\times289}{27}}\text{m}^3\text{/sec.}$$

= 32.71 m³/sec.

or Q_T = 32.71 × 60

= 1962.60 m³/min = Q_B

(Because the Q_T has increased)

(ii) Q_B = 1962.60 m³/min = 32.71 m³/sec.

$$R_B = \frac{90{,}000}{567\times567} \text{ Kilomurg from (2)}$$

∴ $P_B = P_C$ (for booster) = $Q_B^2 \times R_B$

$$= \frac{90{,}000}{567\times567}\times(32.71)^2$$

= 299.3 Kg/m² or mm of w.g.

= 29.93 cm of w.g.

***Answer*:**

(1) P_C (of booster) = 29.93 cm of w.g.

(2) $Q_T = Q_B$ - 1962.6 m³/min.

***Example 14*:** The evasee chimney of a fan has an area of 4m² at the base and 14m² at the outlet. Calculate the saving of water gauge and H.P. by virtue of the evasee when the output of the fan is 6000m³/min (Air density 1.2 kg/3).

***Answer*:**

$$\text{Air velocity at the base, } V_1 = \frac{6000}{4\times60} = 25\text{m/sec.}$$

$$\text{Air velocity at the outlet, } V_2 = \frac{6000}{14\times60} = 7.14 \text{ m/sec.}$$

Gain of pressure $= \frac{V_1^2}{2g} - \frac{V_2^2}{2g}$

$= \frac{(25)^2 - (7.14)^2}{2 \times 9.8}$

= 29.287 m head of air

= 29.287 × 1.2 kg/m²

= 35.14 kg/m² or mm of w.g.

∴ Saving in w.g. is 35.14 mm.

Air H.P. is given by the formula

$$\text{H.P.} = \frac{PQ}{75}$$

where P = Pressure in kg/m²

and Q = Quantity in m³/sec.

Gain of air $\text{H.P.} = \frac{PQ}{75} \times 0.60$, assuming the evasee 60% efficient.

$$= \frac{35.14}{75} \times \frac{6000}{60} \times \frac{60}{100}$$

The saving in H.P. by virtue of the evasee is 28.11.

Example 15 : Find the percentages of blackdamp, whitedamp, firedamp and air in a mine air sample having the following analysis (percent by volume) :

Q_2 = 19.11 ; N_2 = 79.04 ; CO_2 = 0.25 ; CO = 0.02;
CH_4 = 1.58;

What is the composition of blackdamp ?

Answer :

Atmospheric air entering the mine has the following composition :

O_2 = 20.93 ; N_2 = 79.04 and CO_2 = 0.03 (Volume %)

∴ N_2 equivalent to 19.11 litres of O_2 in the sample

$$= 79.04 \times \frac{19.11}{20.93} = 72.16 \text{ litres}$$

CO_2 equivalent to 19.11 litres of O_2 in the sample

$$= 0.03 \times \frac{19.11}{20.93} = 0.03 \text{ litres}$$

∴ Excess nitrogen in the sample = 79.04 – 72.16

= 6.88 %

Excess CO_2 in the sample = 0.25 – 0.03 = 0.22%

∴ Blackdamp in the sample = 6.88 + 0.22 = 7.1%

Atmosphere air = 19.11 + 72.16 + 0.03 = 81.3%

The composition of the sample, therefore, is as follows (%) :

Blackdamp = 7.10 ; Whitedamp = 0.02; Firedamp = 1.58; Air = 91.3.
Total = 100.