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Mine Ventilation & Environmental Hazards

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MINE VENTILATION & ENVIRONMENTAL HAZARDS

COMPOSITION OF AIR

Air is the homogeneous mixture of gases that covers the Earth's surface. It consists mainly of N_2, O_2 and CO_2. Air composition is given below.

Gas	Vol%	Wt%
Nitrogen	78.09	75.55
Oxygen	20.95	23.13
CO_2	0.03	0.05
Other gases	0.93	1.27

For calculations involving quality control, it is customary to assume dry air and compute problems on a volume basis, taking the composition approximately as

Oxygen 21%

Nitrogen and "inert" gases 79%

MINE AIR/GASES

While mine air starts as regular atmospheric air, it can become contaminated with various gases and particulates during mining activities. These contaminants can include:

- **Gases**: Methane, carbon monoxide, carbon dioxide, hydrogen sulfide, nitrogen dioxide, etc.
- **Particulates**: Dust from rock drilling, blasting, and other operations.

A diverse group of gases can be found in underground environments. The main sources of these gases is shown below.

Gas	Sources
CO	Incomplete combustion Emanations Explosions Blasting

CO_2	Complete combustion Fermentation Breathing Chemical reactions between acidic water and carbonates Blasting
CH_4	Decomposition of organic matter in the presence of water Carbon blasting (methane in pores)
N_2	Blasting
NO_x	Diesel engines Explosions Blasting
H_2S	Rarely present Bacterial decomposition of organic matter Stagnant waters Diesel engines (sulphur in fuels)

OXYGEN

The human respiratory system requires oxygen in varying amounts to maintain life. The quantity of oxygen required is a function of physical activity. That is. the more active the individual, the higher the respiratory rate and the larger the volume of oxygen consumed. Oxygen inhaled (21% by volume of the air inhaled) is much greater than the oxygen consumed. When one breathes normal air, the air that is exhaled contains approximately 16% O_2, 79% N_2, and 5% CO_2. All active workings in coal mines be ventilated by a current of air containing not less than 19.5% oxygen and not more than 0.5% carbon dioxide.

Respiratory quotient, gives the ratio of carbon dioxide expelled to the oxygen consumed. In other words, a miner working at a moderate rate (respiratory quotient - 0.9) would consume 3.3 x 10^{-5} m^3/s) of oxygen and expel 2.97 x 10^{-5} m^3/s) carbon dioxide.

Physiological effects of an oxygen-deficient environment

The physiological effects of an oxygen-deficient environment vary from individual to individual and with the length of exposure. However, the following effects have been observed:

% Oxygen in air	Effect
17	Faster, deep breathing
15	Dizziness, buzzing in ears, rapid heartbeat
13	Possible loss of consciousness with prolonged exposure
9	Fainting, unconsciousness
7	Life endangerment
6	Convulsive movements, death

Detection of Oxygen

The percentage of oxygen present in an atmosphere can be estimated by using the principle of electrochemical method, paramagnetic method or the flame safety lamp.

Electrochemical method

In the **electrochemical method**, very small concentrations of the gas are detected by its influence on the output from an electrochemical cell. The MSA Oxygen Indicator Model 244, Auer Oxygen Indicator Model P etc. are based on this principle and use fuel cells as oxygen sensors. The cells contain a gold cathode and a lead anode in a basic electrolyte.

The whole setup is encapsulated in an inert plastic which is sealed off. A thin membrane is used to protect the sensing electrode and it allows oxygen to get diffused into the cell. The oxygen gets dissolved in the electrolyte and reacts with the gold cathode. Thus the gold cathode becomes positively charged. The lead anode becomes negatively charged through the formation of lead oxide and water. A current proportional to the oxygen concentration flows between the cathode and the anode which is used as a measuring signal.

Flame safety lamp method

The **flame safety lamp** is also used to find out the percentage of oxygen in the atmosphere. In this method, the length of the flame produced by the lamp because of the burning of the gas is used. Accordingly, the oxygen percentage can be found out by noticing the flame height.

NITROGEN

This gas is quite abundant in atmospheric air comprising about 4/5th of the atmosphere. It is colourless, odourless, 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 bums nor supports combustion but it is important for the growth of plants and animal tissues.

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.

Oxides of nitrogen are formed underground during blasting and from the operation of internal-combustion engines. Nitric oxide is rapidly oxidized to nitrogen dioxide in the presence of moisture and air and is therefore seldom found in significant amounts underground.

Nitrogen dioxide is very toxic. The accepted TLV-TWA for both an 8-h exposure and short-term exposure is 5 ppm. The toxic oxides of nitrogen react with moisture to form nitrous and nitric acid. In this manner, relatively small quantities of these gases may cause death by combining with the moisture in the lungs and corroding the respiratory passages. Death from exposure to the oxides of nitrogen may be very quick if the exposure level is high or may occur several days later as a result of pulmonary edema (water in the lungs) or even weeks later as a result of infectious pneumonia. Some of the physiological effects at various concentrations are listed below.

Concentration (ppm)	Effect
60	Least amount causing immediate throat irritation
100	Least amount causing coughing
100-500	Dangerous even for short exposure
200-700	Rapidly fatal

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 and forms 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, burning 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 outburst 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.

Effects of breathing CO_2

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.

Death occurs rapidly at 18% CO_2. Quite often, a mixture of carbon dioxide and air is referred to in mining as "**blackdamp**." An example will demonstrate how minimum breathing requirements may be calculated on the basis of (1) oxygen depletion and (2) carbon dioxide contamination. Flowrate equations establish the quantity balance in each case.

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%.

Physiological effects of blackdamp

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.

Detection of blackdamp

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)

Carbon monoxide is a colourless, odourless, tasteless, toxic, and flammable gas produced by the incomplete combustion of carbonaceous material. Carbon monoxide is both poisonous at very low concentrations and explosive over a wide range (12.5-74% in air). It is formed underground by mine fires and explosions, blasting, frictional heating prior to open burning, low-temperature oxidation, and internal-combustion engines.

Physiological effects of CO

Although explosive, the property that makes carbon monoxide one of the gases most feared by the underground miner is its extreme toxicity. Carbon monoxide acts as a type of asphyxiant by displacing the oxygen normally carried by the haemoglobin of the blood. The affinity of the blood for carbon monoxide is approximately 3000 times that for oxygen. Therefore, if the air breathed into the lungs contains only a small amount of carbon monoxide, the haemoglobin will absorb it in preference to the oxygen present. The compound formed when oxygen joins with haemoglobin is called oxy-haemoglobin, and the compound formed by carbon monoxide and haemoglobin is known as carboxy haemoglobin (COHb).

When discussing the poisoning or toxic effects of carbon monoxide, the percentage COHb or blood saturation level is usually used. At levels as low as 5% COHb, the first effects of carbon monoxide poisoning appear. The COHb blood level is dependent on the carbon monoxide

concentration, the length of exposure, and the level of activity of the individual exposed. At blood saturation levels of 70% COHb and above, the amount of oxygen being carried by the blood is insufficient to sustain life.

Blood Saturation (%COHb)	**Symptoms**
5-10	First noticeable effect, loss of some cognitive function
10-20	Tightness across forehead, possible headache
20-30	Headache, throbbing in temples
30-40	Severe headache, weakness, dizziness, dimness of vision, nausea and vomiting, and collapse
40-60	Increased likelihood of collapse and unconsciousness, coma with intermittent convulsions
60-70	Coma, possible death
70-80	Respiratory failure, death

Example

Find the percentages of blackdamp, whitedamp, firedamp and air in mine air sample having the following analysis (percent by volume):

O_2 = 19.11

N_2 = 79.04

CO_2 = 0.25

CO = 0.02

CH_4 = 1.58

What is the composition of blackdamp?

Solution

Standard composition is not given. These values can be taken as CO_2 – 0.03, O_2 – 20.93 and N_2 – 79.04 respectively (if not given).

CO_2 eq = CO_2 wrt O_2

$$= \left(\frac{CO_2^{std}}{O_2^{std}}\right) O_2^{ret} = \left(\frac{0.03}{20.93}\right) 19.11$$

$$= 0.027$$

N_2 eq = N_2 wrt O_2

$$= \left(\frac{N_2^{std}}{O_2^{std}}\right) O_2^{ret} = \left(\frac{79.04}{20.93}\right) 19.11$$

= 72.17

Excess N_2 = 79.04 – 72.17 = 6.87

Excess CO_2 = 0.25 – 0.027 = 0.223

Blackdamp = 6.87 + 0.223 = 7.09%

Example (GATE 2024)

The data obtained from an air sample analysis of an old working in a coal mine are given.

O_2 17.15%

CO_2 3.40%

CH_4 2.20%

N_2 77.25%.

Considering atmospheric air contains

O_2 20.95%

CO_2 0.03%

N_2 79.02%,

The percentage of blackdamp in the old working, is __. (round off up to 2 decimals). (GATE 2024)

Solution

CO_2 eq. = $(CO_{2,std}/O_{2,std})$ x $O_{2,std}$

= (0.03/20.95) x 17.15 = 0.025

N_2 eq. = $(N_{2,std}/O_{2,std})$ x $O_{2,std}$

= (79.02/20.95) x 17.15 = 64.69

Excess N_2 = 77.25 - 64.69 = 12.56

Excess CO_2 = 3.40 - 0.025 = 3.375

Blackdamp = 12.56 + 3.375 = 15.94

Example (GATE 2020)

A sealed-off area air sample consists of 16.0% O_2, 3.0% CO_2, and the rest is N_2. Assume that the standard composition of atmospheric air is 21.0% O_2 and 79.0% N_2. The percentage of blackdamp in the air sample is —— (round off to 2 decimal places).

Solution

CO_2 eq = CO_2 wrt O_2

$$= \left(\frac{CO_2^{std}}{O_2^{std}}\right) O_2^{ret} = \left(\frac{0}{21.00}\right) O_2^{ret} = 0$$

Because CO_2 = 0%; O_2 = 21% in standard air.

N_2 eq = N_2 wrt O_2

$$= \left(\frac{N_2^{std}}{O_2^{std}}\right) O_2^{ret} = \left(\frac{79}{21.00}\right) 16 = 60.19$$

Because N_2 in standard air = 79%

Excess N_2 = 79 – 60.19 = 18.81

Excess CO_2 = 3.00 – 0.00 = 3.00

Black damp = 18.81 + 3.00 = 21.81

Detection of Carbon Monoxide

The P. S. detector

This detector consists of a glass tube 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 (120 cm^3) is drawn through the tube at a constant rate over a period of two minutes through a calibrated orifice by operating a rubber aspirator. Carbon monoxide in the air turns the light yellow colour of potassium palladium sulphite to brown and the length of colour change from one end of the tube indicates its concentration. Range of the instrument is 0.005% to 0.12% of CO.

The Hopcalite detector

The generation of heat on the oxidation of carbon monoxide to carbon dioxide has been utilised by a this detector.

The analyzing cell has two compartments, one filled with **active Hopcalite** catalyst and the other with **inactive Hopcalite**. Only the active catalyst will oxidize carbon monoxide.

A set of differential thermocouples, connected in series, extends through the partition separating the two compartments. One set of junctions of the thermocouples is embedded in the active Hopcalite, the other set in the inactive Hopcalite.

Carbon monoxide in the air passing through the cell is oxidized in the compartment containing active Hopcalite liberating heat, which subjects that set of junctions to a temperature higher than that of the set in the inactive catalyst, thus generating an electrical potential in the thermocouple circuit. The potential is indicated on a millivoltmeter that has a scale representing carbon monoxide in concentrations ranging from 0 to 0.15 percent.

Drager Multigas Detector

The Drager multigas detector is used to test the atmosphere within a space for toxic gases such as carbon dioxide, carbon monoxide, nitrogen dioxide, and hydrogen sulfide.

A separate detector bulb is used for each type of gas.

The unit consists of two main components— the bellows pump and the detector tube.

HYDROGEN SULPHIDE (STINK DAMP)

Hydrogen sulfide, often called "**stink damp**" because of its odour, which resembles that of rotten eggs, is a colourless, toxic, and explosive gas formed by the decomposition of sulfur compounds. Low concentrations may be found in air from heated gobs or may be released from water seeping in from the strata. Large concentrations occur in natural gas and oil fields and in some sulfur and gypsum mines. Hydrogen sulfide is quite soluble in water and may be carried into active mine workings by groundwater. It is slightly heavier than air and is explosive in air in the range of 4-44%.

Hydrogen sulfide is extremely toxic. Although hydrogen sulfide has a distinctive odour, the sense of smell cannot be relied upon as a means of detection, because after one or two inhalations, the olfactory nerves become paralyzed and the odour can no longer be detected. The lowest published lethal concentration is 600 ppm.

Concentration	Symptom
0.025 ppm	Threshold of odour
0.005-0.010%	Slight symptoms such as eye and respiratory-tract irritation after 1 h
0.010%	Loss of odour after 15-min exposure
0.02-0.07%	Increased eye irritation, headache, dizziness, nausea, dryness, and pain in nose, throat, and chest
0.07-0.10%	Unconsciousness, cessation of respiration and death
0.10%	Death in a few minutes

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%.

Detection of H_2S

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.

METHANE

The most common contaminant gas found in coal mines is methane. Although frequently associated with coal and other carbonaceous rocks, methane is also found in some non coal mines.

Methane is colourless, odourless, tasteless, non-toxic, highly flammable, and lighter than air. This last attribute results in methane accumulations forming along rooflines and in high areas of mines.

During the formation of a coal bed (coalification), methane is produced along with carbon dioxide, higher hydrocarbons, and other inert gases. Increasing pressure and temperature during coalification tends to proceed toward early total elimination of oxygen with concurrent removal of some hydrogen and carbon, and then total removal of hydrogen with concurrent removal of some carbon. While carbon dioxide is a significant constituent of seam gas in brown coal, the main constituent of higher-rank coal seam gas is methane. The volume of these by-product gases increases with the rank of coal and is the highest for anthracite, where, for every ton of coal formed, nearly 565 m^3 of carbon dioxide and 765 m^3 of methane is produced. A large fraction of these gases is lost during burial of the decayed plant material.

The amount of gas retained per ton of coal is known as the **seam gas content**.

One popular index of seam gas content in mines has been **specific methane emission**, expressed as the volume of methane emitted per ton of coal produced.

Methane emission equation

The equation is:

Methane in intake air + methane emitted = methane in return air

Ventilating air required will be given by

$$Q_v.\frac{C_i}{100}+q_g=(Q_v+q_g).\frac{C_{r,\max}}{100}$$

or $$Q_v=\frac{100q_g}{C_{r,\max}-C_i}-q_g$$

where

Q_v = ventilating air required, m^3s^{-1}

q_g = rate of gas emission, m^3s^{-1}

C_i = gas % in intake air

$C_{r,max}$ = permissible maximum gas % in return air

Example (DGMS FCMC 2021)

In a longwall face, % of methane at intake and return gates indicate 0.1 % and 0.4 %, air quantity flowing at the face is 1800 m^3/min, production from the face is 1500 TPD, methane emission rate in m^3/tonne of coal output is equal to ——.

Solution

Using $$Q_v=\frac{100q_g}{C_{r,\max}-C_i}-q_g$$

Given that

Q_v = 1800 m^3/min = 30 m^3/s

C_i = 0.1%

$C_{r,max}$ = 0.4%

Production capacity = 1500 tpd

q_g = ?

Putting values

$$30=\frac{100q_g}{0.4-0.1}-q_g$$

$$30 = 333.33q_g - q_g$$

$\therefore q_g = 0.09\ m^3/s$ per 1500 t

$\therefore q_g = 7766\ m^3/day$ per 1500 t

$= 7766/1500\ m^3/day$ per tonne

$= 5.184\ m^3/day$ per tonne

Example (GATE 2015)

Intake air containing 0.2% methane enters a section of an underground mine where emission of methane is 0.05 m^3/s. Assuming that the threshold limit value of methane is 1.25%, the minimum quantity of fresh air required in m^3/s is ———.

Solution

We know that

$$Q_v.\frac{C_i}{100} + q_g = (Q_v + q_g).\frac{C_{r,\max}}{100}$$

Given that

Q_v = ?

$C_i = 0.2\%$

$C_{r,max} = 1.25\%$

$q_g = 0.05\ m^3/s$

Putting these values

$$Q_v.\frac{0.2}{100} + 0.05 = (Q_v + 0.05).\frac{1.25}{100}$$

$\therefore$ $0.2Q_v + 5 = (Q_v + 0.05) \times 1.25$

$\therefore$ $0.2Q_v + 5 = 1.25Q_v + 0.0625$

$\therefore$ $Q_v = 4.9375/1.05 = 4.70\ m^3/s$

Methane Layering

Methane is a lighter gas with specific gravity of 0.554. Hence, methane emitted into underground coal mine atmosphere tends to stratify in the form of horizontal layers near roof level. This phenomenon is called **methane layering**.

Methane layering number

A dangerous effect called layering has been observed in coal mines with methane. If the ventilation airstream generates insufficient turbulence, any methane present remains stratified and forms a persistent layer moving along the roof.

It is given as

$$L = \frac{V}{1.8}\left(\frac{b}{Q}\right)^{1/3}$$

where V is air velocity in the upper half of the airway in (m/s), Q is methane inflow in (m^3/s), and b is airway width in (m).

Example

Given the following data,

V – 0.152 m/s

Q – 0.00708 m^3/s

b – 3.96 m

Determine the layering number and whether the air velocity is adequate to control layering in a horizontal airway.

Solution

$$L = \frac{V}{1.8}\left(\frac{b}{Q}\right)^{1/3}$$

$$= \frac{0.152}{1.8}\left(\frac{3.96}{0.00708}\right)^{1/3}$$

$$= 0.70$$

which is substantially less than the value (~5) needed to control layering under the given conditions.

Firedamp

Firedamp is a term used to describe a mixture of gases, primarily methane (CH_4), that can accumulate in coal mines. It is a highly flammable and explosive gas that poses significant safety risks to miners. When ignited, firedamp can cause devastating explosions, leading to loss of life and injury. Normally 10% of nitrogen, 5 % of higher hydrocarbons and 3-5 % of carbon dioxide may be found in firedamp.

Firedamp or methane is usually found in coal mines-and sometimes in tunnels passing through carbonaceous shales.

Flame safety lamp

In environments containing methane, a flame safety lamp will continue to burn at lower oxygen concentrations; however, at oxygen contents of less than 13%, the lamp will be extinguished regardless of the methane content of the air. It should be recalled that at an oxygen content of approximately 13%, there may be a loss of consciousness if the exposure is prolonged.

Test for methane is usually done in two stages. The two stages are:

- Accumulation test
- Percentage test

Accumulation test

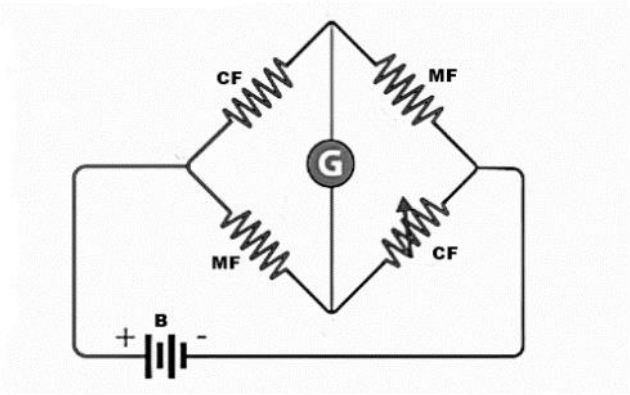
It is carried out with a luminous flame of standard height. This test gives a better indication of methane. This is because luminous-flame test produces more heat inside the lamp which causes better lamp ventilation resulting in a larger quantity of methane being drawn inside the lamp. Also for carrying out accumulation test it is not necessary to create a dark surrounding. The test carried out indicates that the flame spires/jump, if the percentage of methane in the air is 3% or more. It is because of this reason that we have to raise the lamp to the roof very slowly so that we can see the behaviour of flame as well. If the flame spires/jumps, it indicates that the methane content in the air is more than 3% and percentage test is not necessary in this case.

Percentage test

This test is also called cap test. The surrounding is made dark, before carrying out this test. This test is carried out with a reduced flame. The flame can be reduced with the help of the regulating knob. The flame should be reduced to an extent such that there appears a continuous blue line just above a speck (a very small mark or shape) of white/yellow light. The lamp is raised slowly to the roof and the percentage of methane is indicated by the height of the cap produced.

Methanometer

Methane Detectors Handheld methane detectors or methanometers use two basic methods of detection: catalytic oxidation and optical interferometry. Catalytic-oxidation detectors operate on the Wheatstone bridge principle.



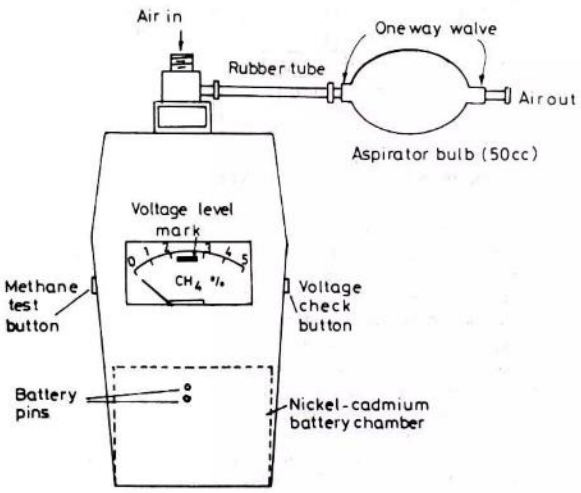
In the conductivity type methanometer shown above, following are labels used.

B Battery

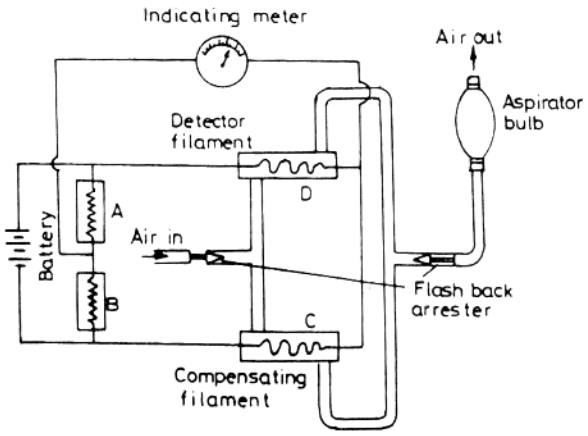
CF compensating filaments sealed in aluminium chamber and filled with atmospheric air heated to 343K by a battery

MF measuring filaments enclosed in aluminium chamber filled with methane-air mixture to be tested.

An MSA D6 methanometer is shown below.



Its schematic diagram is shown below.



Degree of gassiness

Gassiness of a belowground coal seam is characterized by the quantity of emission of methane from the seam and is usually expressed as relative emission, that is, cubic meter of methane emitted per tonne of coal produced.

All coal seams in India are treated as gassy and are categorized into three different degrees of gassiness by the Directorate General of Mines Safety, Dhanbad, India under the Coal Mines Regulations 1957 (DGMS, 1957). Classification criteria for categorization of degree of gassiness of belowground seams is described below:

- "**Degree I** Seams" means a coal seam in which the percentage of inflammable gas in the general body of air does not exceed 0.1 and the rate of emission of such gas does not exceed one cubic meter per tonne of coal produced.
- "**Degree II Seam**" means a coal seam in which the percentage of inflammable gas in the general body of air at any place in the workings of the seam is more than 0.1 or rate of emission of inflammable gas per tonne of coal produced exceeds one cubic meter but does not exceed ten cubic meters.
- "**Degree III Seams**" means a coal seam in which the rate of emission of inflammable gas per tonne of coal produced exceeds ten cubic meters.

METHANE DRAINAGE

Methane drainage is the process of removing the gas contained in the coal seam and surrounding strata through pipelines.

The organized extraction of methane from carboniferous strata may be practiced in order

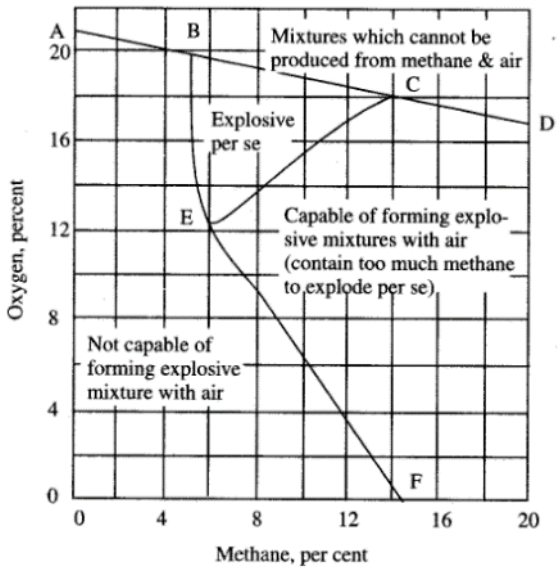
- to produce a gaseous fuel,
- to reduce methane emissions into mine workings or
- to carry out a combination of the two.

Methods of methane drainage

- Cross-measures Methane Drainage
- In-Seam Gas Drainage
- Gob drainage by surface boreholes

Coward flammability diagram

Coward was the first to represent graphically the relationship between the quantitative composition and the flammability of mixtures of methane and air. In the diagram (See figure) one distinguishes the following four areas:



1. An area above the line AD in which no mixture of methane and air is possible impossible mixtures range). The point A represents the normal air of 20.93 per cent O_2 and 79.07 per cent N_2 and CO_2.
2. An area to the left of the line BEF in which methane does not form any flammable mixture with air at all (non-flammable or safe range). The line BE represents the lower limits of flammability of methane-air mixtures.
3. An area to the right of the line CEF in which too rich methane can form flammable mixtures when mixed with air in suitable proportions (possible flammable range). The line CE represents the upper limits of flammability of methane air mixtures. As the oxygen content of the mixture decreases, BE and CE approach each other and meet at, E which is called the 'nose flammability limit'.
4. A triangular area BEC known as the Coward triangle, in which any methane-air mixture is flammable (flammable range). The shape of the Coward triangle does not alter with the percentage of CH_4 in the sample.

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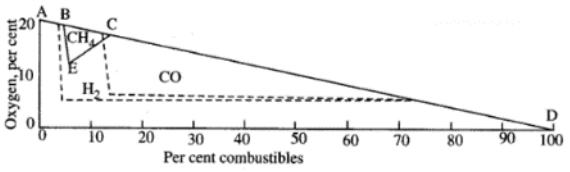
It can be seen from the diagram that no mixture that contains less than about 12.1 % oxygen is explosive by itself.

Coward triangles for gases other than methane can also be drawn. Following figure shows the diagram with flammability triangles for CH_4, CO, and H_2.



UEL (Upper explosibility limit) can be computer from following equation

$$P_i/UEL = (P_1/L_1) + (P_2/L_2) + (P_3/L_3)$$

where $P_i = P_1 + P_2 + P_3$

Example

Construct the Coward flammability diagram and classify the following samples of "normal air and methane mixtures" in terms of their explosibility characteristics:

(i) CH_4: 5% and O_2: 12%

(ii) CH_4: 10% and O_2: 10%

(iii) CH_4: 7.5% and O_2: 16%

(iv) CH_4: 10% and O_2: 20%

Answer

(i) Not capable of forming explosive' mixture with air

(ii) Capable of forming explosive mixtures with air

(contain too much methane to explode per se).

(iii) Explosive per se

(iv) Mixtures which cannot be produced from methane & air

Example (GATE 2022)

The point 'A' as shown in the Coward flammability diagram represents the gas composition of a sealed-off area of a coal mine. The volume of the sealed-off area is 10000 m^3. Inert gas is proposed to be injected into the sealed-off area so that the gas composition comes below the LEL (lower explosibility limit). The minimum volume of the inert gas required (at the same pressure as that of the sealed-off area), in m^3 is_____. (round off to one decimal place).

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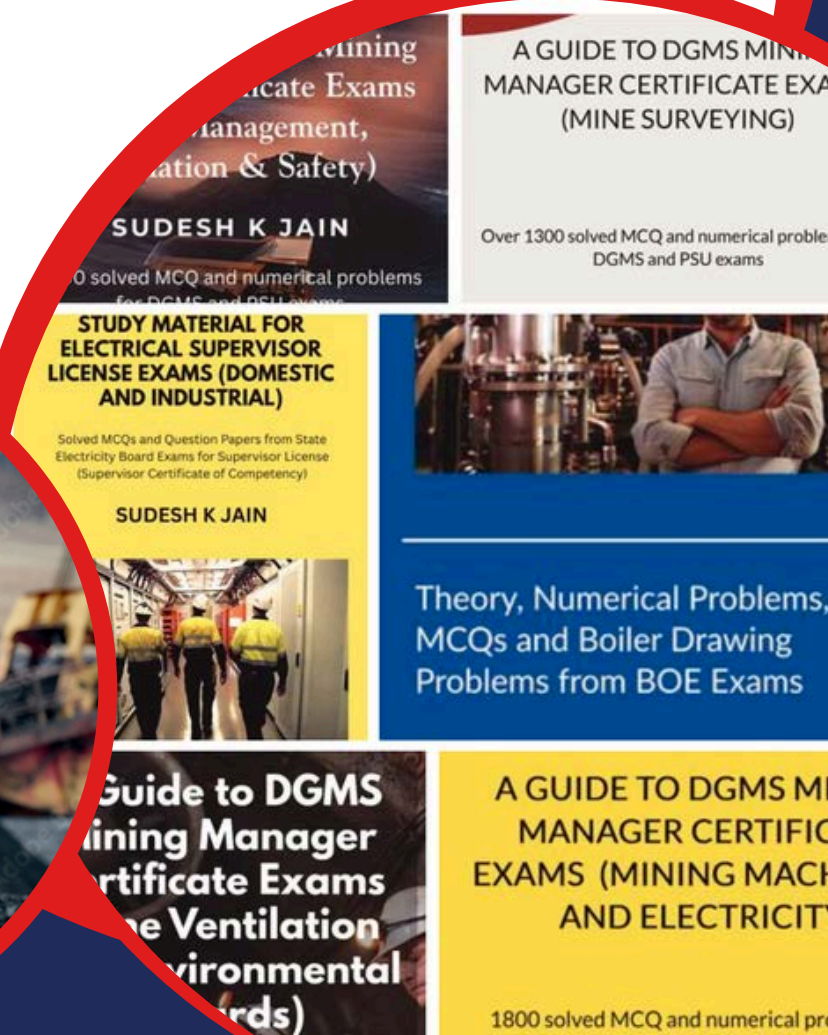
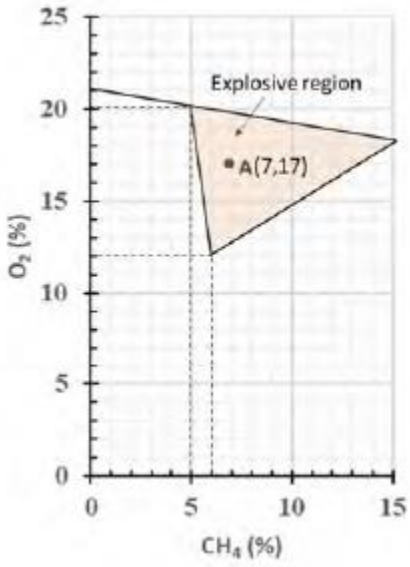




Solution

$$V_{excess\ inert} = V[(C_{methane}/C_{methane,\ cr}) - 1]$$
$$= 10{,}000 \times [(7/5.8) - 1]$$
$$= 2068\ m^3$$

Here $C_{methane,cr}$ is the value of methane at nose which is taken as 5.8%. That of oxygen is taken as 12.1%.

DUSTS

Dust consists of solid particulate matter *suspended* in a gas. Dusts constitute the most common aerosol problem in the mineral industries and are usually formed by fragmentation processes such as drilling, crushing, and grinding but can also result from resuspension due to equipment operation or air movement. Dusts particles vary from 1 to 100 μm (micrometer) in diameter, but the size range is normally 1-20 μm. This is a result of the fact that particles below 1 μm are not formed in abundance, and particles above 20 μm are usually relatively quick to settle.

Physiological effects of dusts

- Breathing in dust, such as coal dust, silica dust and other finely powdered materials, can damage the lungs and airways.
- Exposure to dust can cause irritation to the eyes, skin and respiratory tract, and prolonged exposure can lead to a range of serious lung diseases including silicosis, coal workers' pneumoconiosis (CWP), chronic obstructive pulmonary disease (COPD) and lung cancer.

- Different types of dust particles have different health effects. For example, respirable crystalline silica dust causes scarring of the lungs, and inhalable lead dust can damage the central nervous system.
- Many occupational diseases are the result of many years of exposure to dust and it may take years or decades before the disease becomes noticeable.
- High levels of dust can reduce visibility and become a safety hazard.
- Coal dust is an important cause of underground explosions.

Dynamics of airborne particles

Particle diameter more than 1 μm

Stokes' law is used to determine the settling velocity of dust particles greater than 1 μm in diameter that are falling in a quiescent fluid with Reynolds numbers less than 1.

Terminal velocity is given as

$$V_t = \frac{\rho_p D_p^{\,2} g}{18\mu}$$

where ρ_p is density of the particle in kg/m^3, D_p is diameter of the particle in m, g is the gravity constant = 9.807 m/s^2, and (μ is viscosity of air = 1.81 x 10^{-5} Pas. This equation applies if the particle diameter D_p is greater than 1 μm and the particle Reynolds number is less than or equal to 1.0.

Reynolds number is given as

$$\mathrm{Re} = \frac{\rho_p V_t D_p}{\mu}$$

Particle diameter less than 1 μm

In such a condition, we will apply following relation given by Cunningham.

$$v_{true} = v_{stokes}\left(1+1.7\frac{l}{D}\right)$$

where v_{true} = true terminal velocity

v_{stokes} = Stokes terminal velocity

l = mean free path of gas molecules (~ 10^{-7} m)

Example

If a 0.5 μm diesel particle is released into quiescent air at a height of 2 m above the floor of a mine opening, how long will it take for the particle to fall to the floor of the opening if it has a specific gravity of 0.6? Assume that the mean free path for air is 6.6 x 10^{-8} m.

Solution

Given that

D_p =0.5 μm = 0.5 x 10^{-6} m

ρ_p = 0.6 g/cm^3 = 600 kg/m^3

$$V_{st} = \frac{\rho_p D_p^{\ 2} g}{18\mu}$$

$$= \frac{(600)(0.5x10^{-6})^2(9.81)}{18(1.81x10^{-5})} = 45.42x10^{-7}\, m/s$$

$$V_{true} = V_{stokes}\left(1+1.7\frac{l}{D}\right)$$

$$= 45.42x10^{-7}\left(1+1.7\frac{6.6x10^{-8}}{0.5x10^{-6}}\right)$$

= 4.542 x 10^{-6} x 1.224

= 5.56 x 10^{-6} m/s

Diseases due to Dust in Mines

The physical ailments that result from aerosols in the mine environment range from simple irritation of the respiratory system due to overloading the lungs with dust to death from lack of respiratory function or cancer.

The most common form of lung dysfunction found among miners is pneumoconiosis, a condition characterized by the formation of fibrous tissue in the lungs arising from dust particles deposited in the lungs. Pneumoconiosis is a general term; specific names are generally attached to the resultant maladies, depending on the specific causal material. Some of the specific terminology for the more prevalent mineral-related diseases and the dusts believed to be causal are as follows:

- Silicosis (dusts of quartz, tridymite, and cristobalite)
- Silicate pneumoconiosis (dusts of silicate minerals such as kaolin, talc, tremolite, actinolite, and anthophyllite)
- Asbestosis (asbestiform dusts such as dusts of amosite, chrysotile, and crocidolite)
- Coal workers' pneumoconiosis (coal dust)
- Beryllium disease (dusts of beryllium compounds including ores)
- Siderosis (dusts of iron including ores)

Of these diseases, silicosis and coal workers' pneumoconiosis are of most importance in mines.

PSYCHROMETRIC PROPERTIES OF AIR

Humidity

It is defined as the water-vapour content of the air.

Normal atmospheric air in most of the cases is humid. Humid air is also called moist air. The capacity of air to hold moisture increases with temperature. But, one should not be in confusion that in summer, air should be more humid compared to other seasons. This is because air may have more capacity to hold water, but we require source of water as well. Also, even if air has high water content at higher temperature, but it may not be saturated or have higher relative humid compared to air having low water vapor content but more relative humidity at lesser temperature. This is because relative humidity is not only governed by temperature of air/atmosphere (dry bulb temperature) but also wet bulb temperature.

Now, let us discuss the different ways of expressing humidity.

Relative Humidity

It is defined as the ratio of vapour pressure at a temperature to the saturation vapour pressure at that dry bulb temperature. It should be kept in mind that for calculating relative humidity, saturation vapour pressure is taken at dry bulb temperature and not at wet bulb temperature. Numerically it can be expressed as:

$$\text{Relative humidity} = (e/e_{sd}) \times 100\%$$

Specific Humidity

It is defined as the mass of water vapour present in kg per kg of dry air. Mathematically it is expressed as follow.

$$Sp\ Heat = 0.622 \frac{e}{P_b - e} kg / kg\ der\ air$$

Where,

e = Vapour pressure or Partial pressure due to water vapour (kPa)

P_b = barometric pressure (kPa)

Absolute Humidity

It is defined as the amount of water vapor present in a unit volume of air, usually expressed in kilograms per cubic meter. It is mathematically expressed as

$$m' = \frac{10^2 e}{461.9T}$$

Where,

m' = absolute humidity (kg/m^3)

T= temperature in Kelvin (K)

e = Vapour pressure or Partial pressure due to water vapour (kPa)

This is rarely used in analysis. Volume of air passing through ventilation system keeps changing because of variation in temperature and pressure. Thus, use of absolute humidity is *discouraged*.

Dew Point

It is defined as the temperature at which air attains saturation and a further addition of water vapour leads to dew formation because of condensation of water vapour. It is rarely used to indicate the moisture content of the air/atmosphere. The temperature recorded in this case is *dry bulb temperature*. At dew point, dry bulb and wet bulb temperature are same.

Dry-Bulb Temperature

It is the temperature recorded by using a conventional thermometer. The thermometer without muslin cloth in the psychrometer records dry-bulb temperature. It just reads the ordinary temperature of the air and is a measure of sensible heat content of the air. Its unit is °F or °C or Kelvin (K).

Wet-Bulb Temperature

It is recorded by thermometer having wet muslin cloth on its bulb. The temperature recorded is in general lower than dry-bulb temperature because of cooling effect of the evaporating water of wet muslin cloth. They are equal only when air is in saturation and no net evaporation of water from wet muslin cloth takes place. Wet-bulb temperature can never be higher than dry-bulb temperature. From the definition point of view, it is defined as the temperature at which water vapour evaporating into the air can bring down the air in saturation adiabatically at that temperature. It is a measure of the evaporating capacity of the air. Its unit is °F or °C or kelvin (K).

Barometric pressure

It is simply the pressure recorded by a barometer at a particular place. It is usually expressed in kPa.

KATA THERMOMETER

The first Kata thermometer was developed in the 1920s by Sir Leonard Hill (UK) to determine the cooling power of an environment while measuring thermal comfort. This was an alcohol

filled thermometer with a large cylindrical bulb, with markings at 35°C and 38°C on the graduated stem.



In order to determine the total cooling power of air, the bulb is covered with a wet cotton cloth and the apparatus is heated in hot water (50 to 60° C) in a thermos flask. It is then wiped dry and suspended for the combined exposure to the prevailing temperature, air velocity and radiation, but not humidity. The time in seconds for the fall of the temperature from 38 to 35°C is noted. By dividing the calibration or the Kata factor, determined beforehand for the thermometer, by the total number of seconds it took to cool from 38 to 35°C, the Kata cooling power of air in W/m^2 is obtained.

The wet-kata cooling power is related to the wet bulb temperature and the air velocity by the following relations:

$$K = (14.65 + 35.59v^{1/2})(309.65 - T) \quad \text{for air velocity} < 1 \text{ m/s}$$

and

$$K = (4.19 + 46.05v^{1/2})(309.65 - T) \quad \text{for air velocity} > 1 \text{ m/s}$$

where K is kata cooling power in W/m^2

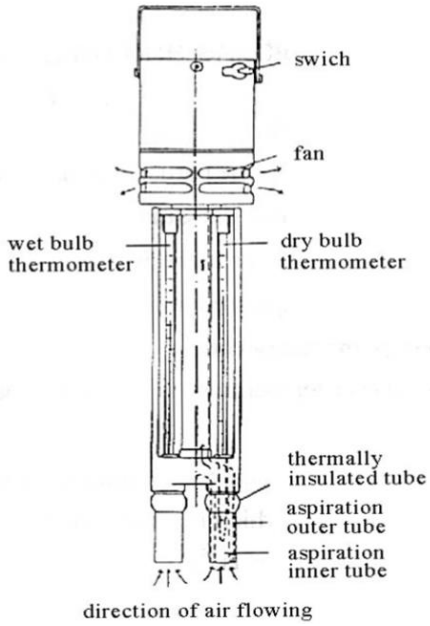
v = velocity of air in m/s

and T = wet bulb temperature in K.

MEASUREMETER OF HUMIDITY

Aspirated hygrometer (psychrometer)

The instrument consists of essentially two identical thermometers placed in sleeves, mounted side by side on a suitable frame, having a dry bulb to indicate the normal temperature of surrounding air, and a wet bulb kept continuously moist by a covering of fine cloth immersed in a reservoir, by which constant evaporation of moisture takes place thereby cooling it and depressing the wet bulb reading by an amount dependent on the degree of saturation of the air. The amount of saturation depends on the air pressure.



At the top of the two thermometers, a small spring-operated aspirator with clockwork mechanism is provided to suck in the air over them at a velocity of more than 2 m/s. When using the instrument, it must be held opposite to the flow direction to avoid temperature disturbances to the instrument caused due the body heat of the observer.

Whirling hygrometer (sling psychrometer)

The whirling hygrometer for measuring humidity consists of two thermometers that are turned by vigorously swinging the handle and exposing the thermometers to rapid air movement. The bulb of one thermometer is covered with a silk or muslin sleeve that is kept moist and this will record wet bulb temperature. The atmospheric humidity is determined by special tables from the difference in reading between the two thermometers.

ATKINSON EQUATION AND SQUARE LAW

Darcy relationship is given as

$$H = f\frac{L}{D}.\frac{V^2}{2g} \qquad (1)$$

where H is head loss in metre of fluid, L is length in metre, D is diameter in metre, V is velocity in m/s, and f is coefficient of friction.

A more versatile equation, one applicable to any shape of conduit, may be obtained by expressing the head loss in terms of the hydraulic radius R_h the ratio of the area A to the perimeter O of the duct. For a circular conduit,

$$R_h = \frac{A}{O} = \frac{(\pi/4)D^2}{\pi D} = \frac{D}{4} \quad (2)$$

From (1) $$H = f.\frac{L}{4R_h}.\frac{V^2}{2g} \quad (3)$$

It can be written as

$$H_f = \frac{KOLV^2}{A} \quad (4)$$

where H_f is friction loss in in. water (Pa), V is velocity in m/s, and K is an empirical friction factor in (kg/m^3).

It can be written as

$$H_f = \frac{KSV^2}{A} \quad (5)$$

where S is rubbing surface area in (m^2) = O x L

Often the air velocity is not known, but the quantity Q is given.

In such a case

$$H_f = \frac{KOLQ^2}{A^3} = \frac{KSQ^2}{A^3} \quad (6)$$

Giving $$\boxed{\Delta P = RQ^2} \quad (7)$$

This is known as the square law of mine ventilation and is probably the single most widely used relationship in subsurface ventilation engineering.

The parameter R is called the **Atkinson resistance** of the airway and, as shown by the square law, is the factor that governs the amount of airflow, Q, that will pass when an airway is subjected to any given pressure differential, ΔP.

POWER

Power of ventilation or air power is the rate at which work is done to maintain' the air-flow through a system.

Power = PQ = $(RQ^2)Q = RQ^3$

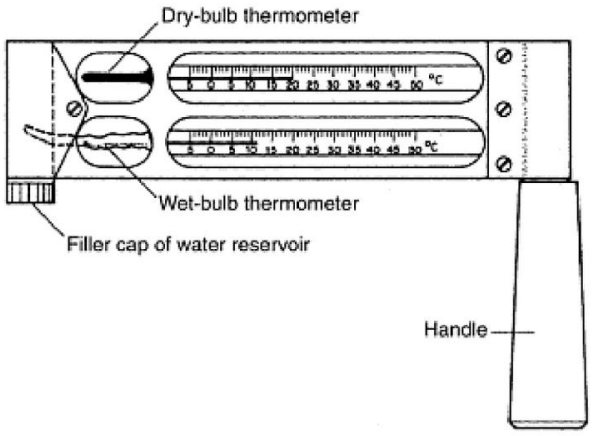
where P is pressure in Pa, Q is air quantity in m^3/s and power is in Watt.

The required power output of the· prime mover driving a fan causing the air-flow can be found by dividing the air power by the efficiency of the fan and gear.

DARCY WEISBACH RESISTANCE COEFFICIENT

$$f = \frac{8K}{\rho}$$

where f is Darcy Weisbach resistance coefficient, ρ is density of air in m^3/s and K is coeff. of friction in Ns^2m^{-4}



The hygrometer contains two mercury-in-glass thermometers, one whose sensor is covered in wet muslin (close meshed cotton) or silk (wet bulb) and one normal (dry bulb). The hygrometer is whirled, hence passing moving air over the two sensors. This allows an effective measure of air temperature (dry bulb) and a measure of aspirated (air moved across sensor) wet bulb temperature.

The decrease in wet bulb temperature due to heat loss by evaporation, for a given air temperature, is related to the humidity of the environment. The atmospheric humidity is determined by special tables from the difference in reading between the two thermometers.

Example

A mine has only one district at a distance of 500 m from the bottom of the shafts. The average size of roadways is 4.3 m x 2.7 m and the average velocity of air is 320 m/min. Find out the pressure difference at the pit–bottom. Value of k is 0.001.

Solution

Given data

$S = 2(4.3 + 2.7) \times 500 = 7000\ m^2$

$V = 320\ m/s = 320/60 = 5.33\ m/s$

$A = 4.3 \times 2.7 = 11.61\ m^2$

K = 0.001

Now $$P = \frac{KSV^2}{A}$$

$$= \frac{(0.001)(7000)(5.33)^2}{11.61}$$

= 17.2 Pa

SHOCK RESISTANCE

We have so far considered the case of frictional resistance to flow in pipes (due to boundary shear), but if the pipe contains an **obstruction** on the path of flow, which not only exposes an extra boundary to flow but also causes eddies and separation in the wake of the obstruction, there occurs a **pressure loss across the obstruction**. The resistance so offered by the obstruction is commonly termed as **shock -resistance**.

Shock pressure loss also occurs at points of **change in cross-sectional areas** as well as **bends** where eddies are formed and separation takes place.

The total pressure loss that takes place through shock can be obtained from equation

$\Delta P_{shock} = X\ P_v$

where

X = Shock loss factor (dimensionless).

P_v = Velocity pressure (Pa) = $\frac{1}{2}\rho v^2$

Here ρ is air density.

$X = 0.5\left(1 - \frac{A_{small}}{A_{bigger}}\right)^2$ when diameter contracts.

$X = \left(1 - \frac{A_{small}}{A_{bigger}}\right)^2$ when diameter expands.

For bends

$$X = \frac{0.25}{R^2\sqrt{a}} \cdot \left(\frac{2i}{\pi}\right)^2$$

where R = radius ratio = r/W where r is radius of curvature and W is width.

a = aspect ratio = H/W where H, is height.

i = angle of deflection in radian (e.g. $90^0 = \pi/2$ radian)

For square bends in rectangular airways

$$X = \frac{0.6}{R\sqrt{a}} \cdot \left(\frac{2x(\pi/2)}{\pi}\right)^2 = \frac{0.6}{R\sqrt{a}} \text{ for } 90^0 \text{ bends in rectangular airways}$$

and $$X = \frac{0.6}{R\sqrt{1}} = \frac{0.6}{R} \text{ for } 90^0 \text{ bends in square airways.}$$

Shock factor for area change (like when there is a door frame in airway)

$$X_a = \frac{\left(\frac{1}{C} - N_a\right)^2}{N_c^{\,2}}$$

where $N_c = A_0 / A_a$

and $$C = \sqrt{\frac{1}{Z - ZN_c^{\,2} + N_c^{\,2}}}$$

Z = 2.5 for door.

Example

A rectangular mine airway of 2.0 m width and 2.5 m height has a bend with deflection of π/4 radian. If the radius of curvature of the bend is 4.0 m, find the shock factor of the bend.

Solution

Shock factor for a bend is

$$X = [0.25/R^2\sqrt{a}]\ (2i/\pi)^2$$

where R =radius ratio = r/W where r is radius of curvature and W is width.

a = aspect ratio = H/W where H is height.

i = angle of deflection in radian

Given values are:

W = 2 m

H = 2.5 m

r = 4 m

i = π/4 radian

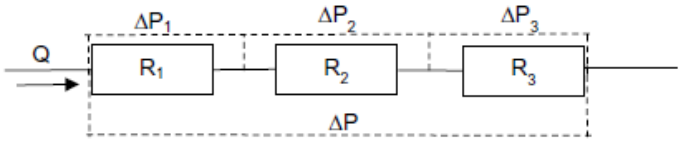
Now, R = r/W = 4/2 = 2

a = H/W = 2.5/2 = 1.25

Hence, $X = [0.25/(2^2)\sqrt{1.25}][(2 \times \pi/4)/\pi]^2 = 0.014$

RESISTANCE (AIRWAYS) IN SERIES

See following figure.



When the airways are joined in series, the total pressure drop across all of them is the sum of pressure drops across each of them while, the same quantity Q flows through each of them.

We see that

$$\Delta P = \Delta P_1 + \Delta P_2 + \Delta P_3 \quad (1)$$

$$Q = Q_1 = Q_2 = Q_3 \quad (2)$$

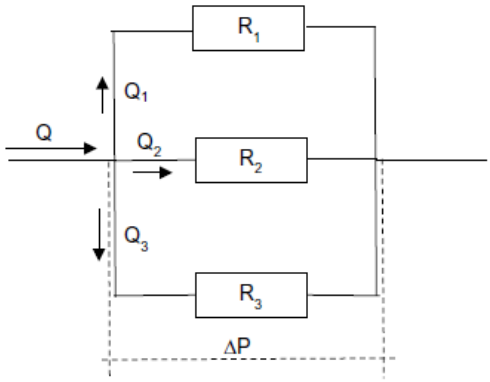
In eq. (1), dividing both sides by Q^2, we have

$$\frac{\Delta P}{Q^2} = \frac{\Delta P_1}{Q^2} + \frac{\Delta P_2}{Q^2} + \frac{\Delta P_3}{Q^2} + ... \frac{\Delta P_n}{Q^2}$$

Giving $R_{eq} = R_1 + R_2 + R_3 \quad (3)$

RESISTANCES (AIRWAYS) IN PARALLEL

See following figure.



$$Q = Q_1 + Q_2 + Q_3 \quad (1)$$

$$\Delta P = \Delta P_1 = \Delta P_2 = \Delta P_3$$

$$= = R_1 Q_1^2 = R_2 Q_2^2 = ...R_n Q^2{}_n \quad (2)$$

Combining (1) and (2)

$$\sqrt{\frac{\Delta P}{R}} = \sqrt{\frac{\Delta P}{R_1}} + \sqrt{\frac{\Delta P}{R_2}} + ... \sqrt{\frac{\Delta P}{R_n}}$$

Giving $$\frac{1}{\sqrt{R_{eq}}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}}$$

Example

An air flow of 150 m^3/s is passing through two splits. Cross section of first airway is 2.5 m x 1.5 m and that of second airway is 2.0 x 1.5 m. First airway is 100 m long and second airway is 125 m long. Calculate the quantity of air passing in each split.

Solution

We know that

$$P = \frac{KSQ^2}{A^3}$$

S is perimeter x length, Q is air flow discharge and A is cross sectional area.

As these airways are in parallel, pressure loss in both will be same and discharge will be different.

$\therefore$ $$\frac{S_1 Q_1^2}{A_1^3} = \frac{S_2 Q_2^2}{A_2^3}$$ [same pressure]

and $150 = Q_1 + Q_2$ [different discharge]

K is cancelled out being same in both cases.

Given data

$S_1 = 2(2.5 + 1.5) \times 100 = 800\ m^2$

$S_2 = 2(2 + 1.5) \times 125 = 875\ m^2$

$A_1 = 2.5 \times 1.5 = 3.75\ m^2$

$A_2 = 2 \times 1.5 = 3\ m^2$

Now $$\frac{800 Q_1^2}{3.75^3} = \frac{875 Q_2^2}{3^3}$$

which gives

$Q_1 = 0.78 Q_2$ (1)

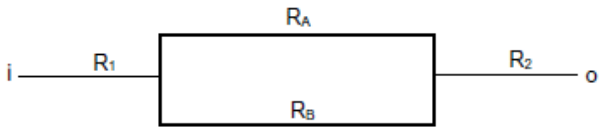
Also $150 = Q_1 + Q_2$ (2)

$Q_2 = 84.27\ m^3/s$

$Q_1 = 0.78 \times 84.27 = 65.73\ m^3/s$

Example

A mine ventilation network is given below. Resistances of the inlet (R_1) and outlet airways (R_2) are 0.25 N s^2 m^{-8}, respectively, and that the pressure difference between the inlet (i) and the outlet (o) is 700 Pa.



It is known also that, the resistance of parallel branches A and B are R_A = 0.8 Ns^2m^{-8} and R_B = 3.6 N s^2 m^{-8}, respectively Determine the volumetric airflow rates traversing each airway.

Solution

$$\frac{1}{\sqrt{R_{parallel}}} = \frac{1}{\sqrt{0.8}} + \frac{1}{\sqrt{3.6}}$$

Giving $R_{parallel} = 0.37\ Ns^2/m^8$

Now Total resistance

$$R_T = R_1 + 0.37 + R_2 = 0.25 + 0.37 + 0.25 = 0.87\ Ns^2/m^8$$

$\Delta P = 700$ Pa

From $P = RQ^2$

$$700 = 0.87Q_T^2$$

Giving $Q_T = 28.37\ m^3/s$

Now $Q_T = Q_A + Q_B = 28.37\ m^3/s$

$$\Delta P_{parallel} = R_{parallel} \times Q_T^2$$

$\therefore$ $\Delta P_{parallel} = 0.37 \times 28.37^2 = 297.8$ Pa

Also we know

$$\Delta P_{parallel} = \Delta P_A = \Delta P_B$$

$\therefore$ $\Delta P_A = \Delta P_B = 297.8$ Pa

Consider branch with R_A

$$\Delta P_A = R_A Q_A^2$$

$\therefore$ $297.8 = 0.8 \times Q_A^2$

Giving $Q_A = 19.29\ m^3/s$

Similarly for branch with R_B

$$297.8 = 3.6 \times Q_B^2$$

$$\therefore \quad Q_B = 9.06 \text{ m}^3/\text{s}$$

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.

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

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

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.

Example

Three splits in parallel, of similar cross section and same type of roadway surface, are respectively 300 m, 600 m and 900 m long. Calculate the quantity of air which would flow in each if the total quantity is 200 m^3/min.

Solution

Ratios of quantities

We know $$P = \frac{KSQ^2}{A^3}$$

$$\therefore \quad Q^2 = \frac{PA^3}{KS} = \frac{PA^3}{K \, x \, L \, x \, perimeter}$$

Now, A, K and perimeter are constants.

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

$$\therefore \qquad Q_1 : Q_2 : Q_3 = \frac{1}{\sqrt{300}} : \frac{1}{\sqrt{600}} : \frac{1}{\sqrt{900}} = \frac{1}{\sqrt{1}} : \frac{1}{\sqrt{2}} : \frac{1}{\sqrt{3}} = 1 : 0.709 : 0.579$$

Quantities in each split

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

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

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

$\therefore$ Quantity in split no. 3 = 200 - (88.03 + 61.97) = 50 m^3/min

EQUIVALENT ORIFICE OF A MINE

In the actual case of an entire mine, the equivalent orifice is the area of the hypothetical circular thin-walled orifice located in the main fan drift through which the main fan is capable of extracting the same quantity when the airlock at the top of the shaft is open (see figure b), as from the mine when the airlock at the top of the shaft is closed (see figure a). Thus, the larger the section of the equivalent orifice compared to that of the airway, the easier it will be to ventilate it.

Let Q = quantity of air flowing through the mine in m^3/s

A = area of the orifice in m^2

A_v = area of flow at the vena contracta in m^2 = 0.6A

v = velocity of flow at the vena contracta and

h = head of air causing flow in m.

The equation is

$$\boxed{A = \frac{1.2}{\sqrt{R}}}$$

Several formulae emerged from above discussion are given below:

Formula #1

$$A = \frac{Q}{0.6\sqrt{2gH}}$$

where A is equivalent orifice area.

Formula #2

$$A = \frac{1.2Q}{\sqrt{P}}$$

Formula #3

$$A = \frac{1.2}{\sqrt{R}}$$

Mines with an equivalent orifice of more than 5 m^2 are generally considered to be very easy to ventilate and orifices smaller than 1.5 m^2 indicate mines difficult to ventilate.

Example

In a mine having equivalent orifice of 2.4 m^2, the water gauge is 25 mm. What will be new equivalent orifice if the water gauge is increased to 40 mm?

Solution

We know

$$A = \frac{1.2Q}{\sqrt{P}}$$

If discharge remains same

$$\frac{A_1}{A_2} = \sqrt{\frac{P_2}{P_1}}$$

Putting values

$$\frac{2.4}{A_2} = \sqrt{\frac{40}{25}}$$

Giving $A_2 = 1.90\ m^2$

Example

The main fan of a mine supplies a static pressure of 4 kPa, generating an air quantity of 50 m^3/s. Determine the equivalent orifice of that mine. Comment on the result.

Solution

$$A = \frac{1.2Q}{\sqrt{P}} = \frac{1.2}{\sqrt{R_{reg}}}$$

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P = 4 kPa = 4000 Pa

Q = 50 m^3/s

$$\therefore \qquad A = \frac{1.2(50)}{\sqrt{4000}} = 0.95\,m^2$$

This is less than 2 m^2, hence this mine is not easy to ventilate. For easy ventilation, it must be more than 5 m^2.

REGULATION OF AIRFLOW BY BOOSTER FAN

A booster fan is installed in **series** with the main mine fan to improve the ventilation of a split having resistance greater than others. The booster fan is specially recommended when it is very difficult or practically impossible to increase the quantity of ventilation by building regulators or enlarging the original airways from cost considerations, by speeding up the surface mine fan or installing a new surface fan.

Example

A mine fan generates pressure of 500 Pa which is sufficient to circulate 25 m^3/s air through the mine which consists of two splits A and B. A circulating 15 m^3/s and B, 10 m^3/s. It is desired to increase the quantity in split B to 15 m^3/s by installing a booster in it. Calculate the size of the booster if the resistance of the shafts and trunk airways is 0.2 Ns^2m^{-8}.

Solution

Let Q = air flow in mine before installation of booster

R_A = resistance of split A

R_B = resistance of split B

R_T = resistance of shaft and trunk airways

P_B = pressure by booster fan

Q_A = Quantity in split A after installing booster fan

Q_B = Quantity in split B after installing booster fan

Q_T = Total air flow in the mine after installing booster fans

Let us calculate resistances

$$500 = R_TQ^2 + R_AQ_A^2 = 0.2(25)^2 + R_A(15)^2$$

From this, $R_A = 1.67\ Ns^2m^{-8}$

Similarly

$$500 = R_TQ^2 + R_BQ_B^2$$

$$500 = 0.2(25)^2 + R_B(10)^2$$

$\therefore$ $$R_B = 3.75\ Ns^2m^{-8}$$

Now, after introducing the booster fan in split B

Consider circuit R_T-R_A

$$500 = R_TQ_T^2 + R_AQ_A^2 = R_TQ^2 + R_A\ (Q_T - Q_B)^2$$

Putting values

$$500 = 0.2(Q_T)^2 + 1.67(Q_T - 15)^2$$

It can be written as

$$1.87Q_T^2 - 50.1Q_T - 124.25 = 0$$

Comparing this equation with $ax^2 + bx + c = 0$

$$a = 1.87;\ b = -50.1;\ c = -124.25$$

Now $$Q_T = \frac{-b \pm \sqrt{b^2 - 4ac}}{2a}$$

$$= \frac{-(-50.1) \pm \sqrt{(-50.1)^2 - 4(1.87)(-124.25)}}{2(1.87)} = 29\ m^3/s$$

To find out value of booster pressure P_B, let us consider circuit R_T–R_B

$$500 + P_B = R_TQ_T^2 + R_BQ_B^2$$

$\therefore$ $$500 + P_B = (0.2)(29)^2 + 3.75(15)^2$$

$\therefore$ $$P_b = 512\ Pa$$

REGULATOR

A regulator is an obstacle to the flow of a current which consists of an opening whose section can vary offering different resistances to the passage of air.

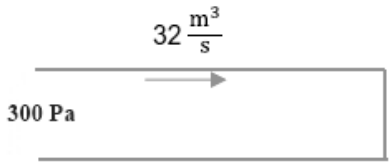
A regulator is a window of adjustable opening in a brick stopping. The shutter of the regulator can be locked in position to prevent tampering by workers. Introduction of a regulator in a roadway increases the resistance to air current and it should, therefore, be used on the return side of a district whose ventilation has to be reduced and fixed in a place where all the air of the district has to pass.

Regulator area can be computed from using equivalent orifice area formula.

Example

The mining district in the figure is traversed by an airflow rate of 32 m^3/s when the pressure difference between the intake and the return airways is 300 Pa. This airflow rate is to be

reduced to 27 m^3/s by means of a regulator installed in the return airway. Determine the area of the regulator.

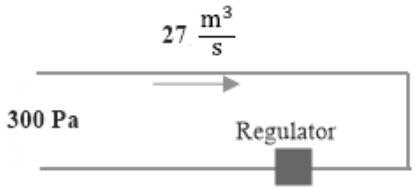


Solution

Let us calculate total resistance of the network with installing a regulator.

$$R_1 = \frac{P}{Q^2} = \frac{300}{32^2} = 0.29\,\text{Ns}^2/\text{m}^8$$

Now let us place a regulator as given in the question.



Let us calculate the resistance of the network with regulator

$$R_2 = \frac{P}{Q^2} = \frac{300}{27^2} = 0.41\,\text{Ns}^2/\text{m}^8$$

The difference of resistance in these two conditions will give resistance of the regulator.

$$R_{reg} = 0.41 - 0.29 = 0.12\ \text{Ns}^2/\text{m}^8$$

Now, we can calculate area of regulator.

$$A = \frac{1.2Q}{\sqrt{P}} = \frac{1.2(Q)}{\sqrt{R_{reg}Q^2}} = \frac{1.2}{\sqrt{R_{reg}}} = 1.2/\sqrt{0.12} = 10\ \text{m}^2$$

NATURAL VENTILATION

Natural ventilation is ventilation produced by natural pressure (kgf/m^2) generated by one or more the following causes:

- **Warming up of mine air** as it passes through the mine workings on account of rock temperature, oxidation processes, disposal of heat from machinery especially diesel engines, hot products of combustion from mine fires resulting in decrease in its density and chemical heat from extensive oxidation of pyrite.

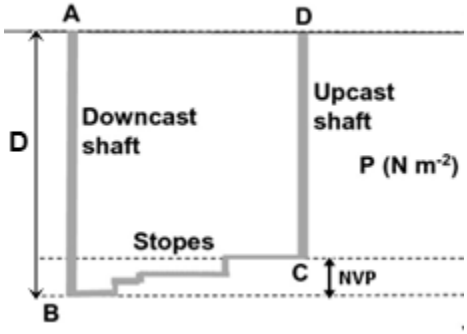
- **Decrease in the density of mine air** on account of taking up of lighter gases like hydrogen, and moisture and methane as it passes through the workings.
- **Impact and suction effect** of the air currents.
- **Diffusion**
- **Effect of continuous falling drops of water** in shafts, which favourable work in downcast shafts and unfavourably in upcast shafts.
- Difference in elevation of the surface and underground mine workings (chimney effect).
- Differences in air temperature between mine openings.
- Effect of circulation of conditioned air through downcast shaft.
- Intensity of mine ventilation
- Heat from human metabolism

NATURAL VENTILATING PRESSURE (NVP)

natural ventilating pressure is not a single pressure, but a *difference* between two pressures. If an air current is caused to flow along an airway, a pressure difference must be set up between its two ends. In mines, fans are used to set up a difference of pressure between the air in the fan drift and the air outside. natural ventilating pressure is always a difference between two absolute pressures.

The natural ventilation pressure between any two points causing airflow is calculated from the *difference in specific weights of air between the two points caused by varying pressure, temperature and moisture*.

The pressure difference between the bottoms of air columns is the natural ventilation pressure (NVP).



Methods to calculate NVP

Various formulae to compute NVP are given below:

NVP from densities

Formula#1

$$NVP = D(\gamma_{mD} - \gamma_{mU})\, kgf/m^2 = D\frac{(\gamma_{mD} - \gamma_{mU})}{13.6} mm\, Hg$$

Here γ is specific weight of air. D is depth of downcast shaft.

Also note that

1 mm of Hg. = 133.3 Pascal = 133.3 N/m^2

1 mm of water gauge = 9.8 Pascal

1 kg/m^2 = 9.81 Pascal

1 N/m^2 = 1 Pascal

Formula #2

NVP = motive column x density of air in DC shaft

Formula #3

$$NVP = D(\rho_d - \rho_u)g\ Pascal$$

where ρ_d = mean density of air in DC shaft (kg/m^3)

and ρ_u = mean density of air in UC shaft (kg/m^3)

Formula #4

$$NVP = \frac{gDB}{287.1}.\frac{T_u - T_d}{T_u T_d} x10^3\ Pascal$$

where D = depth of downcast shaft (m)

B = barometric pressure in kPa

T_u = mean temperature in UC shaft (K)

T_d = mean temperature in DC shaft (K)

NVP from Pressure and Quantity

It is possible to calculate NVP from pressure and quantity measurements with fan running and fan stopped.

$$NVP = P_f.\frac{Q_n^{\ 2}}{Q_f^{\ 2} - Q_n^{\ 2}}$$

Where P_f = Fan pressure

Q_t = Quantity with fan running

Q_n = Air quantity when fan stopped

NVP from Pressure Across a Stopping in Fan-Drift

NVP can also be found out by installing a door in the fan drift or in any part of the main circuit of ventilation and measuring the pressure difference which indicates NVP. In this method it is necessary to make the measurement as soon as the door is closed. Otherwise addition of heat from surroundings changes the conditions.

NVP from Fan Running at Two Different Speeds

Another method is measuring the fan pressure and quantity at two different speeds of the fan.

Since the resistance of the mine at fan speed n_1 and n_2 does not change.

$$NVP = \frac{P_{f2}Q_1^2 - P_{f1}Q_2^2}{Q_2^2 - Q_1^2}$$

Example

The volume of air entering a mine is 1080 m^3/min, the surface fan producing a pressure of 51 mm w.g. When the fan is stopped, quantity of air entering the mine falls to 540 m^3/ min. Calculate the amount of natural ventilation pressure assisting the fan.

Solution

Given that

Air quantity in m^3/s are

1080/60 = 18 m^3/s

and 540/60 = 9 m^3/s

Now

$$R = \frac{P}{Q^2} = \frac{51+N}{18^2}$$

and $$R = \frac{P}{Q^2} = \frac{0+N}{9^2}$$

Equating these two

N = 17 mm w.g.

MOTIVE COLUMN

Motive column is that portion of downcast column that has the same weight as the difference of weight between the two columns. It is the effective part of the down cast column that causes a flow of air to take place.

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 m^2 in cross section which gives rise to NVP is called *motive column*. In other words, the motive column, when referring f to N.V.P., is the preponderant or unbalanced part of the whole D. C. column, 1 m^3 in cross section, i.e. that part of the D.C. column which is not balanced by the UC air column.

It is possible to express natural ventilation as meters of air column or motive column, which is the height of an imaginary column of air that produces a pressure equal to the difference of pressure between the bottoms of downcast and upcast shafts.

Various formulae to compute motive column height are given below:

Formula #1

$$\text{Motive column (h)} = \frac{NVP}{g.\rho_d} \text{ (m)}$$

where NVP is in Pascal

ρ_d is the density of air in the downcast shaft (kg/m^3)

g is acceleration due to gravity (m/s^2)

Formula #2

$$h = \frac{\rho_d - \rho_u}{\rho_d}.D \text{ metre}$$

where ρ_d = air density in DC shaft (kg/m^3)

ρ_u = air density in UC shaft (kg/m^3)

D = depth of shaft (m)

Formula #3

$$h = \frac{T_u - T_d}{T_u}.D \text{ metre}$$

where T_u = average dry bulb temperature in K in UC shaft

T_d = average dry bulb temperature in K in DC shaft

D = depth of shaft in metres

DENSITY OF AIR

For moist air

Density (w or ρ) of air varies with its temperature, pressure and moisture content.

$$\rho = \left(\frac{B - 0.378e}{287.1T_d} \right).10^3 \; kg/m^3$$

where B = barometric pressure in kPa

T_d = temperature of air in DC shaft (K)

e = water vapour pressure in air in kPa

For dry air

For density of dry air, e = 0

$$\rho = \left(\frac{B}{287.1T_d} \right).10^3 \; kg/m^3$$

HORSE POWER DUE TO NVP

$$HP_{nvp} = \frac{PQ}{746}$$

where P is pressure in Pascal, Q is air quantity in m^3/sec.

The number 746 is taken in denominator because 1 h.p. = 746 Watt.

Example

Mean air temp in a DC shaft 400 m deep is 28^0C and in the UC shaft is 38^0C. Calculate (i) the motive column (ii) NVP assuming average barometric pressure in DC shaft to be 750 mm of Hg.

Solution

Formulae

The height of motive column is given by

$$h = \frac{T_u - T_d}{273 + T_u} xD$$

where h = height of motive column

T_u = average temperature in upcast shaft (^{0}C)

T_d = average temp in downcast shaft (^{0}C)

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

Now, NVP = motive column x density of air in DC shaft

Motive column

$$h = \frac{38-28}{273+38} x400 = 12.8\,m$$

Density of air in the DC shaft = $\frac{0.4645B}{273+T_d}$

where B is the barometric pressure in mm of Hg.

Density

$$w = \frac{0.4645x750}{273+28}$$

NVP

It will be motive column x density of air in DC shaft

= 12.8 x 1.157 = 14.81 kgf/m^2 = 148.1 Pa

Example

Two vertical shafts each 6 m in diameter and 300 m deep are connected at the bottom by a level 2 x 2.5 m in cross-section and 800 m long. The average barometric pressure in the shafts being 101.325 kPa. Calculate the velocity of flow in the level due to natural ventilation. Temperature measurements in the shafts are as follows:

Downcast shaft-top = 293 K

Downcast shaft-bottom = 296 K

Upcast shaft-top = 303 K

Upcast shaft-bottom = 303.5 K

Coefficient of friction k for shafts is 0.004 Ns^2m^{-4} and for level k is 0.01 Ns^2m^{-4}.

Solution

Average temperature of downcast shaft = (293 + 296)/2 = 294.5 K

Average temperature of uppercast shaft = (303 + 303.5)/2 = 303.25 K

$$NVP = \frac{gDB}{287.1}.\frac{T_u - T_d}{T_u T_d} x10^3\ Pascal$$

$$= \frac{9.8x300x101.325}{287.1}\left(\frac{303.25-294.5}{303.25x294.5}\right)x10^3\ Pa$$

= 101.7 Pa

Cross-sectional area of the level = 2 x 2.5 = 5 m^2

Perimeter of the level = 2(2 + 2.5) = 9 m

Cross sectional area of shafts = $\frac{\pi}{4}(6)^2 = 9\pi\ m^2$

Perimeter of the shafts = πd = 6π m

Let v = velocity of flow in the level in m/s.

Q = Q_{level} = Q_{shaft} = AV = 5V m^3/s (shafts and level are in series, hence Q is same)

Here A is level area and V is velocity in level.

NVP = NVP in shaft + NVP in level

$$\therefore \quad Total\ NVP = \Delta P = \frac{k_{shaft}S_{shaft}Q^2}{A_{shaft}^{\ 3}} + \frac{k_{level}S_{level}Q^2}{A_{level}^{\ 3}}$$

$$\therefore \quad 101.7 = 2\left(\frac{0.004(6\pi x300)(5V)^2}{(9\pi)^3}\right) + \left(\frac{0.01(9x800)(5V)^2}{5^3}\right)$$

$$\therefore \quad 101.7 = 14.45V^2$$

$$\therefore \quad V = 2.65\ m/s$$

MECHANICAL VENTILATION

Mechanical ventilation therefore refers almost always to the production of air currents in mines by pressure differences produced at particular points in the air circuits by centrifugal fans, acting in conjunction with natural-draft pressures generated throughout the air circuits.

Most of the mines are ventilated mechanically.

BASIC TYPES OF FANS USED FOR VENTILATING UNDERGROUND MINES

Fans are used in mines to cause flow of air.

They are basically of two types:

- Centrifugal Fan
- Axial Flow Fan

It is important to note that, **centrifugal fans** deliver **low quantity but at higher head** whereas axial flow fans deliver higher quantity at lower head.

FAN LAWS

Let Q = quantity of air in m^3/sec

N = r.p.m. of fan

V = peripheral speed of blade tips in m/sec

Law 1

Q varies directly as fan speed,

i.e. $Q \propto N$ or V

Law 2

Water gauge developed varies directly as square of the fan speed or of the quantity i.e. Water gauge a N^2 or Q^2 or V^2.

Law 3

Horsepower required to drive the fan varies as the cube of the fan speed or of the quantity

i.e. H. P. a N^3 or V^3 or Q^3.

Example

A mine is ventilated by a fan producing 6000 m^3 per min at 75 mm w.g. The fan runs at 300 rpm and absorbs 160 BHP. To increase the volume of air flowing through the mine, the fan is speeded up to 400 rpm. Calculate (i) volume of air at new fan speed (ii) new w.g., BHP and efficiency of the fan.

Solution

New volume

From fan laws

$Q \propto N$

$\therefore$ New volume = 6000 x (400/300) = 8000 m^3/min

New W.G.

w.g. $\propto N^2$

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

New BHP

$$BHP \propto N^3$$

$$\therefore \text{ New BHP} = 160\,x\left(\frac{400}{300}\right)^3 = 379$$

Efficiency of fan

$$\text{Air HP} = \frac{PQ}{75} = \frac{133.3(8000/60)}{75} = 237$$

$$\therefore \text{ Efficiency of the fan} = \frac{237}{379}\,x100 = 62.5\%$$

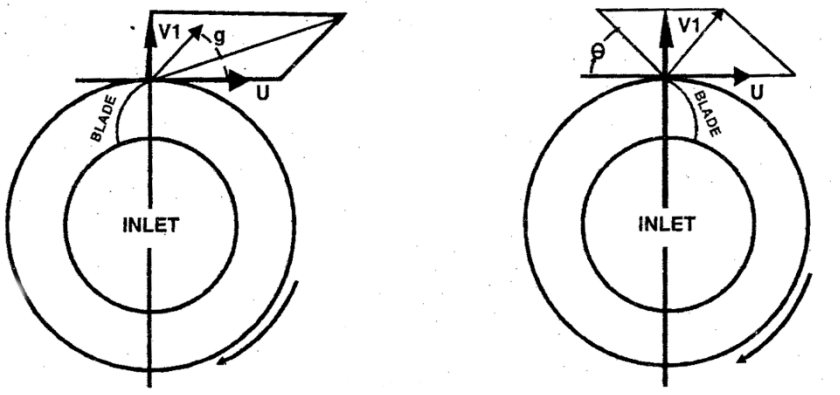
DEFINITIONS OF TERMS USED IN CONNECTION WITH FAN WATER GAUGE

Theoretical Depression (head)

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

$$H = \frac{U(U \pm V_1 \cot\theta)}{g}$$



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

Effective Depression

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 fan drift 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.

Manometric Efficiency

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.

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,

$$P = \frac{PQ}{75} = \frac{RQ^3}{75}$$

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.

Mechanical Efficiency

Overall mechanical efficiency or useful effect = H.P. of ventilation/H.P. input to engine or motor

Fan efficiency = H.P. of ventilation or Air H.P./Fan shaft H.P.

Example

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

Fan dia = 3.6 m

RPM = 300

Width at periphery = 1.5 m

Blade angle = 40^0

Velocity of flow = 4.5 m/sec

Air density = 1.2 kg/m^3

Solution

Blade tip speed

$$= \frac{\pi DN}{60} = \frac{\pi(3.6)(300)}{60} = 56.57\, m/\sec$$

Theoretical depression

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

$$= \frac{56.57(56.57 - 4.5\cot 40^0)}{9.8}$$

= 295.5 m of air column

= 295.5 x 1.2 kgf/m^3 = 354.6 kgf/m^2 or mm w.g.

= 3546 Pa

Peripheral area of fan outlet

It will be

= circumference x width

= πDW

= π x 3.6 x 1.5

= 16.97 m^2

Quantity delivered by a backward bladed centrifugal fan

Quantity = area of periphery x velocity of flow

= 16.97 x 4.5

= 76.37 m^3/sec

= 4582 m^3/min

NOISE CONTROL IN MINES

A wide variety of mining processing and materials-handling equipment is used in underground and surface mining, and in preparation/pro-cessing plants with many noise sources on them having a range of varying noise levels. Frequently, a single piece of equipment will combine several individual noise sources.

The fundamental approach to the problem of noise control and equipment consists of:

- the selection of new low-noise equipment expected to be used for the task from the manufacturer; failing which, use of additional retrofit for the existing older noisy equipment from the original equipment manufacturer and suppliers of noise controls or utilizing the experience of other companies in using similar equipment.
- the use of some or all of the engineering or administrative noise controls, or a combination of both that can be implemented for achieving meaningful reduction.
- the implementation of an effective planned preventive maintenance program for equipment developed by the company, which will reduce noise levels by more than 50%.
- the implementation of preventive maintenance program, especially for existing noise control devices that are added or built into machinery as given in the Environmental Management Plan of the project proposal.
- the use of an alternative technology where available, which may not always be feasible in practice.
- shutting down noisy equipment when not needed.
- the use of personal hearing protection devices when exposed to noise levels above 85 dBA.

Underground Coal Mining Equipment

Most of the noise sources in underground environment in coal and metal mines are individual pieces of equipment operating in confined environments.

Continuous mining machines (CMM) are by far the most commonly used equipment in the underground coal mining industry. They are nearly always selected for room-and-pillar mining

of coal. CMMs generate *excessive* sound levels at the face which can be harmful to face workers. The continuous noise generated results from the power pack and transmission gear, impact noise from the cutting head of the miner, on-board chain conveyor noise, and fan noise from the dust collection system. Of these, cutting, conveying, and dust collection systems generate most of the noise, with the chain conveying system being the dominant noise source.

The engineering noise controls must be addressed to noise at the source and for effective reduction of operator noise exposure, noise from the **conveyor** system must be targeted first. The conveyor noise is caused by the conveyor chain assembly moving the mined coal along the metal deck base to the discharge end, with flight bars impacting the pan sides due to slack chain.

A **barrier** can be used to block or redirect the sound away from the machine operator when it is operating, thereby reducing the operator exposure.

With **continuous haulage system** using several self-powered mobile bridge conveyors linked to each other, proper alignment will reduce the impact noise generated at the transfer points.

Continuous miners used on longwall faces in longwall panels, also called shearers or shearer loaders, travel along the face mounted on an armoured flexible round steel link chain conveyer 150 to 200 m or more in length, cutting a web off the long face by the rotating cutting drum(s) as it is hauled by chain or chainless haulage. The conveying chain assembly may be single, double, or triple-strand type, with flight bars scraping coal over the line pans to deliver it onto a stage loader, which is a heavy-duty scraper chain conveyor serving as a link between the face conveyor and panel gate belt conveyor.

Belt conveyors are extensively used in underground coal mines, in panel gate roads, and on trunk roadways to transport coal to the pit bottom, and even to the surface when a mine has been opened up as an incline or drift mine. A planned preventive maintenance program will uncover all the noise sources on the conveyor installation. Well-maintained belt conveyors with special attention paid to conveyor idlers are not normally a problem, but noise is generated at the drive units and very often at the transfer and loading points, which should be properly designed.

Noise control measures

- At belt-to-belt transfer points using a troughed lined steel feed chute of spiral form, rubber-covered bang or impact board in the trajectory of the material discharge in the enclosed rubber-lined chute.
- When loading, reducing the impact noise by keeping some material in bins as deads, reducing the drop height, and by longer falls using spiral chutes. Enclosing the conveyor drive unit with a vinyl curtain around the perimeter of the equipment.

- With belt conveyor hoisting in drift mines, using multi-motor drive systems located at the discharge end on the mine surface, the drive may be housed in an acoustically treated control room or enclosure.

Example

The sound pressure level is measured as 5 x 10^{-4} N/m^2. What is the noise level in dB?

Solution

$L_p = 20\log_{10}(P/P_0)$

$= 20\log_{10}(5 \times 10^{-4}/2 \times 10^{-5})$

$= 27.96$

Note: Here reference sound pressure is 2 x 10^{-5} N/m^2 for human hearing.

Example

The noise levels measured during the operation of three machines arc 80 dBA, 75 dBA and 85 dBA respectively. Determine the equivalent noise level for the above three machines.

Solution

$L_{total} = 10 \times \log_{10}(10^{L_1/10} + 10^{L_2/10} + 10^{L_3/10})$

$10^{(80/10)} = 100{,}000{,}000$

$10^{(75/10)} = 31{,}622{,}776$

$10^{(85/10)} = 316{,}227{,}766$

Sum the intensities

$= 100{,}000{,}000 + 31{,}622{,}776 + 316{,}227{,}766.0 = 447{,}850{,}542$

$L_{total} = 10\log_{10}(447{,}850{,}542) = 86.5$ dBA

Example

It is required to find out the day-night equivalent noise levels at a location. The three-hourly day average values in dB are 48, 54, 56, 52, 61 and three-hourly night average values in dB are 36, 42, and 48. What is value of L_{dn}?

Solution

The day-night equivalent noise level (L_{dn}) is a 24-hour average sound level where a 10 dB penalty is added to nighttime noise levels to account for increased human sensitivity to noise at night.

The three-hourly night average values are 36,42, and 48 dB. We add 10 dB to each to account for the nighttime penalty be 46 dB, 52 dB and 58 dB.

$L_{dn} = 10 \times \log_{10} (10^{L_1/10} + 10^{L_2/10} + 10^{L_3/10} + ...)/n$

$= 10 \text{ x } \log_{10}(10^{4.8} + 10^{5.4} + 10^{5.2} + 10^{6.1} + 10^{4.6} + 10^{5.2} + 10^{5.8})/8$

$= 55.68$

PROENGR.IN

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