Indian Institute of Technology (Indian School of Mines) Dhanbad
Department of Mining Engineering
Mid Semester Monsoon Examination 2023-2024 for B.Tech. + Int. M.Tech. Students (101 nos.)
Subject: Mine Environmental Engineering (Code: MND 406) **Venue: NLHC LH 1-5**

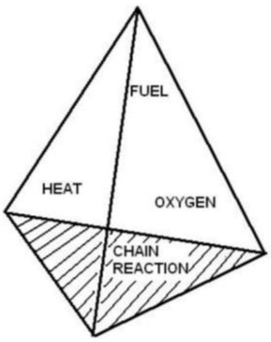
Time: 120 minutes (08:00 AM – 10:00 AM) **Date: 20-09-2023** **Maximum Marks: 56**

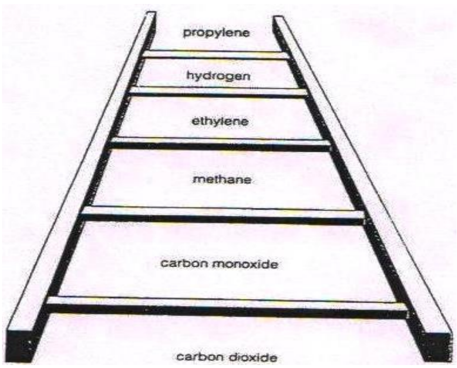
Answer questions of all the three sections and marks are assigned to each question

Section A: Short Answer Type Question (answer in 1-2 line) – 15 marks

Answer all the questions and each question carries 0.5 mark

1. Name the inert gases found in coal mines.
   Carbon dioxide (CO_2) and nitrogen (N_2)
2. Whether carbon monoxide is a flammable gas or not?
   flammable gas
3. What is the density of air at atmospheric temperature and pressure?
   1.225 kg/m^3
4. Which one of these is more susceptible to coaldust explosion: Type A coaldust requires 10 kg of stonedust for inertization to form a non-explosive mixture whereas type B coaldust requires only 2 kg for inertization.
   Type A
5. What is the optimum percentage/concentration of methane and coaldust in air to form a stoichiometric mixture causing maximum damage?
   Methane – 9.5% and coaldust- 30-300 g/m^3 also depend upon particle size and ignition temperature
6. Which one of the following is the best ground for constructing dam: clayey shale, sandstone and shaly sandstone.
   shaly sandstone.
7. What is the ratio of cement to aggregate in case of construction of a concrete dam?
   1:3 or 2:3 (20:65)
8. What is indicated by the symbol () in a mine plan as per the Coal Mines Regulation (2017)?
   Water dam
9. What is the minimum width of safety pillar to be kept against inundation between adjacent mines and water bearing strata as per the Coal Mines Regulation (2017)?
   20 m
10. Name an extinguisher suitable for fighting an underground electrical fire.
    Carbon dioxide snow extinguisher/dry chemical powder extinguisher/multipurpose dry chemical (dry powder) extinguishers
11. What is fire triangle, fire tetrahedron and fire ladder?



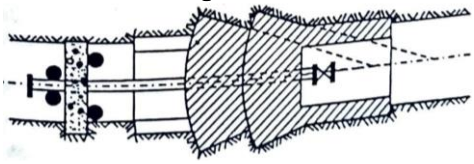


12. Prove that the molecular weight of atmospheric air is 29 g.
    (0.21 x 32 + 0.78 x 28 + 0.005 x 40 + 0.005 x 44) = 29 g/mol

13. Assuming no diffusion, reaction and bonding of the gases, arrange the following concentration of the gases from bottom upwards in a mine gallery: H_2S, CO, CH_4, CO_2 and O_2.
H_2S= 34, CO= 28, CH_4=16, CO_2 = 44 and O_2= 32 (CH_4, CO, O_2, H_2S, CO_2)
14. Why methane exists together with coal?
Coalification involves organic matter and its decaying produce methane
15. What is spontaneous combustion?
It means self-heating of coal or of any easily oxidizable substance due to auto-oxidation at ordinary atmospheric temperatures.
16. How spontaneous heating is detected in mine?
Chemical (Graham's ratio) and physical detection (smoke, odour)
17. Differentiate between incipient and open fire.
Incipient fires are also called coal seam fires which originate in abandoned and discontinued mine workings, in goafs/gobs of coal seams being worked, in falls of roof, in fractured coal pillars and in the vicinity of geological disturbances.
Open fires which originate in open mine working such as shafts, roadways and coal faces due to a variety of causes including spontaneous combustion and are characterized by visible active combustion or flame.
18. Differentiate between aquifer and aquiclude with suitable example of rock formation.

Aquifer	Aquiclude	Aquitard	Aquifuge
These are permeable	These are impermeable	These are partly permeable	These are impermeable
There is a yield of water	These do not yield water	There is a yield of water but the yielding will be so slow	These do not yield water
This can store water	This can store water	This can store water	This cannot store water
Sand and gravel are some of the examples of aquifer	Clay is an example of an aquiclude	Sandy clay is an example of aquitard	Compact rocks like basalt and granite are some of the examples of Aquifuge

19. Write the formula to estimate thickness or length of each arch or cylindrical dam with suitable neat diagram in case where 'n' number of such dams are required to be erected.



$t = \frac{r_i}{\frac{n\sigma_s}{P} - 1}$ **m; where p is the maximum water pressure (N/m2); r_i is the internal radius of the arch ring (m); σ_s is the safe compressive stress of the material used (N/m^2).**
20. When it is required to seal off the entire mine?
When it is almost impossible to control the fire by direct and indirect attack or when fire cannot be tracked
21. What are the causes of explosion and fire in a metal mine?
Presence of sulphur
22. State with reason whether a foam extinguisher (chemical or mechanical) can be used for fighting an electrical fire or not?
No, as foam aqueous solution can act like conductor
23. How triggered barrier is different from other barriers (water and stonedust) during prevention against coaldust explosion propagation?
It uses electronic sensor to act instead of coaldust explosion pressure in case of water and stonedust barrier
24. State with reason whether a mixture of methane-air can be ignited below and above its flammable limit or not?

Yes, A mixture containing less than 5 per cent methane, although not flammable under normal conditions, may explode when at a high temperature or when compressed adiabatically due to blasting or a coal-dust explosion. Also, a mixture containing 15 to 100 per cent methane in which electric sparks can be generated without any explosion taking place is dangerous in a mine as it may pass through the flammable range on dilution with air in mine workings.

25. Define antipyrogenes with examples.
    antipyrogene (a fire-proofing composition) forms a protective surface coating and infusing the inhibitor chemical to fill up the microcracks in the solid coal. $Ca(OH)_2$, $Ca(HCO_3)_2$, NH_4CL, Na_2SiO_3, $CaCl_2$, Montan Powder, NaH_2PO_4, $NH_4B_4O_7$, H_3BO_3
26. How to deal with the situation where the safe compressive stress of the material used in construction is exceeded by the water pressure?
    Construct more number of flat/arch/spherical dams
27. Distinguish between firefighting by direct and indirect attack.
    Direct attack- action taken in close contact with fire
    Indirect attack- isolation or sealing
28. Illustrate the mechanism of coaldust explosion.
    surface oxidation of coal-dust particles, partial de-volatilization of solid coal particles, ignition of the volatiles-air mixture in the space between particles, and combustion of solid particles are conjointly responsible for the development of a coal-dust explosion.
29. What is primary flame and secondary flame in a firedamp explosion?
    When the concentration of methane is greater than 9.5, two types of explosion flame occur, the primary and secondary flames. The primary flame propagates at a greater velocity and consumes the entire available oxygen. The secondary flame is produced later by the burning of the unburnt gas with the help of oxygen supplied by the backlash. It propagates slowly in the direction opposite to that of the primary flame.
30. Mention two unique characteristics as differences between firedamp and coaldust explosion. (any of the two mentioned below)
    Both have lower and upper limits of flammability. Explosions of limit mixtures are weak. The ignition temperature of firedamp is 650° to 750°C while that of dry airborne dust is 600° to 900°C. The heats of combustion and explosion temperatures are nearly the same. The flammability of firedamp is generally the same throughout the mine. On the other hand, the ignition and flammability of dust in mine workings vary greatly, depending on fineness, volatile matter, ash, moisture, etc. The propagation of firedamp explosions takes place due to conduction of heat. With coal-dust explosions, on the other hand, radiation of intense heat by the pressure wave as well as the explosion flame plays an important part in their propagation. The maximum pressures developed in some dust explosions are higher than in firedamp explosions. The rates of pressure rise are, however, generally lower than those obtained in firedamp explosions. Coal-dust explosions are frequently more disastrous in their effects than firedamp explosions because of their longer duration. With firedamp explosions, carbon monoxide is frequently found. With coaldust explosions, on the other hand, carbon monoxide is always found. The ignition of a coal-dust explosion even with the strongest igniting source requires at least 100 ms, while with a firedamp explosion only 1 to 2 ms is required. The velocity of propagation of coal-dust explosions is generally higher than that of firedamp explosions.

Section B: Long Answer Type Question – 23 marks

31. **(i)** What are the different causes of firedamp explosions? **(ii)** Explain the different measure for prevention of firedamp explosions. **(iii)** Draw the Coward's diagram showing explosive nature for different composition of methane-air mixture.

    The various causes of firedamp explosions in mines may be grouped under the following headings:

(a) Negligence of miners

(b) Use of damaged safety lamps and their improper handling

(c) Blasting

(d) Mine Fires

(e) Friction

(f) Electric sparks

(g) Other special causes

Measures against accumulation of dangerous firedamp mixtures in mine workings from the beginning

The most effective method of preventing firedamp explosions in mines is by providing adequate ventilation which will dilute the firedamp, besides other harmful gases, to well below limits that may be prescribed for different mine workings and carry it away to the surface.

Frequent sampling of mine air for methane at several points in the mine is, therefore, an important measure for prevention of firedamp explosions.

The following important points should be borne in mind:

(i) The mine should be mechanically ventilated by the exhaust ventilation method. A reserve or standby main fan having an independent drive and power circuit when electric motors are used should be provided especially in mines with a methane emission greater than 5 m^3 per tonne of daily output. Every main fan shall be so installed and positioned that it is not damaged by a possible explosion as far as possible. It should be fitted with an automatic alarm or signal device to alert persons on duty, should it slow down or stop.

(ii) The mine equivalent orifice should be as large as possible ($> 2\ m^2$)

(iii) The ventilation of mine workings greater than 3 m in length should not be done by diffusion alone.

(iv) The ventilation of development headings should be done by utilizing the mine ventilating pressure as far as practicable. Air which has passed through any abandoned area which is inaccessible or unsafe for inspection should not be used to ventilate any active workings in a mine. If auxiliary fans are required to be properly used, they should be installed and located so that air is delivered within 5 m of the face and recirculation of ventilating air is eliminated to keep the working face clear of flammable and noxious gases, dust, and explosive fumes. In places where auxiliary fans are used, the ventilation during weekends and idle shifts should be by means of the primary air current conducted into the place in a manner to prevent accumulation of firedamp.

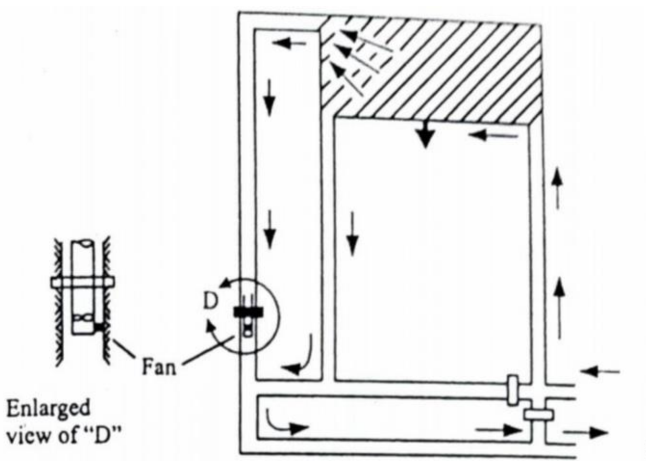
(v) Ventilation doors should be correctly located and kept closed except when men, equipment and trains are passing through them. They should always be in good repair and be self-closing.

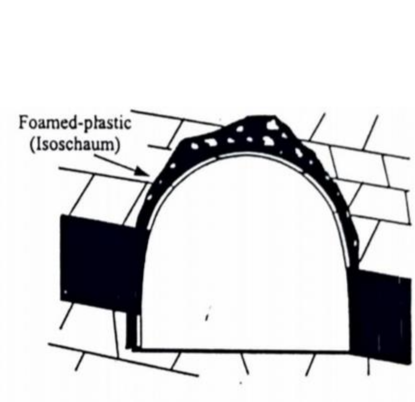
(vi) The mine ventilation system should be planned so that simple, effective, and reliable ventilation of all workings is assured. Where multiple main fans are used, the ventilation system should be so arranged that no adverse air reversal will occur in the event of failure or stoppage of any fan or fans. Predictions of firedamp emission for different sites in a coalfield will greatly assist in planning safe ventilation systems for mines.

(vii) Seams should be extracted, as a rule, from top downwards to decrease the methane levels in the lower seams.

(viii) The method of extraction should be selected so that it guarantees an easy and safe ventilation of the faces by air dilution with adequate velocities at the face and at the waste edge. Face air velocities of up to 4 m/s or more are being used today on modern longwall faces in very gassy seams to mine coal safely. In high-capacity longwall faces with strong methane emissions and high ventilating air requirements, faces laid with W-ventilation system will enable larger air quantities to flow through them besides providing a middle gate for drilling additional methane drainage holes.

(ix) A particularly high standard of unit ventilation by separate ventilation splits should be maintained in each mechanized mining section and in districts/panels liable to gas outbursts.
(x) In bord and pillar and longwall retreating panels in very gassy seams worked with caving, the firedamp content in the goaves behind the active faces must be controlled by in-mine local or central drainage of goaves or drainage through surface ventilation boreholes. Bleeder systems with bleeder entries and bleeder entry connections as in American mining practice can be successfully employed to move firedamp-air mixtures continuously from the caved areas behind retreating longwall faces away to the return airways, and to minimize the hazard of expansion of goaf gases due to changes in atmospheric pressure (Fig. 2.5) and their release into faces and gate roads as unusual emissions during the rapid closure of the goaf, just behind the face supports or not far away from them, when caving takes place periodically.
(xi) Mining with backfilling (Fig. 2.6) or solid stowing of the waste/goaf, especially hydraulic filling, prevents formation of methane reservoirs in goaves.
(xii) Horizontal methane drainage holes drilled into a seam in advance of a panel must be sealed before they are intercepted during extraction of the panel to prevent methane from being discharged forming flammable methane mixtures.
(xiii) Air currents and methane emission should be checked by systematic measurement of air quantities and their methane concentration. Special examinations for firedamp layering should be made during periods of falling barometric pressure in ascensionally-ventilated roadways adjacent to old workings, within the areas of moving ground behind faces, and high points at which gas will tend to accumulate and layer. If, at any time, the air at any working place contains a methane accumulation, changes or adjustments in ventilation should be made at once so that such air contains less than the prescribed value. While making such changes or adjustments, power to electric equipment located in such places should be cut off. All personnel may have to be withdrawn from the area, depending on the concentration of methane. For judging the possibility of formation of methane layering, the Middendorf formula [33] may be used to calculate the 'layering index": $S_{index} = \sqrt[3]{\frac{24v^2}{c\sqrt{F}}}$.
where v is the mean velocity (m/s); c is the mean methane content of air current (% CH_4); F is the area of the cross-section of the airway at the measuring station (m²). If S_{index} <2, there is probable danger of methane layering. If S_{index} >2, there is no danger of methane layering.
A correction factor must be used for the rising inclination and change in the cross-sectional area of the airway.





Roof voids/cavities filled with foamed plastic.

Measures against ignition of flammable firedamp mixtures
The various preventive measures to be taken against ignition of flammable firedamp mixtures in mines are:

(i) All persons should be prohibited from carrying smoking articles (pipes, cigars, cigarettes, tobacco other than chewing tobacco or snuff), matches or other spark- or flame-making devices into the workings. All men entering the mine should be searched for contraband.
(ii) All coal mines should be treated as safety-lamp mines as a number of explosions in the past had occurred in the so-called naked-light mines. In short-life naked-lamp mines where naked lights are to be retained, special attentions should be paid to ventilation, gas-testing, and to precautions against coal dust.
(iii) (a) Only certified flame-proof and intrinsically-safe apparatus should be used in coal mines. The apparatus should be properly installed, protected, operated, and maintained to assure safe operating conditions.
(b) If, in a district or part of a mine, electrically-operated equipment is not required for immediate use and men are not working there, power should be cut off in that district or part of the mine. Tests for methane should be made immediately before the equipment is energized.
(c) Trailing cables which are vulnerable to damage should be suspended from hangers, specially provided for the purpose, and, if they are present in the face area, should be suitably protected against damage from any cause.
(d) To prevent ignition from electrostatic charges, all ventilation ductings should be earthed and only antistatic polythene sheeting, hoses, and belts used.
(e) A reliable methane monitor or cut-out that will automatically cut off power supply to the electrical equipment when the methane concentration reaches the prescribed maximum percentage may be installed in endangered mine workings.
(f) A methane monitor should be installed, when available, on any electric face-cutting equipment, continuous miner, longwall face equipment, and loading equipment to automatically de-energize equipment or give a warning automatically when the concentration of methane reaches the maximum prescribed limit. The methane monitors shall be properly maintained to keep them operative and checked at regular intervals for operating accuracy.
(g) When a main fan is stopped for any reason, electrical power should be immediately cut off in return airways. After the fan has been restarted, the power shall not be switched on unless normal ventilation and safe conditions have been restored.
(h) In places where auxiliary fans are used, the auxiliary fans should not be operated during stoppage of normal mine ventilation. Accumulations of methane should be removed after restoration of normal mine ventilation before the fans are operated. In a place ventilated with an auxiliary fan, if the auxiliary fan is stopped or fails, electrical equipment operating in the place should be stopped and the power disconnected at the power source until the ventilation is restored. The auxiliary fan should be maintained so that the impeller does not strike the casing.
(i) Changes in ventilation affecting the main air current or any split thereof should be made only when the mine is idle. The power supply should be cut off from the affected area before changes are made.
(iv) Production of excessive frictional heat with conveyors, brakes, and bearings should be avoided by good installation and proper maintenance. The production of frictional sparks, especially by metal-to-rock contact as with face-cutting, drilling, continuous mining, and longwall powerloading equipment, should be avoided as far as possible. Pyritic intrusions in coal seams have a significant frictional ignition potential due to sulphur flames produced by cutting bits striking pyrite. Besides producing frictional sparks, cutting bits of mining machines, when they are worn or broken, are especially hazardous if they scrape against instead of cutting into the material. The precautionary measures against frictional ignitions during cutting of seams by jib coalcutters consist in selecting an appropriate cutting horizon and lower pick speeds, wet-cutting using external water sprays directed at the ingoing and outgoing pick or using ‘whale jibs’ in which the water js piped to a number of points around the apex of the jib, providing a water mist in the cut, introducing an inert gas such as carbon dioxide or nitrogen into the cut, or ventilating the cut. Adequate ventilation of the cut using

a water spray device provides the most effective known means of preventing frictional ignitions.

(iv) With power-loading on longwall faces using shearer loaders, the frictional ignition hazard varies greatly, depending on the type of shearer loader used (conventional fixed drum shearer loader, ranging drum shearer loader, floor or conveyor-mounted trepanner), whether cutting with or against the ventilation, presence of hard dirt bands or quartzitic pyrite nodules in the seam or floor, standard of ventilation, cutting speed, type of pick, and sharpness of cutting picks. Research has shown that point-attack picks cause ignitions much more easily than radial or forward-attack picks and the risk of ignition diminishes significantly below pick linear cutting speed of 1.5 m/s. Experience in British mines working thin seams has shown that with good standards of ventilation, the frictional ignitions can be confined to the neighbourhood of the power leader, that the incidence of ignition in thin seams resulting from picks striking the floor is much greater than from picks striking the roof due to the cutting zone at the floor being most difficult to ventilate, and that the risk of ignition is least with L-shaped outboard arm ranging drum shearer loaders and maximum with conveyor-mounted trepanners [15].

(iv) The risk of frictional ignitions by ripper-type continuous miners can be prevented or minimized by modified bit design and using carefully-engineered back sprays on the ripper drum with adequate spray pressure at the nozzle applied directly at the coalcutting bits. Owing to the dominant role played by friction as an ignition source, one may divide the mine into abnormally high-suspect zones or areas of frictional ignitions, in which an intensive programme to eliminate frictional ignitions may be undertaken.

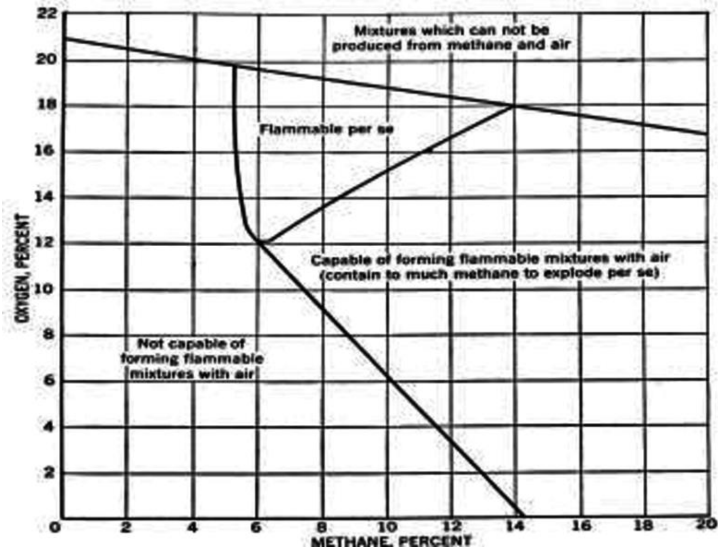
(v) Use of materials constructed of light alloys should be prohibited to eliminate incendive sparking.

(vi) Spontaneous heating of coal should be controlled by proper planning of mine development as well as coal extraction, good ventilation system, and inspection. Mine fires should be detected in their early stages.

(vii) Blasting with explosives should be restricted to a minimum. Immediately before firing a shot or a group of multiple shots and after blasting is completed, examination for methane should be made. Mechanical winning and heading machines and blasting devices such as Hydrox, Cardox, and Airbreaker shells may be used as alternatives to ordinary explosives for breaking down coal. Pulsed-infusion shotfiring in combination with deep-hole water infusion may also be employed for getting coal without machine-cutting. Methods of charging a shothole with water stemming are shown in Fig. 2.7.

(viii) Welding, flame-cutting, grinding, and soldering operations must be done only after taking extreme precautions. Examinations for methane should be made immediately before and periodically, during such work.

(ix) All mobile diesel-powered equipment must be inspected and maintained in the approved condition in accordance with the instructions furnished by the manufacturer. Particular attention should be paid to the flame arresters in the intake and exhaust systems, the exhaust cooling system, and the surface temperature, of the engine and components of the exhaust system.



or

(i) What are the different causes of coaldust explosions? **(ii)** Explain the different measures for prevention and suppression of coaldust explosions. **(iii)** What is most dangerous particle size which can cause coaldust explosion and why there is loss of flammability of coaldust below 10 µm m size?

The various causes of direct ignition of a dust cloud can be classified as

(a) naked flames
(b) friction
(c) electric sparks
(d) firedamp explosions

Naked Flames: A naked flame is the easiest means of igniting a dust cloud as the source of heat is of considerable size and a larger part of the dust cloud can be heated.

Friction: Hot surfaces as a result of mechanical friction, such as overheated bearings, may ignite surrounding explosive dusty atmospheres.

Electric sparks: Electric sparks from short-circuiting and arcing at electrical equipment or overhead trolley wires may ignite an explosive dust—air mixture. Sparks of higher voltage and amperage are necessary than in the case of flammable firedamp mixtures. Static electric sparks can also ignite explosive dust—air mixtures. Fine particles of dust may readily become electrified by friction with air or ducting through which they pass. As the electric charge on a body resides on the surface, a dust cloud has a very large capacity. Under suitable conditions, a discharge or sudden recombination of separated positive and negative charges can occur which can then act as a source of ignition. With increasing humidity, however, the electric potential falls.

Firedamp Explosions: A firedamp explosion is the commonest source of initiation of a coal-dust explosion. Besides posing the danger of such direct ignition, a firedamp explosion may raise the deposited dust from the mine floor, sides, or roof into mine air very quickly before its flame has ceased and then propagate as a coal-dust explosion. A very small gas explosion initiated by accidental ignition of a small quantity of flammable firedamp mixture (approximately 0.4 m3 volume may thus bring about a much bigger coal-dust explosion. This danger is particularly great in long headings than on long coal faces due to the low air velocities and a lack of adequate pressure relief except in one direction towards the entrance of the heading. An explosion in pure coal dust may not develop when layered gas is ignited or where the gas is ignited at the outbye end. It is significant that most firedamp explosions do not develop into coal-dust explosions due to their failure to raise a sufficiently dense dust cloud.

Measures which prevent or reduce formation and dissemination of coal dust underground

The elimination of coal dust in mines would be the logical means of eliminating coal-dust explosion hazard, but unfortunately, this is virtually, impossible.

Much can, however, be done to reduce the formation, distribution and accumulation of coal dust in mine workings.

The following important measures should be adopted:

(i) On longwall faces, water infusion of the coal face to reduce respirable dust at low or high water pressures where the seam and adjacent strata allow. The different methods of infusion of coal in situ before it is mined as practised in European mines are shown in Fig. 2.31. Infusion is generally practised on faces in seams which are ploughable. As it is difficult to infuse fast-moving mechanized faces with non-cyclical mining, the method of long (deep)-hole low-pressure infusion by means of holes drilled into the seam parallel to the face in advance of the coal face from the advanced gate roads will be the most satisfactory method. Long-hole high-pressure infusion may also be applied as a larger area of coal can be infused. The effectiveness of coal face infusion depends on the amount of water injected and its uniform distribution in coal. The amount of water required lies between 8 and 10 litres per solid cubic metre of coal. Insertion of a solid calcium chloride cartridge into the infusion hole prior to infusion has been shown to decrease the total dust make considerably (up to 50 per cent) [45]. Water mixed with surfactants improves dust suppression compared to plain water. The disadvantages of wetting of coal seams before its extraction are consumption of considerable amounts of water and energy and the possibility of wetting a whole section of the seam.

(ii) Wet winning of coal using wet pneumatic picks where used.

(iii) With machine-cutting by means of coal cutters, using sharp picks of a suitable type, selecting optimal cutting and travelling speeds of the machine, using gummer and wet-cutting; with continuous miners by using scrubbers on them.

(iv) When blasting in coal, use of stemming cartridges or ampoules of calcium chloride powder containing 82 to 85 per cent $CaCl_2$ and 15 to 18 per cent water of crystallization reduces considerably the dust produced by shotfiring due to the formation of a large number of droplets as a result of distribution of hygroscopic salt particles in the fume cloud containing a considerable amount of water vapour, which bind the dust.

(v) With power loading using the conventional shearer loaders, by not using too many cutting bits which should be sharp and of a suitable type and bit lacing, selecting a suitable drum design with vane angle greater than 12°, operating at optimal rotational speed of the drum by using a two-speed gearbox, and at optimal travelling speed of the machine, with proper direction of drum rotation, preferably roof to floor; introducing water to the bit clearance line through the hollow drumshaft by having sprays in the shell or barrel (barrel release) or mounted on the vanes (vane release); flushing the cutting edges of the bits with sprays located either in or close to the bit boxes (behind-the-bit flushing, Fig. 2.32) and using external water sprays on the body of the shearer on the face side to deliver water to strategic areas. Internal water sprays with a water pressure greater than 15 bar have been found to be more effective than external water sprays. In hot and deep mines, or in seams with soft floor, the internal water feed may be phased so that water comes out through the nozzle outlets near the bits only when they are actually cutting, since spraying the dust already generated is not effective enough.

(vi) Thoroughly wetting the coal pile before it is manually or mechanically loaded. Using such types of conveyors with which the dust production is minimum.

(vii) With belt conveyors; (a) covering them along their length with a hood (Fig. 2.33); (b) complete shrouding or boxing of the conveyor for a certain distance before the entry and after the exit when passing through air-locks with some form of sea] (suspended conveyor belting) to allow passage of materials; (c) having properly designed and installed chutes and skirt boards at transfer and loading points. It is most advantageous to hood or enclose the transfer points; and (d) water spraying transfer and loading points.

(viii) Using large-capacity mine cars.

(ix) Water spraying full and empty trains during their transit.
(x) With rope haulages, raising the haulage ropes by correct siting of the rollers to prevent contact with the floor containing dust.
(xi) Preventing spillage and degradation of coal during transport on haulage roads by:
(a) using undamaged, well-maintained dust-tight cars;
(b) avoiding overloading of cars so that there is no spillage in transit;
(c) using retarders and creepers to avoid violent collisions;
(d) maintaining the haulage track in good condition; and
(e) on coal conveyor roadways, reducing spillage by selecting proper belt width and speed for operating the conveyor at 75% of its full rated capacity, by good alignment and transverse levelling providing adequate bunker capacity at loading points, centralizing the flow of coal, and keeping the number of belt joints to the minimum and also sealing them.
(xii) Restricting velocities of air currents in mine airways to less than 3 m/s if possible.
(xiii) Adopting homotropal ventilation where the ventilating an travels in the same direction as the coal to reduce dust pick-up.
(xiv) Preventing dust accumulation in mine workings by:
(a) dry suction at loading and unloading points at which large quanties of dust are produced which cannot be suppressed in the ordinary way, and removing dust by means of collectors which may be of the fabric, dry cyclone, or wet impingement type;
(b) incorporating a dust collector in the auxiliary ventilation system in development drivages and headings;
(c) cleaning systematically and regularly main haulage roads and maim ' return airways of deposited dust (three to four times a year) using transportable roadway suction apparatus;
(d) cleaning regularly and systematically at and near transfer and loading points;
(e) installing skip-hoisting in upcast shafts; and
(f) locating dry coal preparation plants far away from downcast shafts (not less than 8O m).
(xv) Selecting a method of coal winning with which dust production is the least. Winning by coal ploughs produces much less dust than by shearer leaders.
(xvi) Controlled caving of roof coal in very thick seams mined by the sub-level caving method using close-fitting shields.
(xvii) Consolidating the floor dust to prevent it from being raised. In open cast coal mines, important dust sources include:
(1) Mine haul roads
(2) Removal of top soil ahead of mining
(3) Blasting of coal
(4) Stockpiling of coal and its reclamation by use of reclaiming equipment
(5) Tailings dams

10-100 µm and < 10 µm – issues of caking/agglomeration causes loss of flammability

(2 + 3 + 2 = 7 marks)

32. **(i)** What are the different causes of inundation in mines? **(ii)** Explain the different preventive measures against inundation.

Inundations by surface waters

Inundations by surface waters may occur after sudden and abnormally heavy rains when the surface mine outlets such as shafts, inclines, and adits may get flooded or subsidence fractures reaching the surface get submerged.

They are very dangerous as large quantities of water break into the mine within a short time preventing escape of workmen.

Inundations from overlying strata

Inundations from the overlying strata can occur under one or more of the following conditions:

(1) When impervious strata are pierced by mine workings;

(2) When fissures or fracture planes develop in impervious strata due to subsidence communicating with water-bearing strata above;
(3) Where faults, fissures or fracture planes in communication with a water-bearing bed are intersected by mine workings;
(4) Where the deposit occurs under a water-bearing bed, lying uncomformably on the eroded surface of the deposit;
(5) When a mine working is too near to the surface and accidental holing has taken place into a pond, stream bed, or outcrop workings;
(6) Where boreholes drilled for prospecting have not been sealed-off; and
(7) Mining coal from beneath the seabed as at Ellington mine, UK Coal.

Inundations from water-logged workings

Inundations from water-logged workings occur when they are accidentally holed or when barrier pillars fail due to inadequate design.

They are not easily predictable and the danger from them is especially great in semi-steep and steep formations.

These measures can be divided into: surface measures and underground measures.

The surface measures aim at preventing dangerous accumulations of water on the mine surface.

The underground measures offer protection against inundations.

Both surface and underground measures may be adopted to control inundations with varying degrees of effectiveness depending largely upon geological and hydrological conditions.

Before mining, a suspected water-rich area could be drained by drilling of building a reliable drainage system.

Surface measures

They can be further divided into: (a) measures against flooding of mine main entries or outlets; and (b) measures against seepage of surface waters.

The selection of one or more remedial measures is governed by the cost-benefit ratio and the method of financing.

Measures against flooding of mine main entries

These consist of the following:
(1) Locating shaft sites away from faults especially formed by major tensional stresses;
(2) Laying mouths of shafts, inclines, and adits above the high-water mark and lining them for the first 20 m with watertight lining.
(3) Filling up with debris and other sealing materials all abandoned shafts and boreholes which have ceased to serve any purpose with the existing mine;
(4) Cutting diversion drains or ditches or erecting embankments or concrete walls on the surface to intercept and conduct the surface run-off water away; and
(5) Constructing dams and reservoirs in the upper reaches of rivers to prevent their flooding.

Measures against seepage of surface waters

Depending on the nature of the bed(s), surface water (streams, ponds, tanks, lakes) lose considerable quantities of water which may enter mine workings or form water pools in the strata. The infiltration of surface waters can be controlled by one or more of the following measures:
(1) Straightening, cleaning, widening, and grading stream channels to provide free flow of water especially in areas with vertical fractures, subsidence, or high-permeability rocks;
(2) Silting or lining stream beds with concrete or rubble masonry;
(3) Grouting the river bed with cement to stabilise and seal it;
(4) In the case of near-surface mines, laying ponds and tanks dry if they form a cause of water influx in mines;
(5) Diverting streams/rivers, if technically and economically feasible, with safety to new and safe channels;
(6) Back-filling surface excavations or subsided areas with impervious materials to an established hydraulic gradient that will ensure natural drainage;

(7) Diverting run-off to prevent its infiltration into fissures on subsided areas;
(8) Conducting run-off across pervious or subsided areas of the mining property by means of flumes;
(9) Grouting overlying strata to reduce permeability and the flow; a seismic reflection survey on the surface will reveal the location of subsurface anomalous structures such as fractures, joints or faults for the drilling and grouting programme.
(10) Leaving outcrop barrier pillars of adequate size;
(11) Selecting a method of extraction, especially of thick seams, by which the Strata subside uniformly without fracturing. For extraction under important waterways, hydraulic stowing is the best method of roof control; and
(12) Afforestation along river banks.

Underground Measures

These protect life and property against inundations. They consist of the following measures:
1. General lowering of the water table to below the level of the workings by borehole pumps was extensively applied for a long time for opencast mines but it suffers from the disadvantage that a stage might be reached when further dewatering would severely interfere with water resources, wells, etc.
2. Avoiding blasting and drilling operations near suspected water-logged workings.
3. Leaving safety water pillars or barriers. A safety water pillar is that portion of the bed that is left unmined:
(i) below an overlying water formation along boundary line or lines of adjoining properties;
(ii) between parts of a mine;
(iii) below surface streams, tanks and lakes;
(iv) around shafts; or
(v) along major faults.

Its principal function is to act as a dam to prevent water accumulations from suddenly breaking into mine workings. Lack of maintenance of dependable pillars by depending on old surveys and incorrect old plans had resulted in several mine disasters in the past.

The vertical thickness of the safety pillar against water-bearing strata overlying a bed is generally prescribed by the mining regulations. In coal mines, it should not be less than 20 m.

The width of a safety pillar between adjacent mines is usually prescribed by the mining regulations. In coal mines, it should be at least 20 m on either side of the boundary line at right-angles to it.

If possible, a mine should be laid out in districts or panels according to their hydrology. The panels should have substantial safety barriers on three sides with a minimum number of entries or drivages through them so that, after mining, each panel can be isolated from the others.

The width of a safety pillar-barrier to be maintained against abandoned waterlogged workings depends on the following factors:
(1) The degree of uncertainty regarding the exact location of the old workings;
(2) The thickness of the bed;
(3) The nature of the bed whether it is friable, contains dirt partings, or has been crushed and fractured by folding;
(4) The inclination of the bed and the direction of the pillars in relation to the line of full dip;
(5) The maximum hydrostatic head;
(6) The nature and condition of the adjacent strata whether pervious or broken;
(7) The method of extraction and roof control on the side of the pillar;
(8) The presence of faults, their throw and direction; and
(9) Effect of mining other beds.

The geophysical electrical resistivity determination method can be used with reasonable certainty for determination of coal barrier thickness against water logged workings or delineation of hidden water-filled galleries within the barriers. There is, however, no general

rule or formula for determining the size of safety pillars as no rule can possibly serve all cases.
4. Supporting roadways not coming under the influence of mining operations with water-tight lining.
5. Cementation of fractured strata in the roof containing water in fissures.
6. Driving drainage tunnels or adits to de-water the property. This method had been practised in certain metalliferous mines in the early days of mining.
7. When driving through hard ground traversed by water-bearing fissures, adopting a technique of diamond drilling cover ahead of the advancing face in which long holes (100 m) are drilled ahead of the face from bays (40 m apart) excavated on alternate sides of the roadway (Fig. 8.1);
8.. Providing adequate sump and pumping capacities at predetermined points for dealing with inrushes of water even where safety pillars are provided.
9. Erecting water dams or hydraulic seals to seal abandoned sections of the mine.
10. Erecting bulkhead doors in mine workings with immediate danger of inundation.
11. Providing additional lodgement capacity in worked-out area(s) to which sudden inrushes of water may be directed in an emergency.

or

(i) What are the causes of coal mines fires? **(ii)** How to prevent spontaneous combustion of coal in mines during development and depillaring operation?

Various causes of coal mines fires may be grouped under the following main headings:

Open Flames
Spontaneous combustion
Electricity
Friction
Blasting
Explosions
Miscellaneous

Development

(1) Main shafts should be properly located, lined, and equipped so that there is less pressure loss.
(2) Development within the coal seam should be kept to a minimum. All coal surfaces not being directly worked on must be protected against absorption of oxygen by stonedusting, guniting, or coating with an antipyrogene (a fire-proofing composition).
(3) When working seams by the bord and pillar method:
(a) the dimensions and shape of the pillars must be such that no crushing of the pillars takes place before or during their extraction which will increase the air permeability of the fractured coal;
(b) panels with independent ventilation should be formed as far as practicable;
(c) the panels must be laid so that severe crushing of coal during extraction is minimized;
(d) the panels formed must be of a size which would permit complete extraction within the incubation period; and
(e) the development work should be kept out of abutment areas.
(4) When designing longwall face layouts, face lengths should be selected so that good closure and compaction of the goaf are assured. Face lengths smaller than 60 to 65 m are generally not recommended.
(5) The roof of roadways in-coal should be well-supported and the sides (ribs)well-lagged.
(6) When traversing seams with cross-cuts, coal should be removed to an adequate depth and the cavities so formed should be filled or packed with clay, pulverized ash with portland cement, or other incombustible material.
(7) Side-by-side intakes and returns must be avoided as far as practicable. The main intake and return airways and the main haulage roadways should be driven in stone as far as

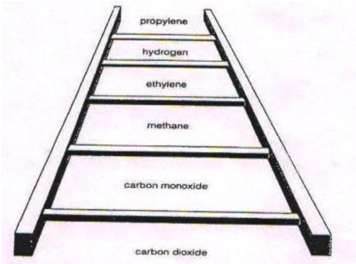
practicable. In thick and inclined seams, the main roadways should be driven in the floor of the seams with crosscuts driven to the seams.
(8) The maximum air travel distance from surface to surface (which ranges from 1 to 20 km) must be kept low so that the number of stoppings and the leakage that takes place are reduced.

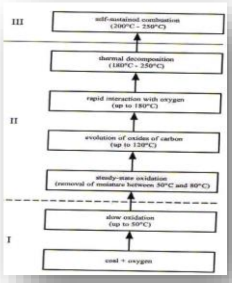
Coal Winning or Depillaring or Pillar Extraction

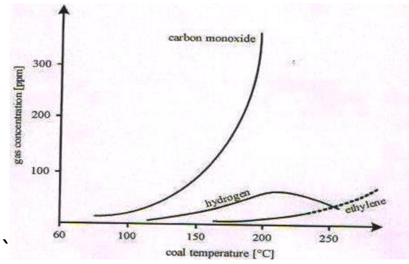
(1) The coal should be won as completely as practicable using an efficient extraction method, especially in disturbed zones, thick seams and strongly folded steep seams.
(2) The working faces should be worked intensiely at a high rate of extraction is the best means of preventing fires due to spontaneous combustion. The advance or retreat of a coal face should never be interrupted and, if any local fall of roof or a fault is encountered, immediate steps should be taken to get over the obstacle or minimize its effect on the face advance.
(3) When mining a group of seams or mining thick seams by longwall slicing, the workings in upper seam slice should be in advance of those in a lower seam or slice and the coal of the upper seam Or slice should be extracted as completely as practicable.
(4) The best methods of working seams are the longwall and stall methods.
(5) When working a seam by-longwall mining, the retreating longwall should be preferred as it eliminates stray leakage currents through the goaf behind the face if regular caving of overlying strata behind the face is accomplished. Retreating is the best method if there is a goaf above the seam. It is the only method for working thick inclined seams by sub-level caving with inertization of the goaves behind faces using liquid nitrogen or nitrogen foam to combat the high spontaneous combustion risk.
(6) As it is not always possible to avoid coal remains in the goaves of longwall faces even when powered supports are used for extracting seam by longwall advancing with caving, the gatesides pack should be airtight in order to increase the airflow resistance of the goaf; also one must balance the pressure on all sides of the goaf. When extracting a seam by retreating longwall, the tail gate and the main gate behind the face line must be stowed as tight as possible or they should be sealed off at regular intervals. When starting a new face from solid, the gate pack must be well tied into the barrier pillar.
(7) Worked-out areas should be effectively sealed off as soon as possible or ventilated. Stowed seals should be preferred to other forms of seals. If the worked-out areas are ventilated, as they are in most American coal mines, a system of bleeder entries must be planned.
(8) Blasting in coal must be restricted as much as possible.
(9) Loose coal and other combustible material, such as oily or greasy waste and decaying timber, should be cleaned up and not be permitted to accumulate.
(10) In bord and pillar workings, the pillars should be mined as fast as practicable at an even rate after their formation.
(11) Partially-built seals or preparatory stoppings should be erected in the entries of panel fire barriers before extraction is commenced.
(12) When extracting folded seams, the limbs of a synclinal or anticlinal fold must be extracted, as far as possible, simultaneously and uniformly with stowing.
(13) When working seam outcrops with shallow overburden, barrier pillars of sufficient size be left to the surface so that air leakage through fissures is avoided,

(3 + 3 = 6 marks)

33. **(i)** What are the different stages of oxidation and spontaneous combustion of coal? **(ii)** What are the factors governing the design of a dam? **(iii)** How salt zones are effective in dealing with the problem of coaldust explosion? **(iv)** What are the different circumstances under which a water dam is required to be erected in underground?







Stages of oxidation and spontaneous combustion of coal

One recognizes three distinct stages of combustion of coal in coal mines

(1) The incubations period;

(2) The indication period; and

(3) Open fire.

The incubations period;

The incubation period is the period between the onset of first oxidation and the particular time at which one can detect it by the senses. From a practical point of view, the term, however, is used in a broader sense to denote the period between the first roof fall after beginning of coal extraction in a district or panel and the appearance of the first signs of heating which may take 90 to 120 days after the coal is mined. The incubation period varies widely depending on the seam thickness, nature of the immediate roof, method of working, method of roof control, regularity and continuity of working, and liability of coal to undergo spontaneous heating. It usually varies between 3 and 18 months, higher rank coals having longer periods. During this period, one does not detect heating during one's passage through the mine workings.

The indication period

The end of the incubation period marks the beginning of the indication period which is marked by 'sweating' and haze in the air, caused by the warmed-up air from a fire coming into contact with the cooler coal, rock, and metallic surfaces and depositing moisture.

The indication period is often of very small duration lasting sometimes only a few hours. It comes to an end with the appearance of a slight delay 'fire stink' when an open fire with visible active combustion breaks out at 70ºC.

Open fire

The fire stink can be easily recognized by the characteristic petrolic smell. Seams seldom burn with a bright flame but glow, developing bluish-white clouds of smoke

The factors governing the design of a dam are:

(1) Size of roadway;

(2) Nature of the adjacent strata;

(3) Estimated water pressure;

(4) Materials used in construction; and

(5) Form of dam.

Water dams are permanent artificial barriers or seals built in mine workings including shafts, slopes, drifts and adits under any one of the following circumstances:

(1) To guard against irruptions of water from adjacent water-bearing strata;

(2) To guard against irruptions of water from adjacent old workings;

(3) When approaching water-logged workings and draining off the water by ~ means of boreholes;

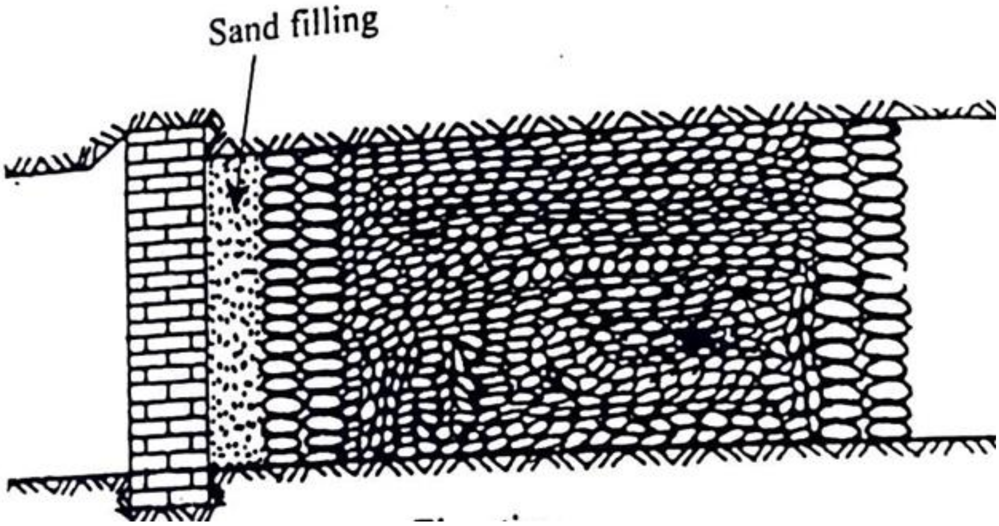
(4) To limit the amount of pumping by allowing water in worked-out areas to accumulate behind the dams;

(5) To reduce or eliminate acid water being discharged in mines with the acid mine drainage problem;

(6) To seal off operating mines that will close in the future;
(7) To isolate acid-producing zones in a mine where they can be identified from active mine workings where natural barriers cannot be provided; and
(8) In most cases, to withstand the maximum hydrostatic pressure that may develop.

or

(i) What are the parameters to be considered before reopening a sealed-off area? **(ii)** What are the methods and preparatory measures of reopening? **(iii)** Draw a neat sketch of combined brick-and-sand stopping in an underground mine. **(iv)** Briefly explain the different factors affecting development of coaldust explosion?



Considerations when Reopening a Sealed-off Area

The reopening of a sealed-off area should not be undertaken lightly.
It should be undertaken only when the conditions ensure the safety of those engaged in the re-opening work and prevent additional damage to property.
A certain amount of time must elapse before conditions become favorable for reopening.
The time of reopening a fire area is governed by the following factors:
(1) Extent and intensity of fire at the time of sealing
The extent and intensity of fire at the time of sealing influence the reduction of oxygen inside a sealed-off are rea (well below 10%) if there is no leakage of air into it.
A larger amount of burning material will bring about a quicker reduction of oxygen than a smaller amount but will, at the same time, lengthen the time required for cooling off.
(2) Nature of burning material and adjacent strata
The nature of the burning material inside a sealed-off area, whether it is timber, coal or both and the rate of burning may influence the possibility of rekindling of fire on admission of air. Coals having a higher volatile-combustible ratio burn faster and are more likely to be rekindled than coals having a lower ratio. The nature of the adjacent Strata, especially when they are combustible, influences the time of reopening as they retain heat for a long time even after the oxygen content has fallen below the limit when combustion ceases. Valuable information can be obtained by the use of thermocouple in the sealed area.
Sub-micrometer particle concentrations in the sealed area have been found to correlate exceptionally well with the temperatures within the area, independent of the combustibles involved in sharp contrast to the gas concentration data which exhibit no correlation with the temperatures, and their measurement may be used as a more universal technique for determining the status of the underground scaled fire.
(3) Airtightness of stoppings and the sealed-off area
Airtightness of stoppings and the enclosed area js essential for control of oxygen.
A fire is effectively sealed off when there is a constant decrease of oxygen in the samples taken from behind the stoppings. Air may leak through the stoppings and adjacent strata due to barometric and temperature changes or due to differences in ventilation pressure

between different parts of the area, thereby prolonging the process of extinguishment of the fire.

(4) Composition of the atmosphere in the sealed-off area

Knowledge of the composition of the atmosphere obtaining in a sealed area is of vital importance in deciding the reasonably safe time of reopening the area.

Trends of several gases may provide a better indication of the status of a fire.

The value of the composition lies in its correct interpretation to ascertain the status of fire and possible explosion hazard upon admission of air.

Samples of atmosphere should be collected through surface boreholes wherever possible. Otherwise, the first trip into the mine after sealing should be for the pupose of collecting samples from the sealed-off area.

In interpreting the composition, it should be remembered that active combustion ceases when the percentage of oxygen falls below 12, but that slow combustion with evolution of CO and CO_2 may continue even at oxygen concentrations of 5 per cent or less.

(4) Composition of the atmosphere in the sealed-off area

The presence of combustible gases other than methane, which reduce the lower flammability of methane, can be disregarded if their total content does not exceed 1 per cent as the amount of oxygen necessary to form an explosive mixture as compared to methane alone is insignificant.

With the composition of firegases as is normally found in sealed-off areas, there is less likelihood of any explosion taking place during reopening if the oxygen content of the gases is reduced to 3 per cent or less.

Carbon dioxide, although exerting an extinctive effect upon the mixture of gases, is seldom more than 6 to 8 per cent and, for practical purposes, can also be disregarded except for the information it provides on the depletion of oxygen in the area. The levels of carbon monoxide may never reach zero even though it may be considered safe to re-enter the area. It should, however, be pointed out that no method of ascertaining the status of fire or evaluating the danger of explosion can offer any guarantee of safety as it is nearly impossible to predict accurately the air leakage, the rate of methane emission and the reliability of the gas samples as representative of conditions in the vicinity of the fire seat.

(4) Original Cause of Fire

If the fire was due to spontaneous combustion, rekindling of the fire may take place and contingency plans should be drawn up to meet the situation,

A fire caused by machinery would be considered in a different light..

(6) Inert gas injection

The time of reopening a sealed area can be considerably reduced and the area may be safely entered by injection of an inert gas into it.

Preparatory Measures for Reopening

For the successful reopening of any sealed-off fire area, certain precautionary measures should be undertaken. They are:

(1) Assembling rescue and recovery crew;

(2). Withdrawal of all men from the mine other than those required in connection with reopening operations;

(3) Stationing a man at the mine main fan to see that the fan continues to run,

(4) Cutting off electric power from the part of the mine in which a fire issealed off as well as in return airways utilized for carrying the firegases;

(5) Making necessary adjustments in ventilation so that the return from the firearea can be directed into the main return;

(6) Heavily stone dusting all roadways leading to and from the fire area;

(7) Establishing telephonic communication between the fresh-air base and thesurface; and

(8) Preparatory measures for nitrogen flushing during reopening where it 1scontemplated.

(9) An action plan should be prepared and agreed to by all concerned parties.

Methods of Reopening

For reopening a sealed-off area, one of the following methods may be employed

(1) Reopening by reventilation;
(2) Reopening by air-locking in stages (stage method); and
(3) Combination of (1) and (2)

Different factors affecting development of coaldust explosion.

Particle size

Thermal ignitibility studies on coals of different volatile contents have shown that coal-dust particles up to 750 to 1000 µm [36] take part in explosion, depending on the rank of the coal, nature and intensity of the ignition source, and oxygen concentration and that fine particles control the ignition, violence, and speed of flame propagation.
Dangerous are the particles which lie between 10 and 100 µm size.
The loss of flammability of dust below the 10 µm m size is explained as due to chemical decomposition of dust at that degree of fineness, a tendency to agglomerate, and rapid oxidation on initial exposure to air, thereby becoming less easily ignitible.
At larger particle sizes, the rate of de-volatilization decreases rapidly due to the relatively short pre-heating time available.
This tends to limit the overall rate of flame propagation through the mixture and the mass concentration of dust required to produce a limit concentration of combustible volatiles increases markedly [36].
Decrease in particle size also increases the capacitance of dust clouds with possible development of electrostatic discharges of sufficient intensity under suitable conditions, which may ignite a dust cloud.

Dustiness of mine workings

It is customary to express the quantity of deposited dust in g/m3 of mine excavation and the concentration of airborne dust in g/Nm3 of air, though, without knowledge of the particle size distribution, these expressions are meaningless.
The determination of the lowest limit of the deposited dust in a mine working at which an explosion can occur is very difficult as the whole of the dust need not necessarily be suspended in air for an ignition to take place.
From experiments conducted in experimental mines [25], it had been found that a mine working is dangerously dusty if it contains 100 to 120 g/m3 and that the most violent explosions occur at 300 to 400 g/m3.
The minimum density of a dust cloud, which will propagate an explosion, depends on the nature of the source of ignition, fineness of the dust, rank of the coal, and other parameters.
The dust cloud must be dense immediately surrounding the ignition source so that one cannot see through it.
As the dust concentration in mine air currents is not sufficient to propagate an explosion, the explosion must stir up the deposited dust along its path to create a cloud for its propagation.
Figure 2.22 shows a typical thermal ignitibility curve for a coal dust showing the relation between the ignition temperature and the coal-dust concentration.
It represents the boundary between ignitable and non-ignitible mass concentrations at a given thermal ignition temperature.
It will be seen that the minimum value of the ignition temperature is approached asymptotically at the higher dust concentrations.

Volatile matter content

The flammability of a coal dust depends greatly on its combustible volatile matter; it increases with increasing volatile content.
The volatile matter is usually calculated on a dry, ash-free basis using the formula.

% VM = $\frac{\%\, VM\ (from\ analysis)\ x\ 100}{100 - \%\, ash - \%\, moisture}$

As the right-hand side of the above formula may be written as $\frac{\%\, VM}{\%\, VM + \ \%\, FC}$ the percentage of volatile matter on a dry, ash-free basis is sometimes called the volatile-combustible ratio or simply the volatile ratio of coal.

Several investigations in laboratories, surface experimental galleries, and experimental coal mines on the flammability of coal dusts had shown that coal dusts with VM < 10 per cent are non-flammable, those with VM = 10 to 14 per cent are less flammable, whilst those with VM > 14 per cent are highly flammable.
The increase in flammability, however, is different for different coals.
For low-volatile coals, the flammability measured by the amount of incombustible required to prevent explosion flame propagation, increases almost linearly with increasing volatile ratio.
Volatile matter content
But above about 25 per cent, the volatile ratio in the range of medium-and high-volatile bituminous coals, the flammability rises only slightly with increasing volatile ratio.
From statistics of coal-dust explosions in the coal mines of Germany and the UK, it was noticed that explosions did not take place with coal dust containing less than 20 per cent volatile matter (DAF).
Percentage of ash
An increase in ash content or presence of inert foreign material reduces the ignitibility or flammability of coal dust because of heat absorption.
Stone-dusting in coal mines is a practical application of the use of inert dust to prevent an explosion from taking place.
Percentage of moisture
Moisture in dust particles raises the ignition temperature of the dust.
It exerts a cooling effect because heat is absorbed during its heating and vaporization, thereby reducing the energy available for ignition of the dust cloud.
It also tends to wet and agglomerate the fine particles of dust, reducing their dispersability.
Coal dust loses its dispersability with a moisture content of 25 to 30 per cent.
It loses explosibility at 50% moisture content.
The effect of moisture on flammability is unimportant below about 10 per cent.
The drying out of the deep mine workings due to air currents increases the hazard due to coal dust in winter more than in summer months.
Oxygen concentration
Variations in oxygen concentration affect the ease of ignition of dust clouds and the explosion pressures.
With decreasing oxygen concentration, the ignition energy required increases, ignition temperature increases, and maximum explosion pressure decreases.
Nature and intensity of ignition source
The nature and intensity of the ignition source (temperature, size of spark or flame, etc.) exert a great influence on the flammability of the mine dust under mining conditions as they determine the dust-raising capacity and turbulence induced within the cloud.
Explosions initiated by strong sources develop faster and cause more damage than explosions initiated by weak sources.
The igniting sources more often than not are the flame of an exploding firedamp, flame accompanying the detonation of an explosive, or an electric spark caused by a damaged electric cable.
Percentage of firedamp in mine air
The presence of firedamp in air in percentages less than its own lower limit reduces the lower explosive limit of coal dust by replacing the coal on a weight thermal basis.
One per cent CH_4 (by volume) in 1 m^3 of air is equivalent to about 12 g of coal dust.
The flammability of coal dust increases almost directly in proportion to the percentage of firedamp (Fig. 2.24) [39].
Turbulence
As the combustion of dust takes place at the surface of the dust particles, the rate of reaction, therefore, depends on how intimately the dust and oxygen are mixed.
Turbulent mixing of dust and air will result in more violent explosions than those caused by ignition in relatively quiescent mixtures.

Surrounding conditions

The size, shape, constrictions, obstructions, branching, length, and nature and condition of the surfaces of mine workings exert an important influence on the development of coal-dust explosions as they increase or decrease the progress of a flame by holding or releasing the pressure.

(3 + 2 + 2 + 3 = 10 marks)

Section C: Numerical Type Questions – 18 marks

34. How long could a man remain conscious in a heading 2.0 m x 2.5 m in cross-section when it has been sealed by a fall 10 m behind the face? The air in the heading contains 20.1% O_2 and 0.1% CO_2. Assume unconsciousness to follow 7% O_2 or 10% CO_2 in the air. The average O_2 breathing rate of the man may be assumed as 0.3 x 10^{-3} m^3/min with a respiratory quotient (which is equal to CO_2 produced by O_2 inhaled) of 0.82. Determine whether O_2 deficiency or the effect of CO_2 will have the dominating influence.

(7 marks)

Volume of air in the sealed 10 m x 2.0 m x 2.5 m is 50 m^3
Initial 20.1% oxygen = 0.201 x 50 = 10.05 m^3
O2 content at which man will fall unconscious is 7% oxygen = 0.07 x 50 = 3.50 m^3
average O_2 breathing rate = 0.3 x 10^{-3} m^3/min
Oxygen will last up to $\frac{10.05-3.50}{0.3 \, x \, 10^{-3}}$ = 21833 min
Respiratory quotient = $\frac{CO_2 \text{ produced}}{O_2 \text{ inhaled}}$ = 0.82
produced rate = 0.82 x 0.3 x 10-3 m3/min = 0.246 x 10^{-3} m^3/min
Initial 0.1% carbon dioxide = 0.001 x 50 = 0.05 m^3
CO2 content at which man will fall unconscious is 10% carbon dioxide = 0.1 x 50 = 5 m^3
Time = $\frac{5-0.05}{0.246 \, x \, 10^{-3}}$ = 20121 min
It is found that the man may remain conscious for 20122 min = 13.97 days and the unconsciousness will be due to CO_2.

35. Three air streams meet to form a common main return air stream of 1550 m^3/min. The initial air streams contain CH_4 as follows: **(a) 300 m^3/min** — No methane (b) 650 m^3/min — 0.7% CH_4 (c) 600 m^3/min — 1.1% CH_4. In addition, there is a blower in the main return airway giving 2.5 m^3 of pure methane per minute. Calculate **(i)** Final concentration of CH_4 in the return airway assuming perfect mixing. **(ii)** Volume required from airway **(a)** to reduce the final CH_4 concentration by 30% assuming other quantities are not changed.

(7 marks)

Methane in intake air + Methane emitted = Methane in return air
Let concentration of methane in return air is C%.
(0 x 300 + 0.007 x 650 + 0.011 x 600) + 2.5 = 0.01C x (300 + 650 + 600 + 2.5)
C = 0.88 %
Final concentration of methane in return air is 0.88 %
Let the quantity required from (a) is Q m^3/min to reduce the final CH_4 concentration by 30% = 0.7 x 0.88 = 0.616 %
(0 x (300+A) + 0.007 x 650 + 0.011 x 600) + 2.5 = 0.00616 x (Q + 650 + 600 + 2.5)
Q = 963
The volume of 963 m^3/min would be required from airway (a) to reduce final concentration of methane in the return airway by 30%.

36. Air enters a bord and pillar panel at 10 ppm CO, 20.95% O_2 and 78.08% N_2. The air at the panel return has 80 ppm CO, 20.05% O_2 and 79.22% N_2. Calculate the CO index and 'Graham's ratio' and also mention about the status of fire in the panel.

(4 marks)

CO Index $= \frac{final\,CO - initial\,CO}{initial\,O_2 - final\,O_2} x100 = \frac{(80-10)\,x\,10^{-4}}{20.95-20.05} = \frac{70\,x\,10^{-4}}{0.90} = 77\,x\,10^{-4}$ (Normal condition)

Grahm's Ratio = $\frac{100\ x\left(Carbon\ Monoxide_{Final}\ x\frac{Nitrogen_{Final}}{Nitrogen_{Initial}} - Carbon\ Monoxide_{initial}\right)}{\left(Oxygen_{Initial}x\frac{Nitrogen_{Final}}{Nitrogen_{Initial}}\right)-Oxygen_{Final}}$

= $\frac{100\ x\left(80\ x\ 10^{-4}\ x\frac{79.22}{78.08} - 10\ x\ 10^{-4}\right)}{\left(20.95x\frac{79.22}{78.08}\right)-20.05}$ = $\frac{71.16\ x\ 10^{-2}}{21.25-20.05} = \mathbf{0.593}$ **(Existence of heating)**

Partial marks for **CO Index / Graham's Ratio** = $\frac{100\ x(Carbon\ Monoxide_{Final})}{(0.265\ x\ Nitrogen_{Final})-Oxygen_{Final}}$

= $\frac{100\ x(80-10)x\ 10^{-4}}{(0.265\ x\ 79.22)-20.05}$ = $\frac{70x\ 10^{-2}}{0.94}$ = **0.74 (Existence of heating)**