Part 5

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21.1 INTRODUCTION

The most feared of hazards in underground mines or other subsurface facilities are those of fires and explosions. Like airplane crashes, these do not occur often but, when they do, have the potential of causing disastrous loss of life and property as well as a temporary or permanent sterilization of mineral reserves. Furthermore, "near-misses" occur all too frequently. The incidence of mine fires appears not to be declining despite greatly improved methods of mine environmental design and hazard control. This is a consequence of several matters; first the growing variety of materials that are imported into modern mine workings, varying from resins and plastics to liquid fuels and hydraulic fluids. A second factor is the continuous increase in the employment of mechanized procedures, many of the machines involving flammable liquids and materials that can produce toxic fumes when over-heated.
Although this chapter concentrates on technical considerations the incidence of mine disasters is greatly influenced by the priority afforded to safety and health by individual companies, states and countries. Mine disasters are more prevalent at times when market prices for minerals and fuels are high. Attention paid to mine safety is greatest in those countries that have active and well-funded research activities in this area and which maintain industry awareness through an ongoing output of reports and papers. Such work also assists in the promulgation of new or updated legislation.

These activities are often inter-related. A period of declining statistics of mine fatalities coupled with a period of limited research tends to create a laissez-faire attitude in both industry and legislative bodies or agencies. A boom market is then more likely to produce a spate of serious mine accidents. Public disquiet results in (often hasty) legislative action to promote new research investigations and the imposition of stricter standards and penalties on mining companies. Unfortunately, public memory tends to be short and history has shown the repetitive nature of this cycle. It is within that context that this chapter has been written.

The enormous loss of life due to mine fires and explosions during the eighteenth and nineteenth centuries preoccupied the minds of mining engineers and scientists of the time (Chapter 1). The majority of deaths arising from mine fires and explosions are caused, not by burning or blast effects, but by the inhalation of toxic gases, in particular, carbon monoxide. There are two major differences between underground fires and those that occur in surface structures. The first concerns the long distances, often several kilometres, that personnel might be required to travel in passageways that may be smoke-filled. Secondly, the ventilation routes are bounded by the confines of the airways and workings, causing closely coupled interactions between the ventilation and behaviour of the fire.

It is difficult for anyone who has not had the experience, to comprehend the sensations of complete isolation and disorientation involved in feeling one's way through a long smoke-filled mine airway in zero visibility. It is a cogent exercise to turn off one's caplamp in an unilluminated return airway and to walk just a few steps, even without the trauma of a highly polluted atmosphere.

It is, therefore, a matter of ongoing importance that all personnel involved in the design and operation of underground openings should have some knowledge pertaining to the prevention and detection of subsurface fires and explosions, as well as procedures of personnel warning systems, escapeways, firefighting, toxic gases, training, fire drills and the vital need for prompt response to an emergency situation. These are some of the topics that are discussed in this chapter.

21.1.1. The fire triangle and the combustion process

Perhaps the most basic precept in firefighter training is the fire triangle shown on Figure 21.1. This illustrates that the combustion process which we term “fire” requires three components: fuel, heat and oxygen. Remove any one of these and the fire will be extinguished. The fuel may be solids, liquids or gases. The liquids and gases might be introduced into the mine environment by natural or mining processes, or may be produced by heating solid materials. Whenever a combustible solid or liquid is heated to a sufficiently high temperature (flashpoint), it will produce a vapour that is capable of being ignited by a flame, spark or hot surface which has the required...
concentration and duration of thermal energy. Gasoline has a flashpoint of -45°C while most commonly available solids require the application of a flame for them to reach flashpoint. The ignition temperature of any given substance is the lowest temperature at which sustained combustion is initiated. Flaming is the process of rapid oxidation of the vapours accompanied, usually, by the emission of heat and light. In the case of self-sustained burning, that heat is sufficient to raise the temperature of the newly exposed or surrounding areas of surface to flashpoint. However, combustion can continue at a slower rate without flaming through the process we know as smouldering. In this case, the oxidation process continues on the surface of the material and produces sufficient heat to be self sustaining, but not enough to cause the emission of vapours in the quantity required for flaming combustion.

The oxygen which forms the third side of the fire triangle is normally provided by the air. Flammable liquids such as the oil of a flame safety lamp will cease to burn when the oxygen content of the air is reduced to some 16 per cent (Section 11.2.2.). Flaming combustion of all kinds is extinguished at oxygen contents of 10 to 12 per cent while smouldering is usually terminated at oxygen concentrations below 2 per cent. However, some materials may contain sufficient inherent oxygen for slow combustion to continue at even further reduced levels of atmospheric oxygen. Coupled with the low values of thermal conductivity of crushed material, this can result in "hot spots" lying dormant in abandoned areas for long periods of time, but capable of re-ignition if a renewed air supply is admitted subsequently.

21.1.2. Classification of mine fires

Fires underground can be classified into two broad groups, open and concealed fires. Open fires occur in airways, faces and other openings that form part of the active ventilation system of the mine and, hence, affect the quality of the mine airflows quickly and directly. As the term might imply, open fires are often accompanied by flaming combustion because of the availability of oxygen and offer the possibility of direct attack by firefighting teams. Conversely, concealed fires occur in areas that are difficult or impossible to access such as caved or abandoned zones. These are usually, but not necessarily, initiated as a result of spontaneous combustion and can occur in both coal and sulphide ore minerals as well as within any imported organic matter such as paper, discarded fabrics (e.g. oily rags) or timbering in abandoned areas. The degree to which concealed fires propagate and pollute the mine atmosphere depends upon the rate at which air leaks through the areas affected. The matter of spontaneous combustion is discussed in further detail in Section 21.4.

21.2 CAUSES OF IGNITIONS

The variety of procedures, processes and materials used in modern mining provides many opportunities for the ignition of flammable materials. However, the most commonly reported causes of fires and explosions in mines are listed in the following subsections.

21.2.1. Mechanized equipment

Machines intended for use underground should be designed to operate with a high degree of safety in a harsh physical environment, and are subject to legal requirements and conditions in most mining countries. It is no surprise, therefore, that the majority of fires attributable to machines arise out of:

- misuse
- lack of proper maintenance
- removal or bypassing of safety features such as diagnostic devices, environmental monitors or thermal trip switches and
- running unattended for long periods of time.
Exhaust systems on diesel equipment should be fitted with particulate filters or water scrubbers that not only reduce airborne pollution (Section 11.3.2.), but also prevent the emission of incandescent particles. Furthermore, hoses, transmission or brake fluids and a variety of components made from synthetic materials on modern diesels may be capable of producing toxic gases when ignited. All vehicles or other diesel equipment should be fitted with on-board fire extinguishers.

It is particularly important that equipment containing significant quantities of oil, such as large transformers or air compressors, should be safeguarded by thermal trips, pressure relief valves and other devices necessary for automatic cut-off in the event of any abnormal condition. Such devices should be subjected to routine testing and maintenance. Wherever possible (and as may be required by law) non-mobile equipment should be located within enclosures with fire-resistant roof, floor and walls, and which are ventilated to a return airway. Again, fire extinguishers and, preferably, an automatic fire suppression system should be available within the chamber. In coal mines, the surroundings in adjacent airways should routinely be coated with stonedust (Section 21.8.3.1.).

21.2.2. Electrical apparatus

In addition to the general comments on mechanized equipment made in the preceding subsection, electrical gear can give rise to incendiary hazards from sparking and overheating. Switchgear or battery charging stations should be sited such that they are not affected by convergence or falls of roof. This is most liable to occur close to mineral winning areas. Furthermore, start switches should be protected against accidental operation by glancing blows from falling debris or passing traffic. Electrical sub-stations and battery charging chambers should be equipped with non-aqueous fire extinguishers.

Cables in airways should be hung in catenary fashion on cradles suspended from the roof. They should be located such that they will not be pinched by convergence or the yielding of roof supports, nor be impacted by vehicles. The insulation and type of sheathing must be suitable for the electrical load and rigours of the underground environment. All such cables should be inspected routinely for evidence of physical damage.

Electrical failures should result in immediate isolation of the power by means of overload and earth leakage protective devices. In gassy mines, all electrical motors and heavy current devices should be enclosed within flameproof casings so that any ignition of methane is contained within the equipment. Signalling or other light current apparatus should be certified as intrinsically safe, i.e. incapable of producing sparks of sufficient energy to ignite a methane:air mixture.

During non-working shifts, the electrical power supplied to each area of the mine should be isolated at the appropriate highest level control centre or substation. Precautions should be taken against power surges caused by lightning strikes on surface power lines, transformers, substations or any other lines that may conduct the surge to underground excavations. Similarly, particular care should be taken against electrical leakage in the vicinity of explosives or fuel storage areas.

21.2.3. Conveyors

Conveyor fires have been subjected to particular study because of the rapidity of fire propagation along the early rubber-based types of conveyor belting. Modern conveyor belting for underground use must be subjected to fire propagation tests (e.g. Verakis, 1991; Mutmansky et al, 2007). Three types of materials are used for mine conveyor belts, namely, styrene-butadiene rubber (SBR), neoprene (NP) and polyvinyl chloride (PVC). Composites of these materials are also employed. Following ignition of the belt material and removal of the igniting source, the fire should preferably fail to propagate or, if it does, move at a slow rate. However, it should be noted that heated belt material may produce hazardous fumes.

Numerous tests have indicated that fire propagation rates along conveyor belting are influenced by airspeed (e.g. Hwang et al, 1991). At a relative velocity of some 1.5 m/s between the belt surface and the
adjacent airstream, a phenomenon known as *flash over* attains its maximum effect. Flash over occurs when a flame front from the burning belt reaches forward over an unburned surface with an optimum angle and length such that the radiant effect on that surface reaches a maximum. This can cause flaming of the top layer of belting and a significant increase in flame propagation rate along the surface of the belt. Deeper layers in the weave of the material may or may not be ignited. The effect has been observed in belts of compositions including SBR, PVC, and SBR–neoprene combination (Verakis and Dalzell, 1988). Flashover involves a serious hazard as belt surface propagation rates may reach some 10 m/min. The spread of fire along mine conveyors is influenced strongly by the turbulence of the airflow. Hence, laboratory tests of small samples of belting can give misleading results. Large scale gallery tests are more reliable.

Conveyor fires are most likely to be initiated by friction. If the belt becomes staked (immobilized) at any point along its length and the drive rollers continue to turn, then high temperatures will be generated at the drive head. Temperature monitors or belt tension transducers can sense this condition. Such devices should be wired to isolate electrical power from the conveyor drive when an alarm condition is detected. Similarly, a seized idler or return roller can become red-hot from the friction of a belt moving over or around it. Conveyors should be patrolled regularly during operation in order to detect the development of faulty rollers. Worn bearings will often be noisy and may also be detected by the smell of heated surfaces. A further frictional hazard can occur if the conveyor becomes misaligned to the extent that the belt rubs against surrounding surfaces such as the conveyor structure or airway sides.

In all of these cases, a fire may be initiated when lubricants, coal dust or flammable debris reach their ignition points. It follows that dust or spillage should not be allowed to accumulate around and, particularly, underneath conveyors. A clean conveyor road is much more likely to be a safe one. In coal mines, conveyor entries should be well rock dusted

21.2.4. Other frictional ignitions

The main cause of methane ignitions on the working faces of coal mines is frictional sparking at the pick points of coal winning machinery. This occurs particularly when the machine cuts through sandstone or pyritic material. Two approaches have been taken to reduce this hazard. One is to ensure that there is sufficient ventilation around the cutting drum to provide rapid dilution of the methane as soon as it is emitted. It is, of course, important that the overall airflow at the working face is adequate to prevent methane layering (Section 12.4.2.) and that the layout of the system minimizes flushes of methane from worked-out areas (Section 4.3.2.). A number of devices have been employed to enhance air movement across the pick points of shearer and continuous miners (e.g. Browning, 1988). Unfortunately, these may exacerbate the dust problem unless combined with a wet scrubber (Section 20.4.2.2.).

The second approach to the incendiary streak of sparks that sometimes trails behind a cutter pick is to quench it with water. This technique combines the suppression of both dust and methane ignitions. It is achieved by pick face flushing and, even more efficiently, by jet assisted cutting (Section 20.4.1.1.).

Rope haulage systems have been the cause of some mine fires. Care should be taken that all pulleys and return wheels are routinely serviced and lubricated. Ropes should not be allowed to rub against solid surfaces such as the roof, sides or floor of airways and, particularly, timber supports. If haulage ropes must pass through holes in stoppings then, again, the ropes should not contact the sides of the orifices. Fluid couplings and enclosed gearings or direct drives are preferred to mechanical clutches, belts or V-drives for the transmissions of mining machinery. However, where the latter are employed then, again, regular inspections and maintenance are required to ensure their continued safe operation. Similarly, mechanical braking systems should be well looked after.
21.2.5. Explosives

The initiation of fires from explosives or igniter cord remains a danger in non-gassy mines. Incandescent particles from blasting operations may contain sufficient heat energy to ignite dry wood or combustible waste material. Igniter cord should never be hung on timber supports. A strict record should be maintained on all explosives and detonating devices at the times of issue and return to the stores. The relevant national or state legislation should be consulted for the conditions under which explosives may be stored or transported underground.

21.2.6. Welding

All welding operations that are permitted underground should be carried out under well-controlled conditions. Where there is any possibility of methane or other flammable gases being present then testing for those gases should be carried out before and, at intervals, during the welding operations. Hot slag and sparks from welding are easily capable of igniting combustible materials such as coal, wood, paper and waste rags. Wherever possible, such materials should be removed from the vicinity of welding operations and the remainder wetted down or coated by stonedust. Molten metal should not be allowed to drop on the floor. Slag pans should be used to capture hot run-off. This is particularly important in coal mines, in shafts and near timber supports. Fire extinguishers must be available at the sites of all welding operations.

Gas containers employed in oxy-acetylene cutting should be stored and used in a secure upright position. Gas bottles must never be stored or used in the vicinity of explosives or concentrations of flammable liquids.

21.2.7. Smoking and flame safety lamps

It is a sad fact that the use of smoking materials has been suspected as the cause of some fires and explosions in mines. In those mines that have been classified as gassy, carrying such materials (often known as contraband) into the subsurface is illegal. This law should be enforced with the utmost rigour. Through well chosen examples during training and refresher classes a workforce will, themselves, ensure compliance with non-smoking regulations.

In subsurface openings where smoking is permitted then, again, education, posters and warning signs should be employed as ongoing reminders of the possible disastrous consequences of careless disposal of smoking materials.

Damaged flame safety lamps have also been suspected of igniting a methane:air mixture. In the few places where these devices remain in use, they should be treated with care and subjected to inspection after each shift. When a high concentration of methane is detected by a blue flame spiralling rapidly within a flame safety lamp (Section 11.4.2.2.) then the lamp should be lowered gently and, if necessary, smothered inside one's clothing. Familiarity with the procedure should be gained through training and will counter the natural reaction of the untrained person to drop the lamp or to throw it away in panic. (Section 11.4.2.2)

21.3. OPEN FIRES

Fires that occur in mine airways usually commence from a single point of ignition. The initial fire is often quite small and, indeed, most fires are extinguished rapidly by prompt local action. Speed is of the essence. An energetic ignition that remains undetected, even for only a few minutes, can develop into a conflagration that becomes difficult or impossible to deal with. Sealing off the district or mine may then become inevitable.
The rate at which an open fire develops depends, initially, upon the heat produced from the igniting source. A fine spray of burning oil from a damaged air compressor can be like a flame thrower and ignite nearby combustibles within seconds. On the other hand, an earth leakage from a faulty cable may cause several hours of smouldering before flames appear. The further propagation of the fire depends upon the availability of fuel and oxygen (Figure 21.1). A machine fire in an untimbered metal mine airway will remain localized if there is little else to burn in the vicinity. Conversely, an airway that is heavily timbered or with coal surfaces in the roof, floor or sides will provide a ready path for speedy development and propagation of a fire.

When an open fire has developed to the extent of causing a measurable change in the temperature of the airflow then it can affect the magnitudes and distributions of flow within the mine ventilation system. Conversely, the availability of oxygen to the fire site controls the development of the fire. This Section discusses the coupled interaction between fire propagation and ventilation, and the means by which open fires in mines may be fought.

**21.3.1. Oxygen-rich and fuel-rich fires**

At the start of most open fires in ventilated areas, there is a plentiful supply of oxygen - more than sufficient for combustion of the burning material. Indeed, if the air velocity is brisk then heat may be removed at a rate greater than that at which it is produced. The heat side of the fire triangle is removed and the fire is "blown out". These are examples of oxygen-rich fires. Assuming that the fire continues to proliferate, it will consume increasing amounts of oxygen and, at the same time, produce greater volumes of distilled gases and vapours. The point may be reached when the heat of combustion produces temperatures that continue to remain high enough to distill gases and vapours from the coal, timber or other available fuels but with insufficient oxygen to burn those gases and vapours completely. The fire has then become fuel-rich.

The development of an oxygen-rich into a fuel-rich fire is a serious progression and produces a much more dangerous situation for firefighters. When flammable gases at temperatures exceeding their ignition point meet relatively fresh air then they will ignite along the gas:air interfaces. The added turbulence may produce intimate mixing of air and unburned gases resulting in explosions. These phenomena can occur downstream from an open fire if air leaks into the firepath from adjacent airways. Firefighters are then faced with a difficult decision. Leakage of air from adjacent airways must be into the firepath in order to prevent spread of the fire into those adjacent airways, yet the admittance of that air may cause explosions and propagation of the fire at a rate much greater than that allowed by burning of the solid material itself.

A similar effect occurs when buoyancy of the hot gases causes roll-back of smoke at roof level against the ventilating current (Section 21.3.2.2.). This can occur over the heads of workers who are fighting the fire from an upstream position. Again, burning of the gases along the air interface can occur, igniting coal or timber in the roof and producing the danger of explosion. Personnel involved in fighting a fuel-rich fire may become aware of pressure pulses or rapid fluctuations in the movement of the air. These are caused by rolling flames and "soft" explosions as gases ignite along gas:air mixing zones. The same phenomena can be observed following ignitions of methane (Section 1.2.). Such pulsations may be a precursor to a larger and more violent explosion.

It follows that every attempt should be made to prevent an oxygen-rich fire from developing into a fuel-rich fire. This underlines the need for early detection and prompt action. An intuitive reaction to a fire may be to restrict the air supply and, hence, remove the oxygen leg of the fire triangle. This can be accomplished by building stoppings or erecting brattice cloths upstream from an airway fire. However, consideration of the dangers inherent in fuel-rich fires indicates that restricting the airflow might be inadvisable. Analyses of gases downstream from fires can be interpreted to indicate whether a fire is oxygen-rich or fuel-rich (Section 21.7).
21.3.2. Effects of fires on ventilation

An open fire causes a sharp increase in the temperature of the air. The resulting expansion of the air produces two distinct effects. First the expansion attempts to take place in both directions along the airway. The tendency to expand against the prevailing direction produces a reduction in the airflow. This is known as the choke or throttle effect. Secondly, the decreased density results in the heated air becoming more buoyant causing local effects as well as changes in the magnitudes of natural ventilating energy.

21.3.2.1. The choke effect

Consider an airway before it is affected by a fire. Air flows along it at a mass flowrate of $M$ kg/s and doing work against friction at a rate of $F$ J/kg. The airpower dissipated against friction, $P_{ow}$, is the product of the two

$$P_{ow} = FM$$

or Watts

(21.1)

The effect of a fire in the airway upon $P_{ow}$ depends upon the reactions of fans, natural ventilating pressures and ventilation controls throughout the system. However, if no deliberate action is taken to change these factors, it is reasonable to estimate that it remains sensibly constant.

Equation (7.10) gave us

$$F = \frac{p}{\rho} = R_t Q^2$$

or J/kg

(21.2)

where

$p = \text{frictional pressure drop (Pa)}$

$\rho = \text{mean density of air (kg/m}^3\text{)}$

$R_t = \text{rational turbulent resistance of the airway (m}^{-4}\text{)}$

and

$Q = \text{mean value of airflow (m}^3\text{/s)}$

Combining equations (21.1) and 21.2) gives

$$P_{ow} = FM = R_t M Q^2$$

or

$$P_{ow} = R_t \frac{M^3}{\rho^2} \text{ W}$$

(21.3)

and

$$Q = \frac{M}{\rho}$$

Equation (21.3) may be written as

$$M = \left(\frac{P_{ow}}{R_t}\right)^{\frac{3}{2}} \rho^{\frac{2}{3}}$$

(21.4)

As $P_{ow}$ and $R_t$ are constants,

$$M \propto \rho^{\frac{2}{3}}$$

(21.5)

where $\propto$ means 'proportional to'

Hence, as the fire causes the density of the air to decrease, the mass flow of air will also decrease for the same energy dissipation. This phenomenon produces the choke effect. It should be noted, however, that the volume flow exiting the airway has increased.

As $M = \rho Q$, proportionality (21.5) can be written as
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\[ Q \propto \frac{1}{\rho^3} \quad \frac{m^3}{s} \]  

(21.6)

Note, also, from \( P_{ow} = F M = \text{constant} \), that as \( M \) decreases, the work done against friction per kg of air, \( F \), must increase - a result of the increased volume flow and, hence, turbulence.

The choke effect is analogous to increasing the resistance of the airway. For the purposes of ventilation network analyses based on a standard value of air density, the raised value of this "pseudo resistance", \( R_t' \), can be estimated in terms of the air temperature as follows:

From equation (21.3)

\[ R_t = P_{ow} \frac{\rho^2}{M^3} \quad \text{m}^4 \]

Hence, for any standard (fixed) value of density and constant air power loss,

\[ R_t' \propto \frac{1}{M^3} \]

But combining with proportionality (21.5) which represents the actual reduction in mass flow,

\[ R_t' \propto \frac{1}{\rho^2} \]

For a given barometric pressure, the general gas law (Section 3.3.1.) gives

\[ \rho \propto \frac{1}{T} \quad \text{where} \ T = \text{absolute temperature (K)} \]

Hence,

\[ R_t' \propto T^2 \]  

(21.7)

The value of the "pseudo-resistance" \( R_t' \), increases with the square of the absolute temperature. However, it should be recalled that this somewhat artificial device is required only to represent the choke effect in an incompressible flow analysis.

Litton et al (1987) have also produced an estimate of the increased resistance in terms of the carbon dioxide evolved from a fire.

21.3.2.2. The buoyancy (natural draft) effect

The most immediate effect of heat on the ventilating air stream is a very local one. The reduced density causes the mixture of hot air and products of combustion to rise and flow preferentially along the roof of the airway. The pronounced buoyancy effect causes smoke and hot gases to form a layer along the roof and, in a level or descentional airway, will back up against the direction of airflow. The layering effect can be estimated using the method given in Section 12.4.2.

This phenomenon of roll-back creates considerable difficulties for firefighters upstream from the fire, particularly if the conflagration has become fuel-rich. The roll-back is visually obvious because of the smoke. However, it is likely to contain hidden but high concentrations of carbon monoxide. Furthermore, the temperatures of the roll-back may initiate roof fires of any combustible material above the heads of firefighters. The most critical danger is that tidal flames or a local explosion may occur throughout the roll-back, engulfing firefighters in burning gases.
One method of reducing roll-back is to increase the airflow in the airway. This, however, will increase the rate of propagation of the fire. Another method is to advance with hurdle cloths covering the lower 60 to 80 per cent of the airway (Section 12.4.2.). The increased air velocity at roof level will help to control the roll-back and allow firefighters to approach closer to the fire. However, this technique may also cause the roll-back gases to mix with the air and produce an explosive mixture on the forward side of the hurdle cloth. Furthermore, the added resistance of the hurdle cloth might reduce the total airflow to the extent that a fuel-rich situation is promoted. The behaviour of open fires is very sensitive to modifications to the airflow. Hence, any such changes should be made slowly, in small increments, and the effects observed carefully.

A third method of combatting roll-back is to direct fog sprays towards the roof. In addition to wetting roof material, the air induction effects of the sprays will assist in promoting airflow in the correct direction at roof level.

A more widespread effect of reductions in air density is the influence they exert in shafts or inclined airways. This was handled in detail under the name of natural ventilation in Section 8.3.1. The effect is most pronounced when the fire itself is in the shaft or inclined airway, promoting airflow if the ventilation is ascentional and opposing the flow in descentional airways. Indeed, in the latter case, the flow may be reversed and can result in uncontrolled recirculation of toxic atmospheres.

If the air temperatures can be estimated for paths downstream of the fire then the methods given in Section 8.3.1. may be employed to determine the modified natural ventilating pressures. Those temperatures vary with respect to

- size and intensity of the fire
- distance from the fire
- time
- leakage of cool air into the airways affected and
- heat transfer characteristics between the air and the surrounding strata.

At any given time, air temperatures tend to fall exponentially with respect to distance downstream from a fire. Climatic simulation models (Chapter 16) may also be employed to track the time transient behaviour of air temperatures downstream from a fire. However, in that case, two matters should be checked. One is that the limits of application of the program may be exceeded for the high temperatures that are involved. Secondly, the transient heat flux between the air and strata will be much quicker than for normal climatic variations. Hence, the virgin rock temperature (VRT) in the simulation input should be replaced by a "surrounding rock temperature" (SRT), this being an estimate of the mean temperature of the immediate envelope of rock around the airway before the fire occurs.

Having determined air temperatures in all paths downstream from the fire, the revised natural ventilation pressures for the mine can be determined. These may then be utilized in network analysis exercises to predict the changes in flow and direction that will be caused by a fire of given thermal output. A number of fire simulation packages have been developed to allow numerical modelling of mine fires (e.g. Trutwin et al (1992), Greuer, (1984); Greuer, 1988; Dziruzynski et al, (1988); Deliac et al, (1985); Stefanov et al, (1984); Wala (1998); Gillies et al (1995). Gillies A.D.S., et al (2004).

21.3.3. Methods of fighting open fires

The majority of open fires can be extinguished quickly if prompt action is taken. This underlines the importance of fire detection systems, training, a well-designed firefighting system and the ready availability of fully operational firefighting equipment. Fire extinguishers of an appropriate type should be available on vehicles and on the upstream side of all zones of increased fire hazard. These include storage areas and fixed locations of equipment such as electrical or compressor stations and conveyor gearheads.
Neither water nor foam should be used where electricity is involved until it is certain that the power has been switched off. Fire extinguishers that employ carbon dioxide or dry powders are suitable for electrical fires or those involving flammable liquids.

**Deluge and sprinkler systems** can be very effective in areas of fixed equipment, stores and over conveyors. These should be activated by thermal sensors rather than smoke or gas detectors in order to ensure that they are operated only when open combustion occurs in the near vicinity.

The two direct methods of firefighting introduced in this Section involve the application of water and high expansion foam. The additional or complementary means of fire management by adjustment of ventilation controls and the injection of an inert gas are discussed in Sections 21.3.4. and 21.6. respectively.

21.3.3.1. Firefighting with water
Except where electricity or flammable liquids are involved, water is the most common medium of firefighting. When applied to a burning surface, water helps to remove two sides of the fire triangle. The latent heat of the water as it vapourises and the subsequent thermal capacity of the water vapour assist in removing heat from the burning material. Furthermore, the displacement of air by water vapour and the liquid coating on cooler surfaces help to isolate oxygen from the fire.

Water is normally applied by hosepipes upstream from the fire. A difficulty in subsurface firefighting is the limited reach of water jets imposed by the height of the airway. This underlines the vital need for water to be available at adequate pressure and quantity in the firefighting range. In order for a water jet to reach some 30 m in a typical coal mine entry, water pressures should be in the range 800 to 1400 kPa (Mitchell, 1990) and capable of supplying up to five hoses from a manifold connected to a single hydrant. In practice, the range of water jets in mine airways may often be no greater than 10 m. The nozzles should preferentially be of the adjustable type to give either a jet or a fog spray.

Hard won lessons indicate the need for careful forethought in designing a mine firefighting water network. The air and the water should flow in the same direction so that firefighters do not become dependent on a water supply that passes through the fire before it reaches them. Hydrants should be located at strategic points with respect to areas of increased fire hazard, at intervals along airways and at cross-cuts with access doors. All fittings for hydrants and range components should be standardized throughout any given mine. Non-metallic caps should be used at hydrant outlets to minimize corrosion. However, these caps must always be removable by hand and without undue force. Supplies at firefighting stations should be inspected at set intervals to ensure their operational efficiency at all times. Range fittings should include tee-pieces, blank-off caps and manifolds. It is particularly important that hosepipes be unrolled and examined for deterioration on a planned maintenance schedule and that they should be stored according to manufacturers’ recommendations.

If access can be gained to an airway that runs parallel to a fire then fog sprays can be directed through doors or holed stoppings into the path of the fire. This can be effective if the sprays are employed at an early stage and immediately downstream from the fire front. (see Fig 21.2). However, for a large conflagration or where the fire has become fuel-rich, it is likely to lose its effectiveness.

The locations of pumps and configuration of their power supplies should be considered carefully with respect to the layout of the mine. The pumps and routes of their cables should be chosen such that they are least likely to be disrupted by a fire. Dual power supplies via alternative routes may be considered. Furthermore, power for firefighting pumps should be capable of being maintained when electricity to working sections of the mine has to be isolated. Underground sumps can provide valuable water capacity. However, the firefighting system should also allow water to be supplied in adequate quantities from surface locations.
21.3.3.2. High expansion foam

Large volumes of water-based foam provide a valuable tool for fighting fires in enclosed spaces such as the basements of buildings or in the holds of ships. It has been employed for mine fires since at least 1956 (Eisner). The method is employed on large fires and, although it has had somewhat limited success in extinguishing mine fires, it can play a valuable role in cooling and quenching an area to an extent that allows firefighters with hoses to approach closer to the firefront. Even when sealing an area has become inevitable, valuable time for rescue operations can be bought by employing high expansion foam.

The bubbles are generated by a fan which blows air through a fabric net stretched across a diffuser. The net is sprayed continuously with a mixture of water and foaming agent. Bubbles can be produced at a rate of several cubic metres per second (Strang and MacKenzie-Wood, 1985). Compounds such as ammonium lauryl sulphate may be employed as the foaming agent while the addition of carboxymethylcellulose improves the stability of the bubbles (Grieg et al, 1975).

The objective is to form a plug of high expansion foam which fills the airway and is advanced on to the fire by the ventilating pressure. The ratio of air to water within the foam may be in the range 100:1 to 1000:1. As the foam advances, bubbles break around the perimeter of the airway when they touch a dry surface. However, the liquid that is released wets that surface and allows advancement of the following bubbles. Shrinkage of the foam occurs continuously at the leading edges and accelerates because of radiant effects as it approaches the burning material.

Control of the combustion process is achieved by two primary mechanisms. First, vapourization of the water removes heat from the site and, secondly, the increased concentration of water vapour may produce an extinguishing atmosphere. As the air within the bubbles is heated to 100°C it will expand by some 30 per cent. However, the vapourization of liquid water to a gas involves an expansion of about 1700:1. Assuming an air:water mix in the foam of 1000:1 a thousand litres of air expands to 1300 litres while 1 litre of water evaporates to become 1700 litres of water vapour giving 3000 litres of mixture. If the air originally had an oxygen content of 21 per cent then the evaporation of water will reduce that to

\[
\frac{21 \times \frac{1300}{3000}}{1} = 9.1 \text{ percent which will extinguish flaming combustion.}
\]

Despite these mechanisms, high expansion foam does have some drawbacks. First, it may be quite difficult to generate a foam plug that fills the airway completely. As the plug builds up, the air velocity will increase through the narrowing channel between the plug and the roof, tending to maintain the gap open. Judicious employment of brattice cloths may assist in forming a complete plug of foam. It is important to control the path of the foam and, in multi-entry systems, this can be problematic. The natural direction of movement of the foam is dictated by the ventilating pressure. Here again, brattice cloths or stoppings in cross-cuts to adjacent parallel entries can assist in controlling the direction of the foam. Major obstructions caused by roof falls are quite liable to occur during a large underground fire. A foam plug may not be able to climb over such obstructions with the ventilating pressure available.

The greatest danger of foam plugs is that the reduction in airflow may promote a fuel-rich fire with the attendant danger of explosion. Downstream gases should be monitored for the development of this condition. Both increases and decreases in combustible gases have been reported in differing fires when high expansion foam has been employed. The reduction in airflow will tend to raise the concentration of combustible gases. However, as the inert mixture of air and water progresses downstream, condensation of the water occurs, allowing the air fraction to increase and, hence, modifying the combustible gas concentrations.

After the application of high expansion foam has been initiated, it is important to maintain it in operation during fire fighting as intermittent production of foam can exacerbate the development of an explosive atmosphere. This underlines the need for good training so that operators are familiar with the equipment and procedure. Furthermore, care should be taken that sufficient supplies of foaming agent are available before the operation is started (Timko et al, 1988).
21.3.4. Control by ventilation

When contemplating changes to airflows and applied pressure differentials during a fire emergency, there are four types of effect that must be considered most carefully.

(a) The effect on the combustion process: The importance of avoiding the progression of an oxygen-rich fire into a fuel-rich fire has already been stressed in Section 21.3.1.

(b) The effect on direction and rate of propagation of the fire: Every attempt should normally be made to prevent an open fire from spreading into other airways. However, exceptions from this general rule may become necessary to guide products of combustion away from trapped personnel. An example may be the deliberate destruction of a stopping or air crossing to divert or short-circuit a fire path from an intake airway into an adjacent return. Again, any modifications of the airflow passing through the fire zone must seek to achieve a balance between speed of propagation and control of the combustion process.

(c) Effects on the distribution of products of combustion: This becomes a critical issue when personnel have become trapped in the fire, particularly if their exact whereabouts are unknown. However, any steps that will improve atmospheric conditions in escapeways require to be investigated.

(d) Effects on airflow distributions in other parts of the mine: While the consequences of ventilation changes in the zone affected by the fire are of immediate concern, the effects of such changes throughout the rest of the mine should not be overlooked, particularly in a gassy mine or when personnel may still be evacuating other areas.

If a computer model of the mine ventilation network has been maintained up to date then this will prove invaluable in investigating the predicted effects of proposed changes to the ventilation system. With a modern network analysis package (Section 7.4.), a personal computer or terminal in the emergency control centre can produce such predictions within seconds. Nevertheless, the uncertainties inherent in a fire situation demand that actual changes to the airflow system be made incrementally while observing the reactions on distributions and gas concentrations. The following subsections discuss the practical strategies that may be employed to control a fire by ventilation.

21.3.4.1. Pressure control

Airways that are parallel and adjacent to the fire path will remain unpolluted provided they are maintained at a higher atmospheric pressure. These allow access for escape, firefighting, building or strengthening of stoppings in cross-cuts, or to apply water sprays into the fire path. In multi-entry workings, control of such pressure differentials can be achieved by the erection of brattice cloths in the adjacent airway as illustrated in Figure 21.2. Even if the pressure differential in the desired direction is not completely achieved, the reduced rate of toxic leakage may allow time for personnel to escape. If necessary, the brattice cloths may be advanced pillar by pillar to remove smoke sequentially from the adjacent airway. Devices such as the "parachute stopping" or "inflatable seal" have been developed to replace brattice cloths in such circumstances. These can be erected quickly and give improved seals around the perimeter of the airway (Kissell and Timko, 1991).

A consequence of this technique is that the airflow over the fire will be increased to an extent that depends upon the configuration and resistances of the local airways. Pressure differentials between airways can also be modified by the use of a temporary fan instead of a restriction in the adjacent airway. In this case, airflow over the fire will be reduced. The location and pressure developed by the fan must be selected with care in order to avoid recirculation of products of combustion. Where pressure differentials are small, even the few Pascals developed by a free-standing auxiliary fan can induce the desired effect (Section 4.4.3.).
21.3.4.2. Airflow reversal

Many mines operate under a legislative requirement that the airflow provided by main fans must be capable of being reversed promptly. The background to such laws is the fear of a fire or other inundation of airborne pollutants occurring within a downcast shaft or main intake airway. Noxious and, possibly, flammable gases could then contaminate all, or most, of the ventilation system including working areas and return escapeways. If the fire is detected at an early stage then pollution of the complete system may be prevented by prompt reversal of the airflow. Even where contamination of the total network has occurred, air reversal may allow clearance of pollutants from return airways to the extent that a fresh air route may be established between surface and refuge chambers where personnel may be trapped. The decision to reverse a complete mine ventilation system is fraught with difficulties and has very seldom been taken in practice.

There are essentially three methods of achieving the reversal of airflow in a ventilation system. Where axial impellers are used on the main fans, then changing the direction of rotation will cause reversal of the airflow. This can be implemented electrically at the fan motor. However, axial fans operate efficiently in one direction only. The aerofoil section of each impeller blade is designed to give aerodynamic stability of flow through the fan. When operating in reverse, breakaway of the boundary layers over the blades occurs, resulting in high shock losses. The “lift” of the blades and, hence, the throughflow of air is greatly reduced (Section 10.3.2.). Similarly, fixed guide vanes, fan casings and evasees, all designed for a forward direction, will produce high shock losses when the airflow is reversed. The reversed air quantity may be reduced to less than 50 per cent of the normal forward flow (Dunn, 1982).

In the case of centrifugal fans, airflow reversal can be achieved only by means of reversal doors. The flow direction through the fan itself remains unchanged. For an exhausting centrifugal fan located at the mine surface, hydraulic or pneumatic activation of the reversal doors opens the fan inlet to the outside atmosphere and, simultaneously, diverts the fan exhaust into the mine shaft or slope. The opposite occurs for a forcing centrifugal fan. Where reversal doors are fitted as part of a surface fan installation, their operation should be checked routinely as part of a planned maintenance procedure.

Although the flow direction through a centrifugal fan remains unchanged, the shock losses incurred when air reversal doors are activated results in a reduced flow. The amount of the reduction is site specific and depends entirely upon the design and siting of the reversal doors, and the configuration of the fan with respect to the inlet and outlet duct arrangements. The possibility of requiring air reversal should be considered when designing the layout of airways and ventilation doors around an underground main fan. Such reversal should be attainable rapidly by opening or closing those doors.

During the course of ventilation network planning exercises (Chapter 9), it is often possible to design systems that allow rapid reversal of airflow in one section of the mine, or in a single airway, without total reversal at the main fans. This may be achieved by the strategic location of doors that can be opened or closed to permit airflow in either direction. Means of such local reversal might be considered, for example, for a conveyor route that is also to serve as an intake during normal operations.
Mandating the provision of air reversal facilities for an underground mine appears to be a reasonable safeguard. However, except in a clear-cut case such as a fire in or very close to a downcast shaft, the potential risks associated with reversing the airflow may be greater than those of maintaining the normal direction of flow, particularly in the short time period that may be available for making critical decisions.

The reasons that mine managements have very rarely decided to reverse ventilation during an emergency are both practical and also because of possible litigation should lives be lost as a consequence of the reversal. During the trauma of a major emergency involving changing conditions in air quality and possible disruptions of ventilation structures and communications, it may be impossible to know with certainty the locations, movements and dispersal of the workforce. Reversal of the airflow could then result in smoke and toxic gases being drawn over personnel who had assembled in a previously unpolluted zone. It may be expected that people who work routinely in a section of the mine will be familiar with the local ventilation system and, in case of an emergency, will act in accordance with that knowledge. Reversing the airflow could create additional uncertainty and confusion in their actions.

The majority of doors in airlocks or access paths between intakes and returns are designed to be self closing, assisted by the mine ventilating pressure. This may be a legislative requirement. In the event of airflow reversal, those doors will be blown open and create short circuits unless they are provided with self-locking devices. Even in the latter situation or where powered doors are employed, the proportion of air leakage must be expected to increase when the pressure differential across the door reverses. Hence, coupled with the diminution in overall flow caused by the reversal procedures, the reversed ventilation reaching the working areas must be expected to be much lower than the normal forward flow.

High temperatures usually preclude firefighting rescue teams from approaching a fire from the downstream side. If the airflow is reversed over a fire in an intake then firefighting teams must transport their equipment and materials to a fresh air base inby the fire.

The expansion of gases held in old workings or other voidage, and resulting from a drop in barometric pressure, is discussed in Section 4.2.2. In the case of a forcing fan being reversed to create an exhausting system, the rapid fall in barometric pressure throughout the system may cause large emissions of voidage gas. If these contain high concentrations of methane, passing it into the fire zone could result in a series of explosions propagating far back into the mine. Another possibility is that during the actual process of reversal, flammable gases from the strata or produced by incomplete combustion or volatilization of hydrocarbons may be drawn back over the fire, again, leading to the possibility of explosions.

Although there may be several hundred tonnes of moving air in a major subsurface structure, the braking effects of viscous shear and turbulence causes it to be a well-damped system. Hence, when a main fan stops, the effect is noticeable almost immediately at all places underground. In the majority of cases, natural (thermal) ventilating effects will maintain movement in the normal direction.

In the situation of forced reversal, the transient effects will exist for much longer time periods than for a simple stoppage of a main fan. This is because redistributions of the natural ventilating effects will not be completed until a new equilibrium of heat transfer has been established between the strata and the airflow. This may take several hours or even days. Furthermore, the fire itself will create thermally induced airflows with, perhaps, local reversals and recirculations. There is, therefore, some uncertainty concerning the speed at which reversal can be attained throughout the system and the stability of the reversed airflows. Research in Poland has indicated the efficacy of accelerating reversal by the use of intensive water sprays directed into the top of an upcast shaft (Trutwin, 1975).
21.4. SPONTANEOUS COMBUSTION

When air is allowed to percolate through many organic materials including coal then there will be a measurable rise in temperature. The same phenomenon can be observed in crushed sulphide ores and is caused by a progressive series of adsorptive, absorptive and chemical processes. These produce heat and an observable elevation in temperature. The percolating airflow will, therefore, remove that heat increasingly as the temperature of the material rises. If the leakage airflow is sufficiently high then a balanced equilibrium will be reached at which the rate of heat removal is equal to the rate at which heat is produced; the temperature will stabilize. The process will also reach an air-constrained equilibrium if the airflow is sufficiently low to inhibit the oxidation processes. However, between these two limits there is a dangerous range of percolating airflows that will encourage spontaneous heating.

Each material that is liable to spontaneous combustion has a critical temperature known as the minimum self-heating temperature (SHT). This is the lowest temperature that will produce a sustained exothermic reaction or thermal runaway. Hence, if the temperature reaches the SHT before thermal equilibrium is attained then the oxidation process will accelerate. The temperature will escalate rapidly, encouraging even higher rates of oxidation until the material becomes incandescent. At this stage, smoke and gaseous products of combustion appear in the subsurface ventilation system. The mine then has a concealed fire.

The primary dangers of such occurrences are the evolution of carbon monoxide, the ignition of methane and combustion progressing into airways to produce open fires.

The phenomenon of spontaneous combustion has been recognized since at least the seventeenth century (PD-NCB, 1978). An early theory postulated that oxidation of pyritic material within coal provided centres of enhanced activity. Bacterial action is a factor in the initial natural heating of hay and other foodstocks. This can play a part in the spontaneous combustion of timber or organic waste material underground but is unlikely to contribute significantly to the self-heating of coal or other minerals. Similarly, while increases in the temperature of materials can be observed in ore passes (Section 15.3.6.), the gravitational energy of collapsing waste areas in mines produces temperature rises that are insufficient to promote spontaneous combustion. Current concepts of the initiation of self-heatings are discussed in the following subsection.

21.4.1. The mechanisms of spontaneous combustion in minerals

Although spontaneous combustion can occur in crushed or caved sulphide minerals and in heavily timbered areas within metal mines, the problem is most common in coal mines. Research in this area has concentrated on the spontaneous combustion of coals.

The development of self-heating requires a large surface area of crushed material combined with a slow migration of air through that material. Hence, the problem arises in goaf (gob) areas, caved zones, crushed pillar edges, fractured coal bands in roof or floor strata, stockpiles and tips on surface, and within abandoned sections of mines. The progressive stages of spontaneous combustion appear to be complex and not yet fully understood. Here, we shall examine the effects of oxygen, elapsed time and water.

21.4.1.1. The phases of oxidation

The oxidation processes of coal occur in four stages (Banerjee, 1985).

(a) **Physical adsorption** (Section 12.2.1.) of oxygen on coal commences at a temperature of about -80°C and is reversible but diminishes rapidly as the temperature increases to become negligible beyond 30 to 50°C. The process of adsorption produces heat as a by-product of the modified surface energy of the material. This causes the initial rise in temperature.

(b) **Chemical absorption** (known also as chemisorption or activated sorption) becomes significant at about 5°C. This progressively causes the formation of unstable compounds of hydrocarbons and oxygen known as peroxy-complexes.
(c) At a temperature which appears to approximate the self-heating temperature (SHT) of the coal, the peroxy-complexes decompose at an accelerating rate to provide additional oxygen for the further stages of oxidation. This occurs within the range of approximately 50 to 120°C with a typical value of 70°C. At higher temperatures, the peroxy-complexes decompose at a greater rate than they are formed (Chakravorty, 1960) and the gaseous products of chemical reaction appear - in particular, carbon monoxide, carbon dioxide, water vapour, and the oxalic acids, aromatic acids and unsaturated hydrocarbons that give the characteristic odour of "gobstink" (Section 21.4.4.).

(d) When the temperature exceeds some 150°C, the combustion process accelerates rapidly. Incineration of the coal occurs with escalating emissions of the gaseous products of combustion.

The rate at which oxygen is consumed varies both with time and the phase of oxidation as illustrated in Figure 21.3. Oxygen is taken up rapidly in the earlier stages of chemisorption and as the pyroxy-complexes are formed. However, this reduces with time and as the surfaces become coated (weathered) with those oxygen compounds. The temperature curve tends to level off and may reach equilibrium. However, if the SHT is reached, as illustrated in Figure 21.3, then both the rate of oxygen consumption and the temperature escalate rapidly.

21.4.1.2. The effects of water vapour
There are two processes involving water that act in opposite directions. First, the moisture content of the coal is driven off by evaporation during the early stages of heating. Hence, some of the heat is removed in the water vapour as latent heat of evaporation, tending to inhibit the temperature rise of the coal. The second process involves adsorption of water vapour from the air by the coal (Hodges and Hinsley, 1964). The heat of adsorption (sometimes called heat of wetting) produces an increase in the temperature of the material. It follows that the net effect depends on which of the two processes is dominant. Coal which is completely saturated with water is unlikely to increase in temperature due to heat of wetting. However, a dry crushed coal that is infiltrated by moist air can exhibit an initial rapid rise in temperature from this phenomenon.

Figure 21.3 Examples of changes in oxidation rate and temperature with respect to time (developed from Muzyczuk, ref. Banerjee, 1985). The time scale is omitted as this can vary from a few hours to many days depending upon the type and fineness of the coal, and the flowrate and psychrometric condition of the air.
Adsorption of water vapour adds considerably to the early stages of spontaneous heating of coal. It is significant that coal mines in dry climates tend to be less troubled by spontaneous combustion. However, it has been observed that re-ignition of coal frequently occurs on re-opening a mining area that has been flooded to extinguish a fire. This is thought to occur for two reasons. First, the flooding and subsequent drainage may produce further disintegration of the coal and the creation of new surfaces; and, secondly, it is probable that many points of higher elevation or entrapped gases may not be fully flooded and remain at a sufficiently high temperature (hot spots) to inhibit adsorption. When cooler air is admitted subsequently it will pick up water vapour as it progresses through the wet conditions. An initial short-lived cooling of the hot spots may be followed by rapid adsorption of water vapour and a renewed escalation of temperature.

21.4.1.3. The path of a spontaneous heating.
The processes of oxidation and adsorption do not occur uniformly throughout a mass of crushed combustible material. The rate and directions of air migration and the air:surface contact area depend upon the geometry of the zone, compaction from overlying strata and the fineness of the crushed material. Hence, the rapid escalation of temperature that characterizes the development of a concealed fire occurs first at discrete foci or "hot spots". The synergistic effects of iron pyrites are now thought to be caused by differential rates of expansion during the early stages of heating. The coal around the pyrites becomes more finely crushed and produces additional area for oxidation. Furthermore, the pyrite itself becomes oxidized, adding to the escalation in temperature. Green crystals of ferrous sulphate are formed which oxidize further to the yellow hydrated ferric oxide, a characteristic feature of zones in coal mines that have been involved in spontaneous combustion.

The migration of the heating again depends upon the rate and direction of air leakage. However, in contrast to open fires, the tendency is for a spontaneous heating to propagate through crushed material against the airflow, i.e. towards the intake airways.

21.4.2. Susceptibility to spontaneous combustion

A large number of tests and indices have been devised as suggested measures of the liability of differing coals and other materials to spontaneous combustion. These have involved:

- coal petrology and rank, the younger and lower ranks of coal being more susceptible
- rates of oxygen consumption or temperature rise at specified phases of the oxidation process
- self-heating temperatures (SHT) or other temperatures at specified stages of the heating process
- rates of heat production during isothermal or adiabatic tests.

No such test or index has been found to have universal application. The difficulty is that susceptibility to spontaneous combustion depends not only upon the material but also its physical state as well as the psychrometric condition and migration paths of the leakage airflows, the latter depending, in turn, upon the mining methods and layout. The matters of additional relevance are:

- amount and degree of comminution of crushed material left in the goaf (gob) area, these depending upon:
  - friability of coal
  - type of coal winning machine
  - efficiency of coal clearance (removal)
  - roof, floor or pillar coal left (percentage extraction)
  - thickness of seams and need for multi-lift or caving mining systems
  - depth and pre-stressing (microfracturing) of coal
  - geological disturbances
methods of stowing and sealing of roadsides
gradient of the seam and proximity of other seams
length of face
rate of face advance or retreat
roof/floor stability and strata stresses in the vicinity of pillar edges, stoppings, air crossings and ventilation doors
the air pressure differential across the affected area
the layout and resistances of surrounding airways and faces, including obstructions
degree of consolidation and, hence, resistance of the caved areas
the relative moisture contents of the coal and air
the reduction of oxygen content in the goaf (gob) areas by the emission of methane or other gases.

With this number and variety of factors it is not surprising that efforts to characterize the susceptibility of coals to spontaneous combustion purely on the basis of laboratory tests have met with rather limited success. An improved approach involves allocating weighted credit to each of the factors listed in an attempt to develop a site-specific indication of liability to spontaneous combustion (e.g. Banerjee, 1985; PD-NCB, 1978).

21.4.3. Precautions against spontaneous combustion

As with most potential hazards in the subsurface environment, precautionary measures against spontaneous commence at the time of planning and design of the mine. Core samples of the seam or ore should be subjected to susceptibility tests discussed in the previous section. The layout of the ventilation network should be designed to minimize pressure differentials between adjoining airways and across caved areas. This might be arranged, for example, by favouring through-flow rather than U-tube arrangements (Section 4.3.1.). While design airflows must be sufficient to deal with gases or other airborne pollutants, consideration should be given to means of reducing those airflows such as methane drainage. Booster fans, where allowed by law, provide a powerful means of air pressure management and, coupled with the techniques of network analysis to investigate locations and fan duties, are most valuable in reducing incidences of spontaneous combustion. Branch resistances in the surrounding ventilation network should be kept as low as practicable by means of larger cross-sections or driving parallel entries. Furthermore, obstructions in those airways should be avoided.

The probability of spontaneous combustion can be reduced by minimizing the amount of coal, timber, paper, oily rags or other combustible materials that are left in gob areas. This may be inevitable if top coal must be left for the purposes of roof control. Nevertheless, efficient clearance of the fragmented coal from the face and good housekeeping should be practiced in mines that have a history of spontaneous combustion.

It is important for the mine ventilation engineer to be conscious of the zones in which spontaneous combustion is most likely to occur. Recalling that some leakage takes place through the strata around stoppings and doors, spontaneous heating may occur in coal which exists within that strata, whether it be in the roof, floor or sides. Good strata control and the liberal application of roadway sealants can help in such circumstances.

Pillars in coal mines should be designed large enough to minimize crushing at the edges and corners. Side bolts can help to maintain the integrity of pillars while the injection of low viscosity grouts might be used as a last resort. Here again, the application of surface sealants assists in preventing ingress of air. However, the most difficult types of spontaneous fires occur within caved zones and, in particular, in the goaf (gob) areas of coal mines.
Figure 21.4 illustrates the air migration paths and the zones most liable to spontaneous heating in the caved area of an advancing longwall. Those critical zones occur where the leakage airflows lie within that range which provides sufficient oxygen to promote continued oxidation of combustible material, but not enough to remove heat at the rate at which it is generated. There are two distinct zones. One of these lies along the original starting line of the face where incomplete consolidation allows a leakage path between intake and return. Within the central portion of the gob, consolidation allows little leakage. However, another critical zone occurs between the fully consolidated central core and the advancing face. This zone is not stationary but advances with the face. Recalling the time factor involved in the development of a spontaneous heating (Figure 21.3), it is clear that if the face is advanced continuously and at a sufficient rate then any potential heating may be "buried" by consolidation of the cave before it has had time to develop into a spontaneous fire. The most hazardous times occur at weekends and over holiday periods when additional precautions may be required. These include the application of roadway sealants, grout injections into roadside packs, and lowering the pressure differential across the gob by pressure balance techniques (Section 21.4.5.4.) In severe cases, injection of an inert gas into the gob may be considered (Section 21.6).

Figure 21.5 gives a similar illustration for a back-bleeder retreating longwall where the gob immediately behind the face is ventilated deliberately in order to prevent flushes of methane on to the faceline. Here again, the face start-up line and a moving zone trailing behind the face provide the critical zones most liable to spontaneous heatings. Despite the fact that the caved area is ventilated, less incidences of spontaneous heatings have been reported using this system than for advancing longwalls. As advancing or retreating systems tend to be favouried in separate geographical regions with differing coal seams and climates, there may be many reasons for this, as discussed in Section 21.4.2. Nevertheless, there are at least two features that may tend to mitigate against spontaneous heatings in back-bleeder layouts. First, there is likely to be a smaller pressure differential applied laterally across the gob area. This is particularly the case at the location of the starting line. Secondly, the fact that the area behind the face is actively ventilated may cause the critical zone between the ventilated and consolidated areas to be narrower and more quickly buried by the caving roof strata.

In mines with a history of spontaneous combustion, it is necessary to seal all abandoned workings. This is particularly important when those areas are adjacent to current workings and even more so when they exist within overlying strata. Spontaneous fires have resulted from undetected leakage airflows between current gob areas and workings that had existed in higher seams many years previously.

Following the completion and withdrawal of equipment from a section of a mine liable to spontaneous combustion, all entries into the section should be sealed and the atmospheric pressures applied to those seals balanced as far as it is practicable to do so. This may be accomplished simply by a re-arrangement of doors or by the dismantling of stoppings and air crossings. Active pressure balancing techniques may be employed as discussed in Section 21.5.5. The sites of seals should be prepared at strategic control points during the development of a section and should involve channels being excavated into the sides, and providing a nearby stock of building materials. This will facilitate sealing the district rapidly in the event of an uncontrollable fire.
Critical zones.

Sufficient oxygen to promote oxidation but not enough to remove heat at the rate at which it is produced. The self-heating temperature may be reached.

Consolidation of material prevents enough oxygen to cause combustion that can reach self-sustaining temperature.

Sufficient air to remove heat as it is produced.

**Figure 21.4** Zones showing liability to spontaneous combustion in the caved area behind a single entry advancing longwall face.

**Figure 21.5** Zones showing liability to spontaneous combustion in a multiple entry retreating longwall with back bleeders.
21.4.4. Detection of a spontaneous heating

There are essentially three classes of detection for incipient or active spontaneous heatings. The oldest and, many would argue, still the most reliable is through the human senses. The aromatics and unsaturated hydrocarbon gases that are produced during the oxidation phases (Section 21.4.1.) give rise to an odour known colloquially as gobstink. This has been described variously as like "petroleum, the oil used in a flame-safety lamp and sterilizing liquid." Although, as with all odours, it can be described only by analogy it is, nevertheless, very distinctive and unlikely to be forgotten. Figure 21.6 illustrates the increasing strength of the smell with respect to gas concentrations and temperature of the oxidizing coal.

The odour is likely to be the first indication of a heating that can be detected by the human senses. The output of water vapour may also be observed either as a haze in the air or from beads of condensation on steel supports or other surfaces in locations where such condensation is unusual. At a later stage, the matter is put beyond doubt by the appearance of smoke.

The second class of detectors are thermal devices used to determine increases in temperature. Infra-red scans of roadway sides have been employed to identify the emission points of warm gases into airways and are useful for localized heatings in pillars or around stoppings (Chakravorty and Woolf, 1980). They are less useful as a permanent and continuous means of warning. Thermocouples or thermistors have been left in gob areas. However, they too have met with little success to the present time. First, their wiring is unlikely to withstand the mechanical stresses of an active caving zone, even when sheathed. Secondly, the thermal conductivity of crushed rock is low. Hence, the temperature even within a metre of an active centre of heating may indicate no abnormal condition.

The most widespread method of detecting the onset and development of a spontaneous heating is by monitoring gas concentrations in return airways (ref. Section 11.4.3). The gases that are evolved are indicated on Figure 21.6. The interpretation of the relative values and trends of these concentrations is discussed in detail in Section 21.7.

Figure 21.6  Example of the development of odour and gases from a medium-volatile bituminous coal. The relative positions of the gas curves will remain the same for other coals but the actual concentrations vary with the rank of coal and magnitude of leakage airflows.
21.4.5. Dealing with a spontaneous heating

There is a procedure that should be followed when detection systems indicate that an active spontaneous heating is developing in a mine. First, a gas monitoring station should be set up downstream from the affected area and air samples taken at intervals of not more than 30 minutes. If an air monitoring system already exists in the mine then even greater detail of gas concentration trends can be followed. Personnel should be evacuated from all return airways affected and, if the condition is developing rapidly, also from the rest of the mine except for those involved in dealing with the situation.

Simultaneously, steps should be undertaken to identify the location of the fire. This may be obvious if smoke appears from discrete places in a close vicinity such as from a fire in leakage paths around a stopping or in the crushed corner of a pillar. The location of the fire is more difficult to detect if it occurs in a caved zone or inaccessible old workings. Whether or not smoke is present, it is useful to conduct a carbon monoxide survey. Measurements can be made by hand-held instruments or by stain tubes (Section 11.4.2.). Indicating the results on a mine map can assist in selecting the most probable sites of the fire.

Having located the fire, the next step is to decide how to control and, if possible, extinguish it. A variety of methods exist for this purpose. The injection of an inert gas as a practical and powerful method of dealing with both open and concealed fires in mines was developed through the 1980's and is discussed separately in Section 21.6. Other techniques of fighting concealed fires are introduced in the following subsections.

21.4.5.1. Excavating the fire

If the fire is known to be within a few metres of an airway then it may be possible to dig out, cool and remove the incandescent material. This can be the case for fires in pillars, around stoppings or air crossings, or for some gob fires. It may be necessary to drill holes into the zone in order to more accurately determine the site and extent of the fire. The excavation should commence from the upwind side of the fire to minimize the exposure of personnel to smoke and carbon monoxide. The airway should be wetted and/or coated in stonedust for a distance of some 10 m on either side of the excavation. Care should be taken to ensure control of the roof which may have become weakened by heat. Readings of methane and carbon monoxide concentrations should be taken frequently downstream from the site. As the fire is approached, it may be necessary to spray water on the workers to cool them. When the seat of the heating is exposed, it should be cooled by water jets applied around the periphery. Spraying water into the heart of a glowing carboniferous mass can result in the formation of water gas (Section 11.2.6.). The hot material should be loaded into metal conveyances and dampened thoroughly before transporting it out of the mine. When all of the incinerated material has been removed, the void should be cooled and, after some 24 hours to ensure no re-ignition, filled with an inert material such as limestone dust or gypsum-based wet fillers.

If it is impracticable to remove the fire physically then it becomes necessary to prevent ingress of air to the fire location. This leads us into the remaining techniques of dealing with a concealed fire.

21.4.5.2. Burying the fire

In some cases, it is possible to prevent or reduce access of air to the fire location by burying it under collapsed roof strata. Localized leakage through a roadside pack can be reduced by bringing the roof down in the airway, employing shotfiring if necessary, removing only a portion of the debris and compacting the remainder over the area to be sealed. The excavated roof must, of course, be well supported leaving the airway with an anticline.

If the heating occurs in the critical moving zone behind a longwall face (Figures 21.4 and 21.5) and is detected sufficiently early, then it is sometimes possible to bury it under consolidated caved material by a temporary increase in the rate of face advance (or retreat). This is likely to be successful only if the incipient heating has been detected by an early warning gas detection system and well before smoke appears.
21.4.5.3. Sealants
A variety of sealants have been employed on stoppings, airway surfaces, roadside packs and pillars in order to increase their resistance and, hence, reduce the access of leakage air. These may be applied on external surfaces or injected as grouts into the strata or packed material. While sealants that include resins or gels produce the lowest permeabilities and allow a degree of flexibility, the choice must often be made quickly and on the basis of local availability. Concrete and gypsum plasters can be sprayed quickly and effectively on to airway surfaces, as well as being useful for grouting and seal infills between stoppings. Water based slurries using mill tailings or other waste material have also been employed as grouts or for injecting into fire zones. These may be applied through boreholes drilled either from an underground location or from the surface. The injection of sodium silicate into coal pillars has been found to be effective (Banerjee, 1985).

While all known leakage paths connecting to the fire zone through roadsides should be sealed, it appears to be particularly advantageous to seal on the inlet or high pressure side. It is, however, often difficult to locate the relevant points of inward leakage on the inlet side. One way of doing this, where allowed by law, is to employ the techniques of pressure management (ref. following subsection) to intentionally and temporarily reverse the direction of leakage across the fire area. The appearance of smoke or elevated concentrations of carbon monoxide in the intake will identify the normal inlet points after which the flow reversal should be terminated or, better still, the leakage reduced to zero. This method should be employed with caution and only when adequate gas monitoring facilities are available in order to avoid unknown recirculation. The approval of governmental agencies should be sought if necessary.

21.4.5.4. Localized pressure balancing
If no differential pressure exists across a level permeable zone then there can be no air leakage through it. Pressure balancing involves raising the pressure on the return side or decreasing the pressure in the intake until the leakage flow is reduced to near zero. This principle can be applied to complete sections of abandoned workings as described in Section 21.5.5. or, in a more localized manner, to gain control of a gob fire without sealing the district.

Figure 21.7 shows the principle of this technique which can be applied to a variety of situations. In the example illustrated, a fire has commenced along the starting line of an advancing longwall. The pressure differential between intake and return in that vicinity has been reduced to near zero by the installation of a fan and regulator in the return airway. In cases of low normal pressure differentials, an induction fan without any surrounding brattice may be sufficient (Section 4.4.3.). The hydraulic gradients shown on the same figure illustrate the pressure differentials with and without the pressure balance. Where applicable, the method can be applied quickly and at low cost to arrest the development of a spontaneous heating and bring about an immediate reduction in carbon monoxide emissions. A pressure survey (Section 6.3.) may be run around the district to determine the effective fan pressure required. If there is convenient access, then a length of pressure tubing can be installed between the relevant points in the intake and return airways including an in-line pressure gauge. The regulator in the return airway can be adjusted until the pressure balance is achieved. Furthermore, an adjustable orifice within the short length of fan ducting can be used to modify the effective fan pressure. The locations of the fan and regulator may be changed in order to achieve finer control of the zone in which the pressure balance is applied.

The method of localized pressure balancing is very flexible and can achieve spectacular and speedy success in many scenarios. However, it requires skilled personnel to devise and control each installation. If applied inexpertly, it can result in partial recirculation of products of combustion. In any case, it is a prudent precaution to employ a carbon monoxide monitor in the intake to detect any reversal of leakage flows.
Flooding can be used in two ways to extinguish concealed fires. First, if the affected zone lies to the dip of current workings then the fire itself can be flooded. This should be done in a controlled manner in order to be able to handle the products of combustion and other gases displaced by the water. Furthermore, if a previously flooded area is re-opened then it may rapidly re-ignite (Section 21.4.1.2.). The second use of flooding is to provide very effective seals in airways that have low-lying sections. The water penetrates fissures in the surrounding strata and provides a near perfect barrier against air leakage.

Figure 21.7  Example of localized pressure balancing to control a spontaneous heating in a goaf (gob) area.
(a) without pressure balance  (b) with pressure balance
This brings us to the larger matter of building seals to control concealed fires in inby areas. While sealing a section of a mine is often considered as a last resort, it is important to recognize those situations in which it will become inevitable and then to seal quickly. This allows an inert atmosphere to build up and to extinguish active combustion. Re-entry and a resumption of mining may then be possible within a relatively short time. If, however, sealing-off has been delayed unduly, then the fire may have developed into such a large conflagration that the area could be sterilized indefinitely. The one situation in which sealing-off must be delayed is while there is the slightest possibility of rescuing trapped personnel. The procedures involved in constructing seals during a fire emergency are discussed in Section 21.5.

21.5. STOPPINGS, SEALS AND SECTION PRESSURE BALANCES FOR EMERGENCY SITUATIONS

This Section concentrates on the role of stoppings and seals in emergency situations involving fires or explosions. The routine use of these devices during normal mining operations is described in Section 4.2.1.2.

There is considerable divergence in differing geographical areas on the meaning of the terms stoppings and seals. In some regions, the two are regarded as synonymous while, in others, they are interpreted as quite different types of structures. To prevent confusion, we shall use the terms here according to the following definitions:

Temporary stopping: A light structure erected from brattice cloths, other fabrics or boarding but which will not withstand any significant physical loading.

Stopping: A single or double walled structure constructed from blocks, bricks, sandbags, prefabricated steel panels or from substantial boarding attached to tight roof supports. Stoppings are routinely employed as ventilation controls to minimize leakage between adjoining intake and return airways in active mining areas but are not intended to be explosion proof. It is to be expected, therefore, that stoppings will be destroyed or severely damaged in areas through which an explosion has passed.

Seal: A barrier designed to withstand mine explosions.

Temporary stoppings, often in the form of brattice cloths or quickly erected alternatives are employed during the fighting of open fires in order to regulate the airflow over the fire, change the air pressure in nearby airways (Section 21.3.4.1) or re-route airflows to assist in rescue operations.

Stoppings may also be used as a more substantial means of modifying and/or controlling the airflow paths in the event of a mine fire and to re-establish controlled ventilation during rescue operations.

Seals are used when alternative means of controlling a fire or spontaneous heating have been exhausted and inby rescue operations have been terminated. The purpose is to stop the flow of air and allow an inert atmosphere to build up within the affected zone. The decision on when to erect seals should be contingent on the particular circumstances. In the case of a deeply seated concealed fire seals may be erected in all entries to the area at a fairly early stage and before the heating reaches an open airway. The district may subsequently be re-entered for salvage of equipment or resumption of mining either after a cool-down period or when further arrangements have been made to control the heating. Sealing an airway fire should be carried out as soon as it becomes clear that the fire is out of control and rescue operations have been terminated.
21.5.1. Site selection of seals

It is important that all openings into an uncontrolled fire zone be sealed except for sampling pipes. Such openings may include boreholes and connections into old workings. In the shallower mines, there may be fractures extending through to the surface. The potential effects on surface structures and their inhabitants should be considered.

The sites should be selected to minimize the number of seals required. These are matters that should be taken into account during planning of the mine layout in order to facilitate rapid sealing of a section should that subsequently prove to be necessary. The precise locations should be in well supported areas with solid roof, floor and sides.

A well organized mine will have pre-prepared sites for seals at control points in each district and with a stock of the appropriate materials nearby (IME, 1985). In any case, those sites should allow ready access for the supply of further materials and, also, for the provision of ventilation to personnel involved in building the seal. The latter may necessitate a temporary duct and auxiliary fan. If the site is polluted by products of combustion then the construction must be undertaken by teams fitted with breathing apparatus. Where there is danger of an explosion or outrush of gases during construction, sites should be chosen that permit rapid escape of the personnel.

It is unlikely that locations will be found which satisfy all of these requirements. Site selection for stoppings and seals invariably requires compromises between optimum positions and practical considerations.

21.5.2. Sequence of building seals

In the case of concealed fires and where there is no imminent danger of explosion, it is preferable to complete intake seals first in order to terminate airflow into the fire zone, then to follow up quickly with closure of the return airways. However, in the case of an open fire, or where monitored gas concentrations indicate that an explosive atmosphere may develop then all stoppings or seals should be completed simultaneously and personnel evacuated from the mine for a period of 24 hours.

21.5.3. Construction of seals and stoppings

It is unlikely that explosions arising from fires in mines will generate pressure peaks of more than some 350 kPa gauge pressure \(^1\) (50 psig \(^2\)) on the faces of seals although considerably higher pressures have been measured in experimental explosions (Strang, 1985) and in explosions initiated by an ignition of methane. Hence, seals which (by our definition) should be explosion-proof must be able to withstand such dynamic pressure pulses. Figure 21.8 illustrates a typical seal. The end walls may be constructed from sandbags, masonry or any form of non-flammable blocks. The strength of the seal should be provided by the infill and any girders or other supports that may be employed in the intervening space. The length of seal is shown as 3 to 5 m but may be calculated as \( \frac{\text{Airway width} + \text{height}}{2} + 0.6 \) with a minimum length of 3 m (Hornsby et al, 1985) and should be chosen with reference to the condition of the surrounding strata and the type of infill available.

Construction of a seal in coal mines should commence by applying stonedust liberally to the airway inby the seal. Stonedust barriers may also be erected inby the seal to arrest an explosion before it reaches the face of the seal (Section 21.8.3.1.). Figure 21.8 illustrates the end walls recessed into the roof, floor and sides. This is considered to be good practice although it may not be necessary provided that the strata is competent, the length of seal adequate and a modern (quick-setting) wet infill is available. Conveyor

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\(^1\) Sometimes known as "overpressure"

\(^2\) psig means pounds force per square inch gauge pressure. British Imperial units have been added as sections in this chapter are of particular interest in the United States following the Sago and Darby Mine explosions.
structure should be dismantled and cleared from the area. Similarly, pipes and cables should be dislocated and removed.

While dry material such as sand, stonedust or fly ash may be used for the infill, a better seal is obtained by employing a gypsum based plaster. This is pumped in as a liquid which penetrates immediate fractures in the surrounding strata then sets to a compact solid material. It is important that the infill completely blocks the seal to the roof.

The gas sampling pipe should extend at least 30 m inby the seal to allow for 'breathing' induced by fluctuations in barometric pressure. Three flexible tubes may be threaded through the metal sampling pipe in order to draw independent samples from roof, floor and middle of the sealed airway.

The steel access tube shown on Figure 21.8 has a number of purposes. First, it allows ventilation to be supplied to the fire while construction of the seal is in process. This tends to retard the development of a fuel-rich situation and, hence, reduces the risk of explosion whilst workers are still at the site. Despite the access tube, workers may be subjected to roll-back of smoke during construction of a seal. When each of the seals has been completed then all access tubes should all be blanked off simultaneously by strong steel plates both ends. All personnel must leave the mine for a period of 24 hours or until monitored signals of gas concentration indicate that an inert atmosphere has been attained.

Following the sudden cessation of airflow, significant changes take place within the sealed zone. The concentration of combustible gases increases while the oxygen content decreases. In the case of open fires, it is probable that the mixture of gases will pass through an explosive range (Section 21.7.). Meanwhile, convection effects will create movements of the changing atmosphere and, perhaps, rolling or 'tidal' flames. The combination of these effects may produce a series of explosions. These can be monitored by pressure transducers connected to the pressure tubes on the seals. An older but simpler method is to leave a glass U-tube containing mercury attached to a pressure tube. If the mercury is subsequently found to have been blown out then an explosion has occurred within the sealed zone.

A second reason for the access tube is to allow subsequent re-entry and inspection by rescue teams equipped with breathing apparatus. The outby end of the tube should be at a convenient height for this purpose. Should it become necessary, the access tube can be filled with infill material. This is the reason for the slight downward inclination of the tube towards the sealed area.

Figure 21.8 Example of an explosion proof seal where an active fire exists in the sealed area.
There is one major disadvantage to the type of seal we have been discussing - the time taken for its construction. In the case of an open fire in a timbered or coal-lined airway, rapid action is vital. It may be necessary to isolate the zone in a much shorter time period than that required to build an explosion proof seal. An alternative is to erect a stopping that contains a pressure-relief flap. This is a technique that has met with some success in the United States (Mitchell, 1990). Figure 21.9 illustrates such a vented stopping. Boards are attached firmly to two or three tight chocks (cribs) on both upstream and downstream sides except for the top quarter or third of the airway. A sealant can be sprayed over the boards. A weighted flap of conveyor belting is attached across the top of the outby side and held open by a cord which is tensioned by the weight of a canister of water. At the appropriate moment the canister at each site is punctured and all personnel leave the mine. The flaps close as the canisters empty.

Except for violent explosions, the structure of a vented stopping will remain intact. Lesser pulses simply blow open the flap which then falls back into place leaving the integrity of the stopping secure. Although the vented stopping does not have the structural strength or resistance of an explosion-proof seal, it provides a temporary expedient until permanent seals can be established, should those prove to be necessary.

21.5.4. Re-opening a sealed area

The simple method of breaching stoppings or seals and re-establishing an air circuit suddenly is not one that can be recommended when a mine or section of a mine has been sealed because of a fire. Even when gas samples indicate that the fire is no longer active, hot spots may still remain that could result in re-ignition when supplied with oxygen. The preferred technique is first to send in rescue teams equipped with breathing apparatus to make a thorough inspection of the area and to take additional air samples. If the sealed area is extensive and, particularly, if the atmosphere will pass through an explosive range when mixed with fresh air, it may be necessary to re-ventilate in stages, building additional seals or stoppings further inby and closer to the original fire zone.
21.5.5. Section pressure balances

No stopping or seal in the subsurface has infinite resistance. Even if the stopping or seal itself were perfect, the potential for leakage still occurs in the surrounding strata. Such leakage can delay the extinction of a fire or, if serious, may maintain the fire indefinitely. Section 21.4.5.4. described how air pressure management can be employed to control the leakage airflows feeding a concealed fire. The same principle can be applied to complete sections of a mine.

The most rudimentary form of *pressure balancing* involves the re-arrangement of doors and air-crossings to ensure that each seal or stopping is exposed to return airway pressure. This reduces the pressure differential applied across the sealed district to the pressure drop over the length(s) of airway between the stoppings. Figure 21.10(a) illustrates an improved version. An additional wall is constructed some 3 to 5 metres in front of each main stopping or seal to form a *pressure chamber*. A duct of about 0.5 m diameter connects the chambers and, hence, equalizes their pressures. There is then no pressure differential applied across the sealed district, nor can there be any continuous flow around it. These are examples of *passive* pressure balancing.

![Diagram of passive pressure balance](image)

(a) A passive pressure balance.

![Diagram of active pressure balance](image)

(b) An active pressure balance chamber.

Figure 21.10  Passive and active pressure balance chambers.

The drawback to passive pressure balances is that they do not prevent "breathing" of the seals or stoppings during periods of changing barometric pressure. This can be overcome by employing an *active* (or powered) *pressure balance* system. This provides a means of equalizing the air pressures on the inner and outer faces of the same seal or stopping and is illustrated on Figure 21.10(b). Two ducts pass through the outer wall into the pressure chamber. One of these (the pressure duct) is supplied by a ventilating pressure either by a small fan or by laying the duct through a nearby door or stopping in order to utilize the mine ventilating pressure. The pressure chamber will be pressurized positively if the seal is on the return side of the main ventilation network and negatively if it faces an intake airway. The second
(balancing) duct simply passes through the outer wall. Both ducts are fitted with variable dampers to give a flexible means of fine adjustment to the pressure within the chamber.

Two tubes are employed to monitor air pressure, one passing completely through the seal and the other into the chamber. A gauge across these two tubes will indicate zero when there is no pressure differential between the chamber and the inby side of the seal. The duct dampers can be adjusted manually when the pressure gauge deviates from zero. The arrangement can be made automatic by employing an electronic pressure gauge which transmits an amplified signal to servo-motors on one or both of the duct dampers.

A number of special purpose adaptations to the principle of pressure chambers have been devised. One example may occur in a nuclear waste repository to ensure that leakage through an air lock takes place in one consistent direction at all times, irrespective of the direction of pressure differential across the airlock (Brunner et al, 1991). Another example, where permitted by legislation, is the use of compressed air to provide positive pressurization of the pressure chamber.

21.6. THE USE OF INERT GASES

The injection of inert gases to assist in the control of subsurface fires has been undertaken since, at least, the 1950's (Herbert, 1988). However, from 1974, significant developments in the deployment of nitrogen took place in Germany. The technique has become commonplace in coal mining areas where spontaneous combustion occurs frequently (Both, 1981). The overall purpose of injecting an inert gas is to reduce the oxygen content in order to prevent or inhibit combustion. The objectives may further be classified as follows:

- To prevent concealed heatings in zones that are highly susceptible to spontaneous combustion.
- To reduce the risk of explosions during sealing or stopping-off procedures.
- To accelerate the development of an inert atmosphere in a newly sealed zone and to prevent the creation of an explosive mixture when it is re-opened.
- To control the propagation of an open fire during rescue, firefighting and sealing operations.
- To prevent an explosive mixture forming due to “breathing” of a seal.

Three types of gases have been used in the procedure for which the term inertization has been coined; carbon dioxide, products of combustion and nitrogen. In this Section we shall discuss the employment of these gases in addition to methods of application and control.

21.6.1. Carbon dioxide

Carbon dioxide has a density of 1.52 relative to air (Table 11.1). This makes it particularly useful for the treatment of fires in low-lying areas such as dip workings or inclined drifts (Froger, 1985). However, the same property can render it difficult to control in horizontal workings. A 20 t tanker of liquid carbon dioxide will produce some 9000 m³ of cool gas. The liquid form may be piped into the area where it is required and expanded through an orifice or, indeed, injected directly into a localized heating. In both cases, the gas removes heat from the fire as well as promoting an inert atmosphere. However, piping the liquid carbon dioxide can give rise to freezing problems as well as difficulties in handling the pipes.

The use of carbon dioxide as an inverting gas has several other disadvantages. It is quite soluble in water and can suffer some loss in wet conditions. More significant perhaps, is the fact that it adsorbs readily on to coal and coked surfaces, even more so than methane (Figure 12.2(b)). When exposed to incandescent carboniferous surfaces it may be reduced to carbon monoxide. Furthermore, it is considerably more expensive than nitrogen.
21.6.2. Combustion gases

Following the sealing of a fire zone, gases produced by the combustion processes, combined with the consumption of oxygen, will produce an extinguishing atmosphere. However, it may be rich in combustible gases and become explosive if air is subsequently re-admitted (Section 21.7.3.). The products of full combustion, primarily mixtures of carbon dioxide, nitrogen and water vapour, have been employed as an injected inert gas. Flue gas from burning coal has been used in China (Sun, 1963) while modified jet engines have been employed in several countries including Poland, Russia and Czechoslovakia (Strang, 1985). The latter method involves burning kerosene at rates of some 0.7 kg/s to produce 30 m³/s of inert exhaust gases. These are cooled by large quantities of water and admitted into the fire zone. The engine produces a power output of about 30 MW which can be usefully employed. Where the law allows the underground use of a jet engine or where it can be employed on surface for a drift mine, then the large output of inert exhaust gases makes it attractive. However, it nullifies the employment of gas analysis as a means of following the progression of the fire, the capital cost is high and a highly specialized team is required to operate and maintain it.

21.6.3. Nitrogen

Liquid nitrogen is the basis for the majority of inertization schemes now employed for subsurface fires. Again, the liquid gas is supplied in tankers of, typically, 20 t capacity giving about 16500 m³ of gas. For continuous operation throughout a period of gas injection, the tankers may unload into a bulk storage vessel of up to 40 t capacity and which has been brought to the mine site.

Due to the low boiling temperature of nitrogen, the liquid must be evaporated before piping it into the mine. Figure 21.11 indicates the principle of a mobile evaporator which, again, has been brought to the mine for the emergency period. Typically, two water circuits are employed; a primary circuit using atmospheric heat and secondary heaters powered by electricity or liquid/gas fuels. The gaseous nitrogen passes through a bank of controllers before entering the mine pipeline. A subsidiary nitrogen line provides a feedback to maintain a constant pressure in the storage vessel. The maximum gas feed rate into the mine depends upon the duty of the evaporator but may, typically, be within the range 1 to 6 m³/s.

Liquid nitrogen is a by-product of the commercial production of oxygen and is much less expensive than liquid carbon dioxide. Furthermore, it is not as soluble as the latter, does not adsorb so readily on carbon surfaces and with a density approximating that of air, mixes readily without stratification.

21.6.4. Methods of application and control

In order to assess the volume flow of inert gas required, the rate of oxygen supply to the fire may be determined from the inlet oxygen concentration and a measured or estimated airflow. It is then a straightforward calculation to determine the flow rate of inert gas required to dilute the oxygen down to 10 per cent in order to extinguish flaming combustion, or to less than 2 per cent to suppress smouldering.

The inert gas may pass into the mine via water pipes or compressed air pipes commandeered for the purpose. Alternatively, gas feed boreholes may be drilled from the surface to intersect a fire zone (e.g. Zabrosky and Klinefelter, 1988). Perhaps the most difficult aspect of inertization is controlling dilution of the inert gas by air leakage. If leakage air enters in significant quantity between the gas injection point(s) and the fire, then the technique may fail. It follows that inertization is most likely to succeed where the fire is in a single entry with no leaking crosscuts or, in the case of a concealed fire, where the air inlet points have been well defined. Conversely, multi-entry systems offer a greater opportunity for
dilution of the inert gas. Although the employment of inert gases can create difficulties in interpreting the analyses of gas samples taken downstream from the fire, it is usually possible to detect whether the fire is being suppressed. If there is no noticeable effect on an open fire within an hour or two then potential air leakage points should be investigated. Additional stoppings or pressure management techniques may be required to reduce the inward leakage.

For the control of spontaneous heatings in the goaf (gob) areas of longwall mines, the location of the fire and air entry points should first be established (Section 21.4.5.4.). Furthermore, a knowledge of air migration paths in goaf areas is invaluable (Figures 21.4 and 21.5). Injection pipes should be inserted from the airways or working face into the inlet zones, using boreholes if necessary. The volume flowrates of inert gas required are usually much lower for spontaneous heatings than for open fires. However, the reaction may be slower. Indeed, the monitored concentrations of carbon monoxide and methane may increase for up to 36 hours as the inert gas displaces those gases from the fire zone. A steady nerve is useful at such times. When the carbon monoxide concentration begins to fall, that can be used as a controlling guide to the required injection rate of the inert gas. Oxygen concentration in the return airways downstream from goaf (gob) inertization should be monitored to ensure that it remains above the relevant mandatory limits (19 to 19.5 per cent).

For coal mines with a history of recurring spontaneous combustion, the trend is towards establishing a permanent nitrogen “fixing” plant on the surface (e.g. BHP Billiton Sustainability Report, San Juan Coal Company, 2005). The lower grade of nitrogen that may be produced by this method is of little consequence for inertization. A permanent network of nitrogen pipelines throughout the working sections of the mine allows the gas to be fed at a relatively low rate but continuously into the caved zone.

Figure 21.11 Simplified schematic of a mobile nitrogen evaporator.
Properly designed, this creates an inert atmosphere throughout the critical zones (Figures 21.4 and 21.5). As a further projection to future developments, the infrastructure is then in place for complete inertization of the working face if and when the techniques of automation and remote control make that cost effective.

Where inertization is having a beneficial effect it is important that it be maintained for as long as required. While a deeply seated fire can be controlled by an inert gas, it will seldom be cooled sufficiently to extinguish it. Hence, premature cessation of the operation may result in a rapid escalation of the fire. Similarly, injection into a sealed area should be continued until the oxygen content falls below 2 per cent.

21.7. FIRE GASES AND THEIR INTERPRETATION

In Section 11.3.3. we introduced the gases that are produced in the majority of underground fires. Other than use of the human senses, monitoring the quality of the air in a mine is the dominant method of detecting a fire or spontaneous heating. Sampling the air downstream from a fire or from within a newly sealed area and plotting the trends is the primary method of tracking the behaviour of the fire and the development of atmospheres that are, or may become, explosive. However, as the gases emitted vary with the phases of oxidation, time and temperature, it is necessary to employ skilled interpretation of those trends.

21.7.1. The processes of burning and the gases produced

When coal or timber are burning, three processes are in progress.

- Distillation of gases from the solid material
- Oxidation of the solid material on its surface with the emission of heat and light (this is why the surface glows more brightly when fanned with fresh air) and
- Flaming combustion - the burning of combustible gases produced by the first two processes. Again heat and light are produced. Some of the heat passes back to the surface by radiation and convection to assist in the promotion of further distillation.

The fire gases and their relative proportions depend upon the contributions of each of these three processes. The gases of distillation from coal are carbon monoxide, carbon dioxide, hydrogen and water vapour. Methane and other hydrocarbon gases are also produced. Timber distills the same gases although the amount of hydrogen may be negligible. When flaming combustion occurs, the combustible gases burn to a degree that is governed by the availability of oxygen in the air. The final mixture leaving the fire zone is, therefore, a result of the gases of distillation and the extent to which the fire has become fuel-rich.

The complexity of the processes involved can be illustrated by considering just a few of the ways in which methane can burn:

\[
\begin{align*}
CH_4 + 2O_2 & \Rightarrow 2H_2O + CO_2 \\
2CH_4 + 3O_2 & \Rightarrow 4H_2O + 2CO \\
3CH_4 + 5O_2 & \Rightarrow 6H_2O + 2CO + CO_2
\end{align*}
\]

Furthermore, secondary reactions may produce water gas and further reduction of carbon dioxide to carbon monoxide.
21.7.2. The detection and trend analysis of fire gases

For many purposes of analysis the sampled atmosphere is considered as comprised of air, combustibles and inerts (excess nitrogen and carbon dioxide). Figure 21.6 illustrates the combustible gases emitted as coal is heated in a limited air supply such as occurs in a spontaneous heating. It is clear that carbon monoxide is a leading indicator of the early stages of such a fire. However, the saturated hydrocarbons, in particular, ethylene, are useful indicators of burning coal. In general, as the fire develops into open combustion, the major gaseous product, carbon dioxide, forms an increasing proportion of the pollutant emission.

During any fire incident, running graphs should be maintained of the concentrations of the gases that are monitored or gained from the analysis of samples. Transmitting transducers installed downstream from the fire give a stream of near continuous data. However, in many cases, samples must still be obtained manually by means of evacuated chambers or hand pumps attached to sample containers. The sample vessels should be clean, dry and free from any lubricants that may contaminate the gas sample. A mobile laboratory should be available at the mine surface during any major incident to provide rapid analysis of samples. The rate of sampling must be dictated by the urgency of the situation and the speed at which conditions are changing.

Since the beginning of the twentieth century a number of ratios and composites of gas concentrations have been suggested to assist in the interpretation of fire gases. Table 21.1 indicates some of these.

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<thead>
<tr>
<th>Ratio</th>
<th>Name</th>
</tr>
</thead>
<tbody>
<tr>
<td>( \text{CO} / \Delta \text{O}_2 )</td>
<td>Graham's Ratio or Index for Carbon Monoxide (ICO)</td>
</tr>
<tr>
<td>( \text{CO}_2 / \Delta \text{O}_2 )</td>
<td>Young's Ratio</td>
</tr>
<tr>
<td>( \text{CO} / (\text{Excess N}_2 + \text{CO}_2 + \text{combustibles}) )</td>
<td>Willett's Ratio</td>
</tr>
<tr>
<td>( (\text{CO}_2 + 0.75\text{CO} – 0.25\text{H}_2) / \Delta \text{O}_2 )</td>
<td>Jones and Trickett Ratio</td>
</tr>
<tr>
<td>( \text{CO} / \text{CO}_2 )</td>
<td>Oxides of Carbon Ratio</td>
</tr>
</tbody>
</table>

Table 21.1. Gas ratios used in interpreting trends of gas concentrations produced by mine fires.

A feature of several of the ratios is the oxygen deficiency, \( \Delta \text{O}_2 \). This is a measure of the oxygen that has been consumed and is based on two assumptions; first, that the air has been supplied with 20.93 per cent oxygen and 79.04 per cent inert gases (excepting 0.03 per cent carbon dioxide). That 79.04 per cent contains traces of other gases but is referred to simply as nitrogen. Secondly, it is assumed that no nitrogen has been consumed or added (except from the air) through the area under consideration. If no oxygen is consumed, then the \( \text{O}_2 / \text{N}_2 \) ratio would remain at 20.93/79.04 = 0.2648 irrespective of the addition of other gases. For any measured values of \( \text{O}_2 \) and \( \text{N}_2 \), the concentration of oxygen that was originally in place can be calculated as

\[
\frac{20.93}{79.04} \times \text{N}_2
\]

Hence, the amount of oxygen that has been consumed, or oxygen deficiency is given as

\[
\Delta \text{O}_2 = \frac{20.93}{79.04} \times \text{N}_2 - \text{O}_2 \quad \text{per cent}
\]

(21.8)
The oxidation of coal and the corresponding production of gases were studied by Dr. J.S. Haldane and others at the beginning of the 20th Century (e.g. Haldane and Meachen, 1898). By 1914, Ivon Graham had recognized the importance of carbon monoxide as an early indicator of the spontaneous heating of coal and the equally vital influence of the oxygen that was consumed. He first suggested using the index $\text{CO}/\Delta \text{O}_2$, now known as Graham's Ratio or the Index for Carbon Monoxide (ICO).

Graham's Ratio is the most widely used indicator of an incipient heating in coal mines and has often given warnings several weeks before any odor could be detected. It has the significant advantage that it is almost independent of dilution by leakage of air as this affects both numerator and denominator equally. However, none of the indices listed in Table 21.1 is infallible and Graham's Ratio does have some drawbacks. First, its accuracy becomes suspect if very little oxygen has been consumed, i.e. Graham's Ratio is unreliable if the oxygen deficiency, $\Delta \text{O}_2$, is less than 0.3 per cent. This is a weakness shared by the other indices that involve oxygen deficiency. Secondly, it will be affected by sources of carbon monoxide other than the fire including the use of diesel equipment, or if the air supplied to the fire is not fresh. The latter can occur if the fire is fed, partially, by air that has migrated through old workings and contains blackdamp (de-oxygenated air). Again, like other trace gases or indices, a normal range of Graham's Ratio should be established for any given mine. This will usually be less than 0.5 per cent. Any consistently rising values in excess of 0.5 per cent is indicative of a heating.

Example
An air sample taken from a return airway yields the following analysis:

<table>
<thead>
<tr>
<th>Nitrogen (N$_2$)</th>
<th>79.22 per cent</th>
</tr>
</thead>
<tbody>
<tr>
<td>Oxygen (O$_2$)</td>
<td>20.05 per cent</td>
</tr>
<tr>
<td>Carbon monoxide (CO)</td>
<td>18 ppm = 0.0018 per cent</td>
</tr>
</tbody>
</table>

Oxygen deficiency, $\Delta \text{O}_2$ = \[
\frac{20.93 - 20.05}{79.04} \times 79.22 = 0.93 \text{ per cent}
\]

Graham's Ratio = \[
\frac{\text{CO}}{\Delta \text{O}_2} = \frac{0.0018}{0.93} \times 100 = 0.19 \text{ per cent}
\]

The $\Delta \text{O}_2 / \text{CO}$, or Young's Ratio is, again, nearly independent of dilution by fresh air. Carbon dioxide is the most prolific of the gases produced in mine fires. Hence, the values of $\Delta \text{O}_2 / \text{CO}$ will be much higher than $\text{CO}/\Delta \text{O}_2$. As a fire progresses from smouldering to open flame, the burning of carbon monoxide will produce an increase in carbon dioxide. Hence a simultaneous rise in $\Delta \text{O}_2 / \text{CO}$ and fall in $\text{CO}/\Delta \text{O}_2$ indicates further development of the fire. However, as both ratios have the same denominator, the straightforward plots of carbon dioxide and carbon monoxide show the same trends. Young's Ratio suffers from similar limitations to Graham's Ratio. Additionally, the concentration of carbon dioxide may have been influenced by adsorption, its solubility in water, strata emissions of the gas and other chemical reactions.

Willett's Ratio was introduced by Dr. H.L. Willett in 1951 with specific reference to situations where there is a higher than usual evolution of carbon monoxide by ongoing low temperature oxidation. In these cases, the gradual extinction of a fire in a sealed area may not be reflected well by the carbon monoxide trend alone but as a percentage of the air-free content of the sample.

The Trickett or Jones-Trickett Ratio is used as a measure of reliability of sample analysis and also as an indicator of the type of fuel involved. It can be used for the gaseous products of both fires and explosions. Typical values are shown on Table 21.2. Dilution by fresh air has no effect on the Jones-Trickett Ratio. However, it is subject to the limitations of oxygen deficiency.
The Oxides of Carbon Ratio, CO/CO₂ is a useful pointer to the progression of the fire, rising during the early stages and tending to remain constant during flaming combustion. However, the CO/CO₂ rises rapidly again as a fire becomes fuel-rich and is an excellent indicator of this condition. This ratio may also be favoured because it is unaffected by inflows of air, methane or injected nitrogen (Mitchell, 1990). It is, however, subject to variations in carbon monoxide and carbon dioxide that are not caused by the fire.

<table>
<thead>
<tr>
<th>Fuel</th>
<th>Jones-Trickett Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fires</td>
<td></td>
</tr>
<tr>
<td>methane</td>
<td>0.4 to 0.5</td>
</tr>
<tr>
<td>coal, oil, conveyor belting, insulation and polyurethanes</td>
<td>0.5 to 1.0</td>
</tr>
<tr>
<td>timber</td>
<td>0.9 to 1.6</td>
</tr>
<tr>
<td>Explosions</td>
<td></td>
</tr>
<tr>
<td>methane</td>
<td>0.5</td>
</tr>
<tr>
<td>coal dust</td>
<td>0.87</td>
</tr>
<tr>
<td>methane and coal dust</td>
<td>0.5 to 0.87</td>
</tr>
<tr>
<td>no combustion process</td>
<td>&lt; 0.4</td>
</tr>
<tr>
<td>impossible mixture (reliability check)</td>
<td>&gt; 1.6</td>
</tr>
</tbody>
</table>


Gas concentrations and indices give pointers to the average intensity but not the size of the fire. In some cases, where there is limited leakage between the fire and sampling points, an estimate of the extent of the fire may be made from the flowrate of each of the gases. This is given as the product of the air flowrate and the relevant gas concentration.

The principles of gas detection and methods of sampling are discussed in Section 11.4. Ionization smoke detectors draw smoke particles through a chamber where they are charged by a radioactive source such as americium 241 or krypton 85. An ion collecting grid and amplifier produce an electrical output that is a function of the smoke concentration (e.g. Pomroy, 1988). Versions of this have been produced that can distinguish between smoke from a fire and diesel particulate matter (Lytton, 1988).

The improvements that have been made in electrochemical methods of detecting carbon monoxide (Section 11.4.2.5.) have led to this being a common method of fire detection as part of a mine environmental monitoring system. The employment of computer analysis allows filtering of false alarms from diesels or blasting operations that gave earlier versions a doubtful reputation (e.g. Eicker and Kartenberg, 1984). The filter program may be fairly simple such as ignoring short term peaks and giving audio visual alarms only when a significant upward trend is indicated. More sophisticated systems take
into account the time-variant generation of carbon monoxide between successive sensors distributed along an airflow path (Boulton, 1991).

The combination of computer controlled monitoring systems and ventilation network analysis programs allows the possibility of not only detecting the existence of a fire at an early stage, but also its probable location. This is facilitated by strategic location of the sensors (Pomroy and Laage, 1988). The employment of *tube bundle sampling* is particularly useful for detecting the early and slow development stages of a spontaneous heating. This technique is described in Section 11.4.3.2.

Temperature monitors, often known rather loosely as "heat sensors" have an application when mounted above fixed equipment and, particularly, when used to activate deluge or sprinkler systems (Section 21.3.3.). They are of somewhat limited use as fire detectors in airways as the air temperature drops rapidly downstream from a fire. Such sensors may be subjected to greater variations in air temperature from normal operations (such as passage of a diesel vehicle) than from the early stages of a fire. If made sufficiently sensitive, temperature sensors may lose credibility because of an excessive recurrence of false alarms.

### 21.7.3. Explosibility diagrams

When a fire becomes fuel-rich, there is a danger that explosive mixtures of gases will propagate away from the immediate fire zone. Furthermore, following sealing of a fire area, it frequently occurs that the rising concentration of combustible gases and falling concentration of oxygen produce mixtures that pass through an explosive range. Similarly, the dilution of combustible gases that occurs when a sealed area is re-opened may, again, result in passing through an explosive range. In order to be able to predict and control such circumstances, it is necessary to have an understanding of the flammability limits of gases and gas mixtures.

This subject was introduced in Section 11.2.4. and illustrated by the Coward diagram for methane shown in Figure 11.1. That diagram should be reviewed, if necessary, as a reminder of the concepts of upper and lower flammability and how these converge to a nose-limit as inert gases are added.

In many situations, the large majority of combustible gas is methane and Figure 11.1 may be employed to determine whether the mixture of oxygen, methane and inerts lies within the explosive triangle or is likely to do so in the near future. However, in other cases, the presence of carbon monoxide and/or hydrogen may cause significant changes in the Coward diagram (Coward, 1952, Strang J. and Mackenzie-Wood, 1990). Figure 21.12 shows the individual explosive triangles for all three of these combustible gases. The corresponding coordinate points are given in Table 21.3.

<table>
<thead>
<tr>
<th>Gas</th>
<th>Flammable limits</th>
<th>Nose limits</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>lower</td>
<td>upper</td>
</tr>
<tr>
<td>methane</td>
<td>5.0</td>
<td>14.0</td>
</tr>
<tr>
<td>carbon monoxide</td>
<td>12.5</td>
<td>74.2</td>
</tr>
<tr>
<td>hydrogen</td>
<td>4.0</td>
<td>74.2</td>
</tr>
</tbody>
</table>

Table 21.3. Vertices of explosive triangles (percentages).
If the method of the Coward diagram is to be useful for mixtures involving more than one combustible gas then we must be able to quantify the explosive triangle for those composite mixtures. Let us attempt to do that.

A basic precept in the world of science and engineering is that if anything is done to upset the equilibrium of a system then that system will react in an attempt to reach a new equilibrium with the minimum adjustment of the component parts (Le Chatelier's Principle). Let us adopt subscripts 1, 2 and 3 for the three combustible gases and call their percentage concentrations $p_1$, $p_2$ and $p_3$ respectively. If they do not react chemically with each other then the mixture will have a total combustible concentration of

$$p_t = p_1 + p_2 + p_3 \quad \text{percent} \quad (21.9)$$

Furthermore, Le Chatelier's Principle leads to the prediction that for gas flammability limits, $L_1$, $L_2$ and $L_3$ (where these can be upper, lower or nose limits), the corresponding gas flammability limit of the mixture, $L_{mix}$, will be given by:

$$\frac{p_t}{L_{mix}} = \frac{p_1}{L_1} + \frac{p_2}{L_2} + \frac{p_3}{L_3} \quad (21.10)$$
Example
An air sample produces the following analysis:

\[
\begin{align*}
\text{CH}_4 & : 8 \text{ per cent} \\
\text{CO} & : 5 \text{ per cent} \\
\text{H}_2 & : 3 \text{ per cent}
\end{align*}
\]

Determine the lower flammability limit of this mixture.

Solution
The total percentage of combustible gases is

\[
\rho_t = 8 + 5 + 3 = 16 \text{ percent.}
\]

Using the individual gas lower flammability limits given in Table 21.3, together with equation (21.10) gives

\[
\frac{16}{L_{mix}} = \frac{8}{5.0} + \frac{5}{12.5} + \frac{3}{4.0}
\]

giving \( L_{mix} = 5.82 \) percent as the lower flammability limit of the mixture. A similar calculation may be used to determine the upper and nose flammability limits of the combustible content.

The upper and lower limits lie on the line AB on Figure 21.12 and, hence, are defined completely. However, the oxygen content at the nose limit remains to be found before we can construct the explosive triangle for the mixture. To find the oxygen content at the nose limit, we must first determine the excess inert gas (let us call it nitrogen) that has to be added in order to reach that nose limit.

Consider the situation for methane. If we start from any point on the line AB and add nitrogen then we shall move in a straight line towards the origin, O. The mixture will become extinctive when we cross the line AC. At that moment we shall have added an amount of nitrogen which, when expressed per unit volume of methane, is a constant. (This follows from the fact that both AB and AC are straight lines.) As we can commence at any position on AB, let us choose point B. On adding nitrogen and moving towards O, we shall cross the extinction line at point C where the methane concentration is 14.14 per cent. The remaining \((100 - 14.14) = 85.86\) percent is nitrogen. We have, therefore, added \(85.86/14.14 = 6.07\) m³ of nitrogen for each m³ of methane. A similar exercise can be carried out for carbon monoxide and hydrogen to give the values in Table 21.4.

<table>
<thead>
<tr>
<th>Combustible gas</th>
<th>Nitrogen to be added to make mixture extinctive: (N^+) m³ of nitrogen per m³ of combustible gas</th>
</tr>
</thead>
<tbody>
<tr>
<td>methane</td>
<td>6.07</td>
</tr>
<tr>
<td>carbon monoxide</td>
<td>4.13</td>
</tr>
<tr>
<td>hydrogen</td>
<td>16.59</td>
</tr>
</tbody>
</table>

Table 21.4. Volumes of excess nitrogen to be added, \(N^+\), in order to make flammable gases extinctive.

This table gives the excess nitrogen to be added, \(N^+\), if the combustible content consisted of one gas only. For a mixture of combustible gases the excess nitrogen required, \(N_{ex}\), is given as

\[
N_{ex} = \frac{L_n}{\rho_t} \left\{ N^+_1 \rho_1 + N^+_2 \rho_2 + N^+_3 \rho_3 \right\} \text{ percent}
\]  

(21.11)

where \(L_n\) = percentage of combustible mixture at the nose [equation (21.10 with \(L_{mix} = L_n\)].
The required oxygen content at the mixture nose limit is then simply 20.93 per cent of the air fraction, i.e.
\[
\text{Oxygen (nose limit)} = 0.2093 \times (100 - N_{ex} - L_n) \text{ percent} \quad (21.12)
\]

Example
A sample taken from a sealed area yields the following analysis.

\[
\begin{align*}
\text{methane} & \quad 8 \text{ per cent} \\
\text{carbon monoxide} & \quad 5 \text{ per cent} \\
\text{hydrogen} & \quad 3 \text{ per cent} \\
\text{oxygen} & \quad 6 \text{ per cent} \\
\text{inerts} & \quad 78 \text{ per cent}
\end{align*}
\]

\[ p_t = 16 \text{ per cent} \]

Construct the Coward diagram for this condition.

Solution
(a) Using equation (21.10) and Table 21.3:
(i) lower flammability limit, \(L_{low}\):
\[
\frac{16}{L_{low}} = \frac{8}{5} + \frac{5}{12.5} + \frac{3}{4}
\]
gives \(L_{low} = 5.82 \text{ percent combustible}\)

(ii) upper flammability limit, \(L_{up}\):
\[
\frac{16}{L_{up}} = \frac{8}{14} + \frac{5}{74.2} + \frac{3}{74.2}
\]
gives \(L_{up} = 23.56 \text{ percent combustible}\)

(iii) nose flammability limit, \(L_n\):
\[
\frac{16}{L_n} = \frac{8}{5.9} + \frac{5}{13.8} + \frac{3}{4.3}
\]
gives \(L_n = 6.62 \text{ per cent combustible}\)

(b) Equation (21.11) and Table 21.4 give the excess nitrogen required for an extinctive atmosphere to be
\[
N_{ex} = \frac{6.62}{16} \left[\frac{6.07 \times 8}{16} + (4.13 \times 5) + (16.59 \times 3)\right] = 49.24 \text{ percent}
\]

and, finally, oxygen content at the nose limit is given by equation (21.12) as
\[
0.2093 \times (100 - 49.24 - 6.62) = 9.24 \text{ percent}
\]

The explosive triangle has now been defined completely and has been constructed as Figure 21.13. The actual mixture point has also been entered on the diagram and illustrates that although the mixture is not explosive, it will become so if air is allowed to enter the area.

Coward diagrams are most useful in tracking trend directions of gas mixtures. They do, however, have a drawback. As each new sample analysis becomes available, the mixture point on the Coward diagram moves - but the explosive triangle also changes its shape and position. It is analogous to shooting at a moving target. Fortunately, the calculations are simple and a new triangle can be developed for each sample in order to follow actual trends. The complete process can readily be programmed for automatic
appearance of the updated Coward diagram and mixture point on a computer screen. Running through a set of consecutive data for any given sampling location produces a dynamic picture of explosive triangles and mixture points moving over the screen. This creates a strong visual impact of time-transient trends.

An older method of dealing with variations in gas mixtures is the U.S. Bureau of Mines composite diagram illustrated on Figure 21.14 and founded on earlier work by Zabetakis et al (1959). In this diagram, the y axis is an "effective combustible" defined by a weighted combination of the volumetric percentages of the three combustible gases. The weighting takes account of the explosibility of carbon monoxide and hydrogen compared with that of methane. The x axis is a combination of the excess nitrogen and 1.5 times the concentration of carbon dioxide, the 1.5 allowing for the greater extinctive power of carbon dioxide.

The "excess nitrogen" required here is the percentage of nitrogen in excess of that justified by the oxygen present.

\[
\text{Excess nitrogen} = \text{actual } N_2 - O_2 \times \frac{79.04}{20.93}
\]

The set of explosive triangles are approximations based on methane but adjusted for incremental combined additions of carbon monoxide and hydrogen.

Figure 21.13 Coward diagram for mixture of gases in the example.
Example

Using a more detailed analysis of the sample used in the previous example gives

- Methane: 8 per cent
- Carbon monoxide: 5 per cent
- Hydrogen: 3 per cent
- Oxygen: 6 per cent
- Nitrogen: 68 per cent
- Carbon dioxide: 10 per cent

All gases expressed in percentage by volume.
**Total combustibles,** $p_t = 8 + 5 + 3 = 16$ per cent

**methane ratio** $R = \frac{\text{methane}}{p_t} = \frac{8}{16} = 0.5$

**excess nitrogen**

$$= 68 - \left( 6 \times \frac{79.04}{20.93} \right) = 45.3 \text{ percent}$$

**effective inerts (x axis)**

$$= 45.3 + (1.5 \times 10) = 60.3 \text{ per cent}$$

**effective combustibles (y axis)**

$$= 8 + (0.4 \times 5) + (1.25 \times 3) = 13.75 \text{ percent}$$

Plotting this point on Figure 21.14 shows that it lies above the $R = 0.5$ triangle, thus agreeing with the Coward diagram that this mixture is not explosive but will become so if air is added.

### 21.8. EXPLOSIONS

There has scarcely been a major mining industry that has not been traumatised by underground explosions of gases, dusts and mixtures of the two. The potential for disastrous loss of life when such an explosion takes place is very high. Fatality counts have too often been in the hundreds from a single incident. Fatalities and injuries produced by explosions arise from blast effects, burning and, primarily, from the carbon monoxide content of afterdamp - the mixture of gases produced by the explosion. In the dust explosion at Courrieres coal mine in France (1906), 1099 men lost their lives. One of the most catastrophic explosions on record occurred at Honkieko, Manchuria (1942) when over 1500 miners died. The carnage that took place in coal mines during the Industrial Revolution (Section 1.2.) was caused primarily by underground explosions and resulted in the start of legislation governing the operation of mines.

#### 21.8.1. Initiation of explosions

For an explosion to occur, the same three components of the fire triangle must exist simultaneously; namely, a combustible material, oxygen and a source of ignition. However, there is a further condition for mine explosions; the combustible material must be a gas or finely divided dust mixed intimately with the air and in concentrations that lie between lower and upper flammability limits.

The majority of explosions in mines have been initiated by the ignition of methane. This, in itself, is a very dangerous occurrence. However, it becomes much worse when the shock wave raises combustible dust into the air such that it can be ignited by the flame of the burning methane. The resulting dust explosion is likely to be much more violent than the initial methane blast. Indeed, the majority of methane ignitions result in blue flames flickering backwards and forwards along the gas:air interface without developing into an explosion. It is only when the turbulence of the airflow or thermal effects produce a mixture within the explosive triangle (Section 21.7.3.) that the combustion accelerates into an explosion.

Finely divided particles of any combustible solid can become explosive, including metallic dusts, sulphide ores and most organic materials. Precautions must be taken against explosions in the manufacturing, processing and silo storage of many food stuffs including grain. However, in this Section, we shall confine ourselves to the igniting sources, mechanisms and suppression of explosions that occur in subsurface ventilation systems.

In Section 21.2. we classified the major igniting sources of fires and explosions in mines. Unfortunately, exploisible mixtures of gases can be ignited by electrical sparks of energy levels as low as 0.3 millijoules, as illustrated by the tiny spark that ignites gas in a piezo-electric or induction coil cigarette
lighter. It is, therefore, prudent to give a little further attention to the initiation of explosions by sparking phenomena in addition to those sources of ignition discussed in Section 21.2.

**Incendiary sparking** arises from heat produced from a chemical reaction. The *thermite* process involves the reaction of aluminium powder with iron oxide.

\[
2 \text{Al} + \text{Fe}_2 \text{O}_3 \rightarrow \text{Al}_2 \text{O}_3 + 2 \text{Fe} \tag{21.13}
\]

This process produces so much heat that it has been used on small scale welding operations. A similar effect occurs when a surface consisting of aluminium, magnesium or their alloys collides with a rusted steel or iron surface. Incendiary sparks are produced that are well capable of igniting a methane:air mixture. It is prudent to prohibit the importation of any light alloys into gassy mines and, indeed, this is enforced by law in a number of countries.

**Frictional sparking** has been the cause of the increased incidence of methane ignitions on coal faces and which have accompanied the proliferation of mechanized mineral winning. This is discussed in Section 21.2.4.

In addition to sparking caused by the misuse or damage to electrical equipment, electrostatic sparks may also be capable of igniting a flammable mixture of gases. Electrostatic charges are built up on non-conducting (or poorly conducting) surfaces as a regular feature of many everyday operations and, particularly, at pointed or sharply curved regions on those surfaces. Electrical potentials of 10 000 volts are commonly generated. This phenomenon may occur, for example, where belts run over pulleys, at the nozzles of compressed air jets (particularly, if liquid or solid particles are entrained) and within non-conducting ventilation ducts. Even the charge that builds up on the human body in dry conditions can produce dangerous sparks (Strang and McKenzie-Wood, 1985). In such conditions, workers should not wear rubber-soled footwear.

All machines with moving parts should be adequately earthed against the build-up of electrostatic charges. Other devices that are liable to this phenomenon should be similarly protected. "Anti-static" materials are available for ducting, belts and pipes.

21.8.2. Mechanisms of explosions

An explosion may be defined as a process in which the rates of heat generation, temperature rise and pressure increase become very great due to the rapidity of combustion through the mixture. In a typical methane:air explosion, the temperature rises to some 2000 °C, i.e. by a factor of about seven. Even higher temperatures may be reached if the explosion is contained within a sealed volume. The speed of the process is so great that it is essentially adiabatic. The result is that the pressure in the immediate vicinity increases to a peak value very rapidly and is relieved by expansion of the air. This produces a shock wave that propagates in all available directions.

In a mine opening the expansion is constrained by the airway surfaces giving rise to high velocities of propagation. Initially, the flame front travels more slowly than the shock wave and the explosion is known as a deflagration. However, if the unburned zone ahead of the flame front becomes more conducive to combustion by approaching closer to stochastic condition (approximately 9.6 percent methane) then the flame front will accelerate and the peak pressure at the shock wave rises. Although the shock wave velocity also increases, the distance between the two narrows. This may continue until the flame front catches up with the shock wave (Figure 21.16). The explosion is then described as a detonation. The speed of the explosion and the peak pressure at the shock wave then escalate significantly. Conversely, if the unburned zone between the flame front and the shock wave moves away from stochastic conditions (e.g. by lack of fuel or the presence of stonedust) then the explosion will weaken.
At the actual site of the initial ignition the flame speed and rate of pressure rise are at their lowest. It is only as the shock wave and following flame front progress along the airway that their intensity increases, given the availability of both fuel and oxygen within the explosive range. It is for this reason that victims at the initiation site of an explosion often exhibit little blunt trauma but have succumbed to carbon monoxide poisoning. An ignition of a 10 per cent methane:air mixture in a heading may produce flame speeds of 660 m/s (twice the speed of sound) as it accelerates away from the face of the heading.

Figure 21.14A illustrates the passage of a deflagrating explosion that has been initiated at the face of a heading. Expansion of the gaseous products of combustion can proceed only away from the face of the heading. However, that expansion is resisted by the inertia of the outby atmosphere. The result is the development of the shock wave which is essentially the moving boundary between the normal mine atmosphere and the elevated pressure of the expanding explosion. Hence, there is a highly concentrated pressure gradient across the shock wave. It is this shock wave that causes “blast” effects and can inflict extremely high dynamic loads on objects that lie in the path of the explosion. The shock wave produces another effect. It causes intimate mixing of any accumulations of methane as it passes and will produce a continuous explosive path provided there is sufficient gas and oxygen to remain in the explosive range. Even worse, it raises dust into the air to produce concentrations which, if the dust is combustible, will themselves be explosive - resulting in even more favourable conditions for enriching the fuel path of the explosion.

The mixing process is further exacerbated by the highly turbulent conditions that exist between the flame front and the shock wave. Such turbulence, coupled with the high speed of the unburned mixture can result in scouring of the sides of the airway, especially if the coal is friable. The impact of flying debris can also result in significant erosion of the airway as well as dispersion of dust into the air.

If there is any obstruction that prevents the air being projected forward in front of the shock wave the peak pressure becomes greatly escalated. This occurs when the shock wave encounters a facing seal, stopping or any other ventilation control. Similarly, high pressures will be generated when a developed explosion enters a blind heading. In such circumstances victims are typically found to have suffered from blunt trauma in addition to severe burning. The latter is thought to be caused by swirling flames that exist until extinguished by lack of oxygen.

These are the basic mechanisms that drive explosions through subsurface ventilation networks. Bends, junctions, obstructions and the availability of fuel and oxygen all affect the rate of propagation. A mine
explosion is a highly dynamic phenomenon and seldom reaches steady-state even for a few milliseconds. For most of its life it is either proliferating or subsiding.

21.8.2.1. Gas explosions
Explosions of methane from coal are by no means confined to mines. Ignitions have too frequently occurred in storage silos and in coal cargo ships, causing great damage and loss of life. Methane concentrations of 40 per cent have been measured in such facilities. Layering phenomena and the production of explosive mixtures of methane and air make unloading operations particularly hazardous. Precautions against ignitions in these circumstances include good ventilation of the facility, monitoring for methane and the gaseous products of spontaneous combustion, ready availability of nitrogen or carbon dioxide cylinders for inertization, strict control of ignition sources and prohibition of smoking in the vicinity. Bacteria have also been used to convert the methane into carbon dioxide (Kolada and Chakravorty, 1987).

Incidences of hydrogen explosions at battery charging stations and ignitions of oil vapours from machines have been reported. However, the vast majority of gas explosions in mines have involved methane. Modern systems and standards of mine ventilation are successful in maintaining general body concentrations of methane at safe levels throughout the ventilated areas. However, strata emissions of gas are often at methane concentrations of over 90 per cent. It follows that between the points of emission and the general body of the airstream, zones of an explosive mixture must exist. The ventilation arrangements should ensure that these zones are as small as possible and, wherever practicable, maintained free from igniting sources. For example, the danger of methane, coal dust, air and sparks existing simultaneously at the pick-point of a coal winning machine can be reduced by pick-face flushing or jet-assisted cutting (Section 20.4.1.1.).

Dangerous accumulations of methane in roof cavities should be avoided by filling those cavities with an inert material. Air velocities should be sufficient to prevent methane layering (Section 12.4.2.) and longwall systems should be designed to maintain the gas:air interface within the goaf (gob) and without flushing on to the faceline. In appropriate circumstances, methane drainage will reduce the amount of gas that enters the mine ventilation system (Section 12.5.).

21.8.2.2. Coal dust explosions
The danger of coal dust being exploisible in air depends upon a number of factors including:

(a) *Concentration of the dust and presence of methane:* The flame of a dust deflagration is propagated by the combustion of each discrete particle in turn. If the particles are sufficiently far apart then the heat produced by a burning particle will be insufficient to ignite its closest neighbour. The flame will not propagate. The lower flammable limit is attained when the particles become sufficiently close to allow sequential ignition. At the other extreme, if the dust particles are too close together, there will be insufficient oxygen to allow complete combustion. The system will become internally fuel-rich and the rate of heat generation will fall. The upper flammability limit is attained when the temperature is reduced below the ignition point of the dust.

The lower flammability limit may be of the order of 50 g/m$^3$ with maximum explosibility at 150 to 350 g/m$^3$ depending upon the volatile content of the coal (Holding, 1982). The upper flammability limit might be as high as 5000 g/m$^3$ (Strang and McKenzie-Wood, 1985) but is dependent on the availability of oxygen. To put these flammability limits into perspective, it should be recalled that threshold limit values for respirable coal dust concentrations can be as low as 2 milligrams per cubic metre (Table 19.2). A concentration of 50 g/m$^3$ produces a suffocating atmosphere. It is unlikely that such concentrations of airborne dust will exist under normal conditions in the active branches of a mine ventilation network. Nevertheless, in the absence of dust suppression, exploisible dust concentrations might occur around the cutting heads of rock-breaking machines. Furthermore, a very thin coating of dust on the surfaces of airways or a conveyor can produce an explosive atmosphere when disturbed by a shock wave or highly turbulent conditions.
Coal dust and methane are synergistic in their explosibility characteristics. The flammability limits of each of them are widened in the presence of the other. The relationship between the two varies with the volatile content of the coal. A typical example, based on results reported for an Australian coal, is shown on Figure 21.15. It should be appreciated that when heated, coal emits its volatile content commencing with methane. Hence, a coal dust explosion involves the combustion of gaseous fuels as well as carbonaceous solid particles. It is common for partially de-volatilized coal dust to impact and adhere to solid surfaces during a coal dust explosion. This leaves coked material and carbon filaments on supports, equipment and airway surfaces which assist investigators tracing the role and path of coal dust in an explosion. On the other hand, a high velocity flame may leave little trace of coking.

(b) Fineness of the dust: There appears to have been little work done in establishing definitive relationships between explosibility and the size of dust particles. However, the increased surface area and more intimate mixing with the air given by the finer particles suggest that explosibility will increase as the average particle size decreases. An accepted assumption is that any combustible dust which can be raised into the air is explosible. This encompasses all particles of less than some 250 microns diameter.

(c) Type of coal: There is considerable evidence to show that coal dust explosibility increases with volatile content (e.g. Holden, 1982). Hence, low rank coals are more prone to dust ignitions. Anthracite dust is normally considered to be non-explosible in mining conditions. However, with a sufficiently high energy of initiation, low volatile coal dusts can produce violent explosions. The disastrous explosion in Manchuria (1942) occurred while mining a coal of volatile content 15 to 19 per cent. The explosibility of coal dust decreases with respect to ash and moisture content as well as with age. The latter factor is considered to be due to the coating

Figure 24.14B. The flame of a coal dust explosion emerging from a 2.5 m (8ft) high test gallery at the Barbara Mine, Poland
of a partially oxidized layer on older dust particles. It follows that coal dust on, or close to, a working face is more liable to ignitions.

(d) **Strength of initiating source:** The energy level of the initial ignition plays a large role in governing the rate of propagation and growth in power level of a dust explosion. The majority of coal dust explosions in mines have been initiated by a methane ignition, this providing the starting shock wave and flame front. However, any other igniting source of sufficient power can result in a dust explosion in the absence of a flammable gas. The Courrieres dust explosion (1906) is thought to have been caused in a methane-free atmosphere by a case of prohibited explosives (Cybulska, 1981).

To comprehend the power levels that can be attained by coal dust explosions, let us return to the matter of a spearheading shock wave driven by a more slowly advancing deflagrating flame front. A flame velocity of 50 m/s will produce a shock wave velocity of some 375 m/s. (A lower flame velocity is unlikely to propagate). However, Figure 21.16 shows that if the dust concentration and methane favour the growth of the explosion then the flame velocity will increase at a greater rate than that of the shock wave until both exceed some 1100 m/s. Beyond that, the process may develop further into a **detonation** when adiabatic compression in the shock wave can produces temperatures that exceed the ignition point of the fuel. Gas and dust particles no longer have to wait until they are ignited by a following flame. They ignite spontaneously. The flame front and shock wave then advance in unison and may reach speeds of over 2000 m/s (six times the speed of sound). It is little wonder that coal dust explosions are so feared throughout the world of mining.

Figure 21.16 also illustrates the dynamic pressure developed across the shock wave. Even at the lower end of this curve the pressures are sufficient to disrupt doors, stoppings and air-crossings, while the upper end explains the devastation that can be caused by a well-developed explosion.

During post-explosion inspections, the dominant path of the explosion may be deduced from the twisting of steel supports, the distortion of track rails, the direction of failure of strong stoppings and the dislocation of heavy equipment. The displacement of less sturdy objects may be misleading and can be caused by secondary deflections of the shock wave, particularly at bends or junctions. Furthermore, an explosion is often followed rapidly by an implosion of fresh air due to rapid cooling and contraction of the afterdamp gases. If open burning is in progress then the fresh air can initiate repeated explosions. These phenomena can also leave a confused picture for subsequent investigators.

The passage of an explosive flame is unlikely to ignite solid coal. However, paper, clothing and, especially, feeders of methane may be left burning if there is sufficient oxygen remaining in the area. This is most probable in the fringe areas of the explosion and can initiate further explosions of any methane that continues to be emitted into the area. The probability of such subsequent explosions is greatly increased if the initial explosion(s) have disrupted the ventilation system.
Figure 21.14C Photographs of post-explosion No.1 Surface Portal, dislodged and damaged steel arches, and remnants of ducting. *Westray Mine Public Inquiry, Exhibit No. 59 (Richard, 1997).*
Figure 21.14D Photographs of roof falls, dislodged steel arches, and damaged equipment. Westray Mine Public Inquiry, Exhibit No. 59 (Richard, 1997).
Figure 21.15 Example of the reduction in the lower flammability limit of coal in the presence of methane. Other curves vary according to the rank of the coal.
*(Based on work reported by Jensen et al (1989))*

Figure 21.16 Variation of shock wave velocity and dynamic pressure with respect to flame velocity.
*(Developed from data reported by Strang and MacKenzie-Wood, 1985)*
21.8.2.3. Sulphide dust explosions
Although explosions of sulphide ore dust are a hazard in some metal mines, they do not have the destructive power of coal dust explosions and have attracted much less attention by researchers (Holding, 1975). These explosions are initiated, primarily, during blasting operations of ores that contain more than 40 per cent sulphur. The main danger to life is production of the highly toxic sulphur dioxide gas which may be retained for several hours, particularly if ventilation controls have been disrupted. Furthermore, the gas reacts with water vapour causing corrosion problems from the sulphuric acid fumes that are produced. A number of precautionary measures have been suggested including hosing dust depositions from the walls of stopes prior to blasting and the use of water atomizers before and during blasting (Section 20.3.4.). Other techniques involve stemming the blast holes with water ampoules or powdered limestone and inserting explosive charges with zero-time detonators into bags of hydrated limestone powder suspended in the stopes. The latter technique fills the air with inert and suppressant dust particles immediately prior to detonation of the main round (Hall et al, 1989).

21.8.2.4. Dust explosibility tests
Explosibility tests for mine dusts vary considerably in the equipment and procedures employed. The essential sequence consists of producing a dust:air or dust:oxygen mixture and applying an igniting source, usually an electrical spark of constant energy level. The parameters that are varied include the dust concentration, the type of dust, added amounts of stonedust (usually powdered limestone) and methane concentration. The results may be recorded simply as "ignition" or "no ignition", or in the more sophisticated tests, the rates of increase of pressure and temperature and the peak values of those variables (Jensen et al, 1989).

Considerable variations occur in the reported explosibilities of similar dusts. These may be attributable, in part, to differences in test conditions. The results are influenced, for example, by variations in the uniformity of the dispersed dust, means of measuring dust concentration, strength of the igniting source and turbulence within the test chamber. This latter factor also indicates that whilst laboratory test results may provide valuable comparisons between differing dusts, they are not necessarily indicative of actual behaviour within the mine environment. Full scale galleries have been constructed in several countries for investigations into mine fires and explosions.

Indices of dust explosibility have been defined and referred to specific equipment and procedures in differing countries (e.g. Holding, 1982). While these produce useful relative information, caution should be applied when comparing one index with another.

21.8.3. Suppression of mine explosions
The primary safeguards against mine explosions are, once again, a well designed and operated ventilation system, planned maintenance of equipment and good housekeeping to control accumulations of combustible dust and to ensure adequate applications of stonedust. This brings us to the most widely used method of suppressing the propagation of mine explosions.

21.8.3.1. Stonedust and water barriers
Finely divided limestone dust serves at least two purposes when mixed intimately with coal dust in a mine airway. First, it will act as an inert dilutent when the mixture becomes airborne, serving to increase the distance between combustible dust particles. Secondly, particles of stonedust will absorb heat and reduce the ability of airborne coal dust to propagate a flame. Provided that there is sufficient stonedust present, this technique provides an efficient means of suppressing the full development of an explosion.

It is important that the stonedust be mixed well with the coal dust. Tests have shown that strips of coal dust interspersed with strips of stonedust will sustain an explosion. Furthermore, combustible dust on a conveyor may propagate an explosion even when the airway is adequately stonedusted. Precautions should, therefore, be taken to ensure that the stonedust is spread on roof, floor and sides often and consistently. An efficient
means of doing this is to employ *trickle-dusters*. These are devices that emit stonedust into the air at a controlled rate. The stonedust mixes with the airborne coal dust and settles with it to produce a uniformly mixed deposition on all surfaces.

An additional means of using stonedust, particularly applicable to single entry longwall systems, involves *stonedust barriers*. These are boards supported on pivots across the airway, usually near the roof, and loaded with stonedust. Dust loadings vary from 30 to 60 kg per metre length of board. A number of boards are located close together within a short length of airway to form the complete stonedust barrier. The intention is that the boards and their contents will be dislodged by the shock wave of an explosion to produce a high concentration of airborne stonedust at the time the flame front arrives and, hence, prevent its further propagation.

The locations of stonedust barriers should be considered carefully. If they are too close to the seat of the explosion then the flame front may have passed before the stonedust has become adequately distributed; too far away then the stonedust will become too dispersed by the time the flame front arrives. A sensible arrangement is to have a lightly loaded (30 kg/m) barrier, that can more easily be displaced, located within some 200 m of face operations and more heavily loaded barriers further outby. While stonedust barriers may be sited in all district intakes and returns, conveyor entries are particularly recommended to be fitted with these devices.

Stonedust barriers do have some disadvantages. They must be inspected regularly to ensure that they remain capable of being dislodged - yet this renders them prone to accidental or mischievous disturbance. In damp conditions, moisture-proofed stonedust should be employed to ensure that the dust will be dispersed adequately by a shock wave. Stonedust barriers may fail to suppress methane explosions. Furthermore, if an explosion is allowed to develop into a detonation then it is unlikely to be halted by a stonedust barrier. Indeed, the objective of barriers is to suppress an explosion at an early stage of its development.

A *water barrier* is intended to serve the same purpose as a stonedust barrier. In this case, troughs holding some 40 to 90 litres of water are positioned across the airway and attached to roof supports. The troughs themselves are constructed from polystyrene foam or a similarly weak material so that they will disintegrate when subjected to a shock wave travelling at 100 m/s. The cross-section becomes filled with water droplets. Suppression of the flame is achieved by the cooling effect of evaporation and displacement of oxygen.

21.8.3.2. Triggered barriers and explosion detectors

Stonedust and normal water barriers both suffer from the disadvantage that they rely upon the shock wave to disperse the stonedust or water. Triggered barriers are designed to incorporate an internal power source. A typical design consists of an enclosed water tank connected to a concentrated array of nozzles along a short length of airway. A containment diaphragm prevents flow from the tank to the nozzles under normal operating conditions. A bottle of compressed nitrogen or carbon dioxide is located inside the water tank. The gas container is also fitted with a rupture disk.

Upon activation, a heater within the gas bottle causes the gas to expand and puncture the rupture disk. This causes very rapid pressurization of the water, breakage of the containment diaphragm and delivery of water at high pressure to the nozzles. Several hundred litres of water can be dispersed in less than one second. Some designs utilize a powdered flame suppressant in place of the water and a soft explosive instead of the gas container.

Activation of a triggered barrier is initiated by an electrical signal from a detector device located closer to the working area, where an explosion is more likely to commence. Infra-red, ultra-violet, temperature and pressure pulse sensors have all been employed as detection devices.

In addition to the active power source to disperse the fire suppressant, an advantage of the triggered barrier is that an optimum distance can be selected between the sensor and barrier. This ensures that the barrier will be activated at the correct moment with respect to the approaching flame front.
21.8.4 Explosions in sealed areas.

21.8.4.1 Causes of explosions in abandoned areas.
The majority of mine explosions are initiated by operations in the active zones of the mine, particularly arising from ignitions of methane at the pick points of mining machines. These explosions have sometimes caused damage to seals separating active mining zones from abandoned workings and allowing methane to flood into the current ventilation network. The fuel provided by this methane can add to the proliferation and violence of the explosion (for example, the Westray Mine explosion in Nova Scotia, 1992, Richard, K. P., 1997). It was for this reason that the designs of seals to unventilated old workings sought to be capable of withstanding explosion forces. It has now been accepted that explosions can occur within abandoned areas (example Sago Mine, West Virginia, 2006). It is difficult to determine with certainty the causes of such explosions. The possibilities include:

* **friction**
The collapse of strata that contains quartz (particularly sandstones) is capable of producing incendiary sparks by friction of rock against rock or rock on steel (Powell, 1975, Nagy, 1960)

* **lightening**
Lightening strikes on surface may be conducted for discharge in mine workings, including abandoned areas, via borehole casings, pipes, cables or any other conductive path (Sacks, H.K. and Novak, T., 2005).

* **adiabatic compression**
The atmospheric pressures developed by sudden collapses of strata into sealed areas can cause rises in temperature to the ignition point of explosive mixtures (McPherson, 1995; Lin, W. 1997).

* **spontaneous combustion** which reaches open areas. (See Section 21.4)

21.8.4.2 Recognition of the hazard.
Following the Sago Mine explosion (West Virginia, 2006, 12 fatalities)) and Darby Mine explosion (Kentucky, 2006, five fatalities) urgent new research was initiated into safeguarding miners against the effects of explosions occurring in abandoned areas of coal mines. The National Institute for Occupational Safety and Health (NIOSH) in the United States reported that during the period 1986 to 2006 there were 12 incidents of explosions in sealed areas of mines in the United States (Zipf et al, NIOSH, 2007). In ten of those explosions seals were destroyed indicating that the U.S. design standard then in use 140 kPa (20 psig) was far from adequate. This contrasted with standards of the 345 kPa (50 psig) in the UK or 500 kPa (73 psig) in Poland and Germany) where no seal failures due to explosions in old workings were reported in the same period.

One effect of the Sago Mine explosion was the issuance of a Mine Safety and Health Administration (MSHA) Emergency Temporary Standard that seals should be designed and constructed to withstand an explosion pressure of 345 kPa (50 psig) and that new research on mine seals be initiated (see Section 21.1). This provided the impetus for NIOSH to embark on such research. The first of their reports was issued in 2007 (Zipf et al) with the promise of further investigations to be conducted.

21.8.4.3 Approaches to the problem and the initial NIOSH Report.
An approach to the safety of abandoned workings has been either to ventilate those old workings continuously in order to remove methane or to seal them off and allow an inert atmosphere to be established. Both of those techniques have disadvantages. With regard to continuous ventilation, even when air passes slowly through caved areas it is still possible for pockets of methane and explosive mixtures to exist within those areas, particularly in inclined workings. Secondly, it is most unwise to pass air through old workings where the coal is prone to spontaneous combustion. And thirdly, recalling that ventilating fans are often the largest consumer of electrical power at a mine, the associated costs can become very high if airflows are used to ventilate increasing areas of abandoned workings.

With regard to sealing off an abandoned area, the intention is that an inert atmosphere be established in sealed areas by the combined effect of oxidation (replacing oxygen by carbon dioxide) and emissions of
methane. However, this can be effective only if the seals are designed to minimize leakage. Furthermore, recalling from Section 4.2.3 that the effective combined resistance, $R_{eff}$, of seals constructed essentially in parallel falls dramatically as their number increases it becomes important that the number of seals into the area is also minimized.

$$R_{eff} = \frac{R}{n^2}$$  where $R$ is the resistance of a single seal and $n = \text{number of seals.}$$

An added complication is that seals will “breathe” as the external atmospheric pressure varies. (Section 4.2.2). The effect is that air will leak through or around seals during periods of a rising or falling barometric pressure. For these reasons an inert atmosphere may never be attained in all parts of an abandoned and sealed area. In particular, the breathing of seals can produce a time variant zone of explosive mixture immediately inby each seal.

The NIOSH Report of 2007 (Zipf et al) described a rigorous investigation involving laboratory (closed volume) tests, full scale gallery tests and the utilization of mathematical models. These tests examined

- the dependence of pressures developed by an explosion on the volume of explosive mixture,
- the case of products of combustion expanding into inert voidage and caved areas
- the case of gases resulting from the explosion venting into other active or inactive parts of the mine,
- the run-up distances through which an explosion can proliferate up to becoming a detonation (Section 21.8.2.2).

On the basis of their investigation, NIOSH suggested that if abandoned working are to be sealed rather than ventilated then mine management decide on a case-by case basis whether to

- (a) build unmonitored seals that will withstand most explosions within old workings, or
- (b) utilize lighter seals but to monitor continuously the atmosphere in a zone inby the seals.

Table 21.5 gives a synopsis of the applications of unmonitored and monitored seals that are described in more detailed in the NIOSH 2007 Report.

At the time of writing, NIOSH is continuing with its investigations. In particular, further work should provide guidance on the construction of seals.

Australia has already adopted regulations that consider the conditions under which unmonitored and monitored seals can be used (Oberholzer and Lyne, 2002). Monitoring is normally carried out by means of tube bundle systems (Section 21.7.2). Furthermore, in addition to monitoring, many Australian mines utilize inertization techniques to maintain an inert atmosphere inby the seals (Section 21.6).
### Table 21.5 A simplified synopsis of the design pressures recommended for unmonitored and monitored seals.

*(Derived from Zipf et al. Explosion pressure design criteria for new seals in U.S. coal mines. NIOSH IC 9500, 2007)*

Figures 21.17 and 21.18 show layouts that illustrate the possible locations for district, panel and cross-cut seals.

#### 21.8.4.4 Requirements of seals

Due to the many uncertainties associated with atmospheric conditions and changing geometries within sealed areas it is difficult to know with any certainty the distributions and concentrations of gas mixtures much beyond the seal locations. Hence the conservative approach is to assume that well developed explosions can occur within the sealed area and to construct seals that can withstand such explosions. The requirements of such seals are:

- they be as leakproof as practicable, taking into account the number of seals to be built
- capable of withstanding repeated explosions
- explosion proof in both directions taking into account the shear strength of strata in the roof, floor and sides, and
- that they can withstand strata convergence over the long term.

It may prove difficult to design practicable and cost-effective single-wall seals that can meet all of these requirements. A wise objective of further investigations would be aimed at designing mine layouts that minimize the number of seals required and the utilization of double walled seals with an infill that is economically and readily available. Figure 21.19, with or without the sampling arrangements, illustrates the type of seal that has proved successful in meeting these listed requirements.

It is a prudent precaution to pre-prepare the sites of future seals as soon as practicable after excavation of the relevant airways. Such preparation should include cut-outs for the seal walls in airway sides and the storage of materials for construction of the seals. Such pre-preparation will prove invaluable in cases of emergency. Furthermore, airways should be liberally stone dusted both inby and outby the sites of seals.
Figure 21.17 Panel extraction with panel and cross-cut seals. Multiple cross-cut seals are required.
Figure 21.18  Barrier pillars left between panels (subject to rock mechanics considerations.)
No cross-cut seals required.
21.9 PROTECTION OF PERSONNEL

21.9.1. Training and preparedness

In all underground emergencies, the first priority is the safety of personnel. The surest protection against the initiation and hazards of mine fires and explosions is training and practice of safety procedures. Classes and practical sessions should be held, not only for new recruits, but at regular intervals of time for all employees. These sessions should include discussions on the causes of fires and explosions and how they propagate. The elements of subsurface firefighting and operation of fire-suppression equipment should be outlined. It is prudent to ensure that supervisors and other selected members of the workforce should receive additional instruction and practice in firefighting. As in mine rescue work, competitions between teams can do much to foster interest. This further training may include the use of temporary stoppings, airflow control and air pressure management. However, training of all underground personnel should concentrate particularly on

- warning systems and location of trapped persons
- self rescuers
- escapeways and
- refuge chambers.

These four items are discussed in the following subsections.

Mine management plays a crucial role in minimizing the risks of fires and explosions, and in responding correctly when they do occur. In addition to matters of training, the planned arrangement of services dictates the efficiency of emergency response (Section 21.10.). Communications, routes of water pipes, cables, airflows and escapeways, availability and effectiveness of firefighting equipment, reliable environmental monitors and an integrated fire-protection policy (Sheer, 1988) can make the difference between a localized ignition that is quickly extinguished and disastrous loss of life. Ventilation and safety engineers should engage in mock scenarios and computer simulations of emergency situations. Such exercises assist in raising questions related to response strategies and mine preparedness such as establishing distances from work locations to refuge chambers and escapeways.
All underground workers should be required to engage in fire drills and emergency evacuation procedures at least once in each three months. These should involve leaving the mine by escapeways other than the normal travelling routes. Such drills should be accompanied by discussions on choices of escape routes and ventilation systems, and should also encourage familiarity with the layout of airways within the mine.

It is a useful exercise, as part of emergency preparedness, for mine management to establish prior friendly relationships with local police, fire, hospital and news agencies. This can be further facilitated by arranging occasional tours of the mine for such people.

21.9.2. Methods of warning and locating personnel

The rates at which mine fires can propagate and products of combustion spread throughout an underground network of airways make it vital that all personnel in the mine be warned of the emergency as quickly as possible. This can be difficult because of the distances involved and wide dispersal of the workforce, particularly in metal mines. Video cameras may be located at strategic locations although these might be obscured by smoke.

Most mining industries employ "firebosses, firemasters" (or similar titles) whose task is to patrol the mine before and, perhaps, during each working shift and following blasting operations to search for any indications of a fire. To ensure adequate and timely coverage of the mine, tags are left at strategic points showing the time and date of the last visit. Fire patrols should follow the direction of airflow in order to more easily locate the probable source of any smoke or fire odour.

Modern paging-type mine telephone systems enable voice warnings to be transmitted to main work areas and along major transport routes. These are particularly suitable for concentrated mining methods such as the longwall system. However, the extension of paging techniques along the several hundred kilometres of active airways in a large metal mine would be cost prohibitive. The methods that have been devised to transmit fire warnings to a widespread workforce in underground openings include:

- stench warning systems,
- radio signals and
- more recent systems

Following receipt of any form of fire warning, mine workers should immediately engage in the emergency evacuation or personnel protection procedures in which they should have been trained and practiced (Section 21.9.1.).

21.9.2.1. Stench warning systems

Versions of this technique have been employed in large metal mines since the 1930's. The procedure involves releasing a gas having a very distinctive odour into intake airways or compressed air systems. Having been exposed to low concentrations of the stench gas during training sessions, miners recognize the odour and can initiate emergency procedures. The gases used resemble those introduced into natural gas distribution systems. Ethyl mercaptan mixed into freon has been commonly employed. However, this can be toxic, corrosive and produce an unbearable stench at high concentrations. Furthermore, it may lose its potency when transported through steel pipes. A safer and healthier alternative is tetrahydrothiaphene (Ouderkirk et al, 1985).

A basic disadvantage of the stench warning technique is that it relies on the mine ventilation system. Transmission times of the stench depend upon the large range of air velocities that exist within a mine ventilation network. In openings of large cross-sectional area, travel times can
become dangerously high to serve as a fire warning system. Furthermore, the concentration of the odour will also vary from very high, close to the emission point, to barely detectable in remote areas of a complex ventilation network. To add to the difficulties, the ventilation system may have been modified or disrupted by the fire (Section 21.3.2.).

These drawbacks can be minimized by selecting multiple injection points in locations reasonably close to major work areas, in addition to downcast shafts or other primary inlets. Stench gas injector mechanisms should incorporate pressure balance arrangements and controlled release rates to prolong the warning and to avoid short peaks of excessively high concentration. Inby injectors should be capable of remote activation. The choice of injector sites and release rates should be made carefully. The objective is to carry a clearly detectable odour as rapidly as possible to all parts of the mine where personnel may be working. Ventilation network analysis packages that incorporate gas distribution modules are very helpful in selecting optimum locations and release rates (Section 7.4.4.).

21.9.2.2. Ultra-low frequency radio signals.

Many investigations have been conducted into the use of radio waves for warning mine personnel of an emergency condition and, also, for locating persons who have become trapped. High or medium frequency radio signals are limited to line-of-sight transmission or depend upon the existence of metal conductors such as pipes, cables or conveyor structure. For personnel warning or location, the radio signals must be capable of transmission through rock and of being detected by lightweight and low-powered personal receiver units.

Older developments in this area have involved ultra-low frequency devices. At frequencies in the range 630 to 2000 Hz and employing transmitting powers of 1 kW, rock penetration distances of over 1600 m have been achieved (Hjelmstad and Pomroy, 1990). This indicates that transmitters located on surface will be suitable for most mines, while underground transmission sites will serve for the deeper workings. The signal strength decays inversely with the cube of the distance through the rock.

The transmitter consisted of a radio (electromagnetic) signal generator, amplifier and antenna. The latter may be made from ten coils of insulated copper wire coiled into a loop of some 30 m diameter. The personal receiver units are small, intrinsically safe and may be powered by the wearer’s caplamp battery. The pencil size receiver antenna is formed from a high-permeability ferrite core wound with copper wire and is very efficient in detecting electromagnetic radiation. More highly powered units can be fitted to vehicles or other equipment. The transmitter and receivers are tuned to a common resonant frequency to discriminate against electromagnetic noise or stray signals. The voltage produced by a receiver unit is amplified and may then be used to generate an audio-visual warning. Incorporation into the caplamp unit can produce on-off blinking of the light until nullified by operation of a switch. Research continues into the use of rock-penetrating radio waves for voice communication. Again, this will serve for both warning and personnel location purposes.

21.9.2.3. More recent systems

Following the Sago Mine explosion (West Virginia, 2006) a plethora of wireless tracking systems have been developed to locate persons in underground mines, particularly in Australia and the United States. These systems have included hand-held radios, leaky feeder and radio frequency devices. The first of these to be approved by the U.S. Mine Safety and Health Administration is normally energized via d.c. power cables with back-up battery power in case of failure of the cable system. (MSHA, 2008)3. The units can be used to track miners both before and after an emergency situation. Text messaging as well as gas detection can be added to the system to maintain communications between equipped miners and a central location where all such

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communications are logged. At the time of writing other tracking devices are under investigation for approval in U.S. mines.

21.9.2.4. Locating trapped persons
The situation has all too often arisen where miners have become trapped in locations that are inaccessible because of accumulations of gases, products of combustion, water or falls of roof. To facilitate rescue operations, it is necessary to identify the locations of the trapped personnel.

A very old method that has often led to life-saving operations is the seismic technique. An explosive charge or heavy hammer (pile-driver) blow on the mine surface may be heard or felt by personnel in relatively shallow mines. In deeper operations, similar seismic signals may be generated from underground locations. On hearing these signals, the trapped personnel respond by hammering on pipes, rails or, simply, the floor or walls. If this response is detected by the ears or geophones of rescue personnel then mine maps are consulted, if necessary, to select the probable location(s) of the trapped persons. Pilot boreholes may be drilled to those locations to supply food and air while rescue operations proceed.

21.9.3. Self-rescuers
An estimated 80 to 90 per cent of fatalities in mine fires or explosions are caused by carbon monoxide poisoning. Self-rescuers are compact versions of breathing apparatus and are intended to allow the wearers to pass through atmospheres that are contaminated by smoke or carbon monoxide. In many countries, it is mandatory for all persons who enter an underground mine to carry a self-rescuer. The lighter versions can be worn on the belt of each person who ventures underground while caches of the heavier self rescuers may be kept close to work areas and on vehicles. The purpose of self-rescuers is to allow the wearers to escape from or through contaminated atmospheres. Indeed, they are often known as escape breathing apparatus (EBA’s). They are not intended for rescue operations or other type of work and should not be confused with the more specialized breathing equipment used by trained rescue teams.

The two essential features of self-rescuers are the types of atmosphere in which they are effective and the time they allow for escape. Their operational duration depends not only upon the type of self-rescuer but also:

- manual effort (e.g. speed of walking)
- breathing habits, experience, physical condition and mental state of the wearer, and
- environmental conditions.

Although many lives have been saved by self-rescuers, others have been lost even when such devices were available. In many of these cases, the error has been in failing to don the apparatus at the appropriate time. This matter should be stressed heavily during training sessions. In particular, self-rescuers should be put on as soon as it is suspected that the air is, or may become, contaminated, i.e. by means of telephone messages, personnel warning systems (Section 21.9.2.), shock waves or strong pulses in the ventilation, unusual odours or the appearance of smoke. Furthermore, during training, personnel should be required to practice unpacking and donning the apparatus. Training versions of the units are available for this purpose. Experience has shown that classroom instruction on the opening and donning of a self-rescuer is insufficient to emulate the physical and psychological conditions under which it may be used in an emergency. It is recommended that the procedure be practiced in a smoke-filled atmosphere or, at the very least, with closed eyes.

There are basically two types of self-rescuer units, the filter self-rescuer (FSR) and the self-contained self-rescuer (SCSR).
21.9.3.1. Filter self-rescuers (FSR's)
Each of these small and compact units fits inside a hermetically sealed plastic or stainless steel case which is worn on the belt. To use the device, the seal is broken by a lever arrangement, the mouthpiece inserted inside the lips and gripped by the teeth, and nose clips put on. The complete unit is held close to the face and chin by head straps. Air is drawn through three types of filter before passing through a heat exchanger to the mouthpiece. Exhaled air also traverses the heat exchanger and through a one-way valve to the external atmosphere.

The first level of filtration simply removes dust particles. The second filter is a drying agent - typically 9 per cent lithium chloride and 91 per cent calcium bromide impregnated into activated charcoal (Strang and MacKenzie-Wood, 1985). The activated charcoal assists in removing sulphur dioxide, hydrogen sulphide and oxides of nitrogen. The third level of filtration contains a catalytic mixture of granulated manganese dioxide, copper oxide and a little silver oxide. The mixture is known widely as hopcalite. This catalyst converts carbon monoxide into carbon dioxide with close to 100 per cent efficiency. It is, however, poisoned by water vapour. Indeed, the duration of the device is governed by the drying agent in the second level of filtration. The period of operation is, typically, about one hour for saturated conditions and carbon monoxide concentrations of up to 1.5 per cent.

Filter self-rescuers suffer from two disadvantages. First, they rely on there being at least 16 per cent oxygen in the ambient atmosphere (Section 11.2.2.). Secondly, both the drying filter and the hopcalite involve exothermic reactions. Indeed, the temperature of the inhaled air is an indication of the level of carbon monoxide present. At high concentrations, the temperature of the filtered air may exceed 90°C and cause great discomfort including blistering of the lips and mouth. However, removing the mouthpiece for temporary relief in such circumstances is likely to be fatal. The small heat exchanger close to the mouth piece absorbs heat from the filtered air and rejects it to the exhaled air. Because of these limitations it is prudent to replace filter self-rescuers by self-contained self-rescuers with the possible exception of small mines with ready access to the surface.

21.9.3.2. Self-contained self-rescuers (SCSR's)
As the name implies, these supply all the respiratory needs of the wearer and are independent of the gaseous constituents of the ambient atmosphere. A considerable variety of self-contained self-rescuers have been produced. The main disadvantage is their weight and bulk. Current generations of SCSR's vary from 2 to 5 kg and may not be sufficiently convenient for personal wear - hence, requiring caches of the units to be available close to active working areas. Research continues in attempts to develop a self-contained self-rescuer that is sufficiently light and compact to be worn on the belt while giving the 60 or 90 minute operating life that may be mandated by law. National and state legislation should be consulted for regulations that govern the requirement, types and operating durations of self-rescuers.

As with the FSR's, the self-contained versions are hermetically sealed in polypropylene or metal containers to ensure a long shelf life. However, if the seal is broken or damaged, the unit must be replaced. Periodic water bath tests should be carried out to ensure that the casings remain airtight. The units include noseclips, fitting straps and goggles. Two types of self-contained self-rescuers are in common use.

The compressed oxygen SCSR is a recirculating system. Expired air passes through a soda lime filter which removes carbon dioxide, then into a flexible "breathing bag" where it is enriched with oxygen from a compact gas cylinder. The oxygen may be supplied at a base rate of some 1.2 to 2 litres per minute but will be increased automatically if deflation of the breathing bag indicates a rising demand for air. The air passes to and from the mouthpiece via a flexible breathing tube. The unit is worn on the chest and supported by neck and waist straps.
The chemical SCSRs are lighter devices and develop oxygen from the reaction that occurs when water vapour and carbon dioxide from exhaled breath pass through potassium superoxide \((KO_2)\). The oxygen in this compound contains an additional electron compared with free gaseous oxygen. This gives a fairly weak bond and free oxygen is released readily according to the following reactions:

\[
4KO_2 + CO_2 + H_2O \rightarrow K_2CO_3 + 2KOH + 3O_2
\]  

(21.14)

and

\[
KOH + CO_2 \rightarrow KHCO_3
\]

Exhaled air from the mouthpiece passes down the flexible breathing tube and through a bed of granulated potassium superoxide where oxygen is added and carbon dioxide removed. It is then collected in a breathing bag for re-inhalation. A heat exchanger can be incorporated to maintain the temperature of the inhaled air to no more than 45°C in most situations.

The chemical production of oxygen can be initiated simply by exhaling vigorously into the unit. However, there may be little or no time to accomplish this safely. Chlorate candles are incorporated into most chemical SCSR's to overcome this difficulty. The chlorate candle is started by percussion or other device when a ripcord is pulled. This provides some three to four litres of oxygen during the following few minutes and until the potassium superoxide reaction becomes effective.

Because of their vital role in cases of emergency, it is important that self-rescuers be inspected according to a timetable that may be imposed by regulations but should occur at intervals of no more than 3 months or after an SCSR has been carried or worn on the person of any individual. In particular the seal of each unit should be examined to ensure its integrity. A log should be maintained on inspections of each individual unit.

The bulk and weight of most current self-contained self rescuers may render them unsuitable for continuous wearing by a miner. Caches of SCSRs may be stored at underground locations. Such caches should be within a travel time of five to ten minutes, depending on the duties of any worker, preferably in a level intake airway but accessible from more than one entry. However, caches of SCSRs should be available at both the main gate and tail gate ends of a longwall panel and in designated escapeways. Caches of SCSRs should be checked as part of pre-shift inspections. These should be at least the number of SCSRs stored underground as the maximum number of personnel underground at any one time. Training should ensure that each miner knows the locations of SCSR caches. It is prudent to have self rescuer units on personnel transportation vehicles.

21.9.4. Escapeways

In any subsurface facility, certain paths should be selected as preferred evacuation routes in the event of an emergency. These escapeways should be highlighted during training sessions and fire drills. There are three matters of importance:

- selection of airways as escapeways
- preparation and maintenance, and
- protection and use of escapeways during an emergency.

The choice of escapeways commences by a pragmatic examination of the airflow routes, travel distances, geographical layout of airways, the physical state of those airways, directions of leakage through stoppings, and the locations of air crossings, doors and other ventilation controls that may be dislocated by a fire or explosion. The primary requirement is that escapeways should be maintained free from products of combustion for as long as possible following the outbreak of a fire.
At least two escapeways must be available from all areas of routine mineral extraction. This is usually mandated by law. In the case of single development headings or other special circumstances, legislation may limit the number of persons who may be allowed into those areas at any one time. For single entry longwall systems, the section intake and return airways must serve as the escapeways. However, even in multi-entry systems it is prudent to maintain at least one return as an escapeway. Incidents in the intakes or mineral extraction zone may prevent evacuation along intake escapeways. Depending upon air velocities and the efficacy of any barricades that may be erected, it might be possible to outpace the airflow and stay ahead of products of combustion along a well maintained return escapeway.

At least one (and preferably more) intake airway for each section of a mine should be maintained as an escapeway. Where two or more intakes are designated as escapeways then they should be truly independent ventilation routes as far back to the primary mine inlets as practicable. Two parallel airways, where one is ventilated by air that has passed through the other, are not independent ventilation routes and cannot be regarded as separate escapeways (MSHA, 1984). Strata stresses should also be considered in selecting escapeways. Routes that are subject to crushing will become excessively expensive to maintain in a good travellable condition and should not be chosen as escapeways.

Network analysis programs with pollutant distribution modules (Section 7.4.4.) can be employed to check and improve the selection of escapeways for any given fire scenario. Computer packages that simulate fire situations are particularly valuable for this purpose (Section 21.3.2.2). Even more specialized programs have been developed specifically for the identification of preferred escapeways (Barker-Read and Li, 1989). The reliability of escapeways can be subjected to objective tests through the techniques of fault-tree analysis through which the causes and consequences of events may be interlinked, tracked and analyzed (Goodman and Kissell, 1989).

The preparation and maintenance of escapeways should be carried out with careful consideration given to the potential conditions in which those escapeways may be used, i.e. in zero visibility by persons wearing self-rescuers and who are in a state of anxiety. Obstructions, patches of poor roof, uneven or tracked floors and unfenced junctions may create little difficulty during routine travel but can make the difference between life and death when the airway is filled with smoke.

Escapeways must be maintained in a condition suitable for unimpeded foot travel at all times. Signs and coloured reflectors should be employed to indicate directions of escape. However, here again, these visual indicators may be rendered invisible by smoke. Reflectors should, preferably, be located on the sidewalls rather than at the roof of airways as smoke is liable to be thickest at roof level. Some form of lifeline should extend throughout the length of an escapeway. This may simply be a water or compressed air pipe, a cable, wire line or part of the structure of a conveyor, but placed at a convenient height to be followed by hand contact. Such lifelines should also have cones or other devices at intervals to give a tactile indication of direction. It is prudent to deposit extra caches of self-rescuers in well-identified locations within escapeways.

Persons who work in any section of a mine should be completely familiar with the relevant escapeways. This can be promoted by using alternative exit routes during fire drills and displaying escapeway maps within the section and at other locations, such as shaft stations, where miners may congregate.

During an actual emergency, every attempt should be made to maintain escapeways free from products of combustion for as long as possible. This might be accomplished by the placement of brattice cloths or other devices to control pressure differentials between airways (Section 21.3.4.1.). Again, training classes and fire drills give the opportunity of imparting familiarity with procedures involved in evacuation through escapeways. These include the advisability of travelling in groups for mutual assistance and the use of lifelines.
21.9.5. Refuge chambers

When personnel have become trapped within underground workings and all escape routes have become inaccessable then refuge chambers provide a last resort to preserve life. These are lengths of airway or fire-resistant prefabricated chambers which might be portable and either rigid or inflatable within which miners may wait until they can be reached by rescue personnel. Blind headings or other single ended zones are the preferred locations. However, totally enclosed prefabricated chambers may be sited in through-flow airways or cross-cuts. The requirement, number and locations of refuge chambers may be mandated by legislation. Where used, they should be within reasonable access from active working sections (NIOSH, 2007).

Refuge chambers should be sized according to the maximum number of persons who work in that area of the mine and for an occupancy period of not less than two days. Assuming that each resting person has need of 0.15 litres of air per second (Table 11.2) gives an air requirement of 13 m³/day. However, this should be multiplied by a safety factor of 3 to allow for increased concentrations of exhaled carbon dioxide. Hence, if twenty persons are to be accommodated, then the chamber should have a minimum volume of 13 x 2 x 3 x 20 = 1560 m³. An opening of cross-sectional area of 15 m² would, therefore, require a length of 1560/15 = 104 m if there is no additional air feed from a compressed air line or surface borehole.

Two types of refuge chamber have bought trapped miners vital waiting time. Pre-prepared and fully equipped refuge chambers should be considered for workings that are distant from surface connections, and in long development or exploration headings. They should ideally be equipped with water, compressed air, communication with the surface, gas and temperature monitors, sanitary arrangements, extra self-rescuers, food in sealed containers, first aid supplies, oxygen cylinders, emergency supplies of breathable air, and reading/writing materials (Halasz, 1985). The entrances to pre-prepared refuge chambers should be well marked. Routes leading to refuge chambers may be fitted with directional blinking lights (similar to Christmas tree lights) to assist miners traversing polluted airways. All underground workers and management should be trained in the maintenance and use of refuge chambers and be aware of the existence and locations of such chambers. Prefabricated chambers should be constructed from non-combustible and heat-resistant materials.

As a last resort, trapped personnel may be able to construct improvised refuge chambers provided that brattice cloths, timber, hammers, saws and nails are available. Leakage points may be sealed by clothing, rags, boards or, indeed, anything else that is available. In such cases, an active compressed air line into the zone is invaluable in maintaining the pressure above that of the connecting and, perhaps, polluted airways. If such barricades have to be erected within a continuous airway then attempts should be made to minimize the pressure differential applied across the ends. This may entail holing stoppings or opening doors in the immediate vicinity.

Following the NIOSH Report of Dec. 2007 the Mine Safety and Health Safety Administration of the United States published for public comment a Proposed Rule on Refuge Alternatives in American coal mines (MSHA, 2008). At the time of writing, it is intended that a Final Rule will be established by the end of 2008. This follows a period since 1969 when federal law allowed the regulatory agency to require that refuge chambers be installed in coal mines. Until 2008, however this authority was never exercised.

21.10 EMERGENCY PROCEDURE AND DISASTER MANAGEMENT

The manner in which a major fire or explosion in a mine is handled depends largely on the forethought and planning that has been expended on such an eventuality. The early stages of an emergency are often fraught with uncertainty, and chaos can easily occur. It is, therefore, vital that those in charge of subsurface operations should have established and documented a definitive
procedure to be followed when a dangerous condition is discovered. All key personnel should be very familiar with that procedure. This final Section discusses the three areas that should be addressed.

21.10.1. Immediate response

There are two sets of actions that should occur immediately and simultaneously when a major fire or an explosion takes place. First, all available warning systems for underground personnel should be activated; the numbers of persons underground and their work locations should be established, return airways should be evacuated, and responsible persons should be sent to inspect the affected area (as far as is possible). While this is in progress, telephone messages should be transmitted to key persons or agency representatives not already at the site. These include

- mine management and senior supervisory staff
- heads of mine specialist departments (e.g. ventilation, safety, electrical, mechanical)
- regional mine rescue centres and rescue stations at neighbouring mines
- government inspectorate or enforcement agencies
- union officials
- local police, fire and medical facilities.

A list of these persons or agencies and their telephone numbers should be posted permanently in a prominent position in the surface operations centre of the mine and updated whenever necessary. At all times, when people are in the mine, there must be a designated person on surface who has the authority and responsibility to initiate these immediate actions.

21.10.2. Command centre

An emergency command centre should be established in a surface office as rapidly as possible. This should be manned by a few key officials, normally the senior management of the mine and others with detailed and expert knowledge of firefighting and rescue operations. Those persons should have the authority to take part in decision making. However, they should be supported and advised by specialist engineers, other government officials and union representatives. Office accommodations equipped with desks, telephones, mine maps and stationery should be made available for these supplementary groups adjoining the command centre. Provision should also be made for rescue personnel and those involved in the analysis of gas samples.

It is important that the chain of command is firmly established. There should be one person in overall control. Decisions are usually arrived at after consultation and discussion. However, the final word lies with that one person. Instructions relating to firefighting, rescue and evacuation must be issued only from the command centre. Direct communication should be maintained between the command centre and the mine telephone system as well as to surface fans, power centres and pumping stations.

21.10.3. Disaster management

In addition to the operations that are conducted underground during an emergency, there are numerous other facets that require detailed organization and management. It is to be expected that many other people will arrive at the mine site throughout the emergency, including the news media, family and friends of underground workers and volunteers varying from willing but unskilled lay persons to specialist consultants. Police authorities should be asked to assume control of traffic and all access roads to the mine property including surface buildings and mine entrances. Medical facilities may require to be set up on the mine surface and vehicles made
readily available for transportation to hospitals. Medical personnel should be established on site and who must be able and willing to be escorted underground if necessary.

Accommodation may need to be found for immediate family members of trapped miners. This should, preferably, be close by but off the mine site. The presence of clergy and specialists in trauma reactions can be of tremendous assistance in those facilities. Responsible persons should be put in charge of continuous catering facilities and janitorial services.

Every effort should be made to establish good liaison with media personnel and to accommodate them with arrangements for interviews and telephones. One spokesperson for the mining company should be responsible for all statements to the press. As part of disaster preparedness, that person should have been trained in communicating through the media; in particular, making accurate and relevant statements and responses to questions within a 30 or 60 second television interview. It assists in maintaining good relationships with media personnel to keep them informed of the time of forthcoming statements.

Both company personnel and reporters should be aware of their mutual responsibilities. The latter have a duty to inform the public while, at the same time, they should not engage in any activities that will interfere with rescue or firefighting operations. As televised interviews can be broadcast immediately or within seconds, it is of great importance that information relating to individuals be given to concerned family members before being made available to the media. While the great majority of news people act in a responsible manner, there is the occasional renegade whose enthusiasm exceeds his/her common sense. In one such incident, a young reporter managed to evade the check-in procedures at the top of a mine shaft and reached an underground fresh-air base during rescue operations. He was brought out on a stretcher, having fainted at the scene.

References


