Chapter 4. Subsurface Ventilation Systems

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4.1. INTRODUCTION

Practically every underground opening is unique in its geometry, extent, geological surroundings, environmental pollutants and reasons for its formation - natural or man-made. The corresponding patterns of airflow through those openings are also highly variable. There are, however, certain features that are sufficiently common to permit classifications of structured ventilation systems and subsystems to be identified.

In this chapter, we shall discuss the essential characteristics of subsurface ventilation systems, first on the basis of complete mines and primary airflow routes. The opportunity is taken to introduce some of the technical terms used by ventilation engineers. The terms chosen are those that are in common use throughout the English speaking mining countries. Secondly, we shall look at district systems for more localized areas of a mine. These, in particular, vary considerably depending upon the geometry of the geologic deposit being mined. Although reference will be made to given mining methods, the treatment here will concentrate on principles rather than detailed layouts. In most countries, state or national mining law impacts upon the ventilation layout. System designers must, as a pre-requisite, become familiar with the governing legislation. In the absence of any relevant legislation applicable to the location of the mine, or where the engineer perceives it to be inadequate, then it is prudent to utilize the pertinent laws from another country that has a well-developed history of mining legislation.

Thirdly, auxiliary ventilation systems will be examined, these dealing with the ventilation of blind headings. The chapter also deals with the principles of controlled partial recirculation and the ventilation of underground repositories for nuclear waste or other stored material.

4.2. MINE SYSTEMS

4.2.1 General principles

Figure 4.1. depicts the essential elements of a ventilation system in an underground mine or other subsurface facility.

Figure 4.1. Typical elements of a main ventilation system
Fresh air enters the system through one or more downcast shafts, drifts (slopes, adits), or other connections to surface. The air flows along intake airways to the working areas or places where the majority of pollutants are added to the air. These include dust and a combination of many other potential hazards including toxic or flammable gases, heat, humidity, and radiation. The contaminated air passes back through the system along return airways. In most cases, the concentration of contaminants is not allowed to exceed mandatory threshold limits imposed by law and safe for the entry of personnel into all parts of the ventilation system including return airways. The intake and return airways are often referred to simply as intakes and returns respectively. The return air eventually passes back to the surface via one or more upcast shafts, or through inclined or level drifts.

**Fans**
The primary means of producing and controlling the airflow are also illustrated on Figure 4.1. Main fans, either singly or in combination, handle all of the air that passes through the entire system. These are usually, but not necessarily, located on surface, either exhausting air through the system as shown on Figure 4.1 or, alternatively, connected to downcast shafts or main intakes and forcing air into and through the system. Because of the additional hazards of gases and dust that may both be explosive, legislation governing the ventilation of coal mines is stricter than for most other underground facilities. In many countries, the main ventilation fans for coal mines are required, by law, to be placed on surface and may also be subject to other restrictions such as being located out of line with the connected shaft or drift and equipped with "blow-out" panels to help protect the fan in case of a mine explosion.

**Stoppings and Seals**
In developing a mine, connections are necessarily made between intakes and returns. When these are no longer required for access or ventilation, they should be blocked by stoppings in order to prevent short-circuiting of the airflow. Stoppings can be constructed from masonry, concrete blocks or fireproofed timber blocks. Prefabricated steel stoppings may also be employed. Stoppings should be well keyed into the roof, floor and sides, particularly if the strata are weak or in coal mines liable to spontaneous combustion. Leakage can be reduced by coating the high pressure face of the stopping with a sealant material and particular attention paid to the perimeter. Here again, in weak or chemically active strata, such coatings may be extended to the rock surfaces for a few metres back from the stopping. In cases where the airways are liable to convergence, precautions should be taken to protect stoppings against premature failure or cracking. These measures can vary from "crush pads" located at the top of the stopping to sliding or deformable panels on prefabricated stoppings. In all cases, components of stoppings should be fireproof and should not produce toxic fumes when heated.

As a short term measure, fire-resistant brattice curtains may be tacked to roof, sides and floor to provide temporary stoppings where pressure differentials are low such as in locations close to the working areas.

Where abandoned areas of a mine are to be isolated from the current ventilation infrastructure, seals should be constructed at the entrances of the connecting airways. If required to be explosion-proof, these consist of two or more stoppings, 5 to 10 metres apart, with the intervening space occupied by sand, stone dust, compacted non-flammable rock waste, cement-based fill or other manufactured material. Steel girders, laced between roof and floor add structural strength. Grouting the surrounding strata adds to the integrity of the seal in weak ground. In coal mines, mining law or prudent regard for safety may require seals to be explosion-proof.

**Doors and airlocks**
Where access must remain available between an intake and a return airway, a stopping may be fitted with a ventilation door. In its simplest form, this is merely a wooden or steel door hinged such that it opens towards the higher air pressure. This self-closing feature is supplemented by angling the hinges so that the door lifts slightly when opened and closes under its own weight. It is also
advisable to fit doors with latches to prevent their opening in cases of emergency when the direction of pressure differentials may be reversed. Contoured flexible strips attached along the bottom of the door assist in reducing leakage, particularly when the airway is fitted with rail track.

Ventilation doors located between main intakes and returns are usually built as a set of two or more to form an airlock. This prevents short-circuiting when one door is opened for passage of vehicles or personnel. The distance between doors should be capable of accommodating the longest train of vehicles required to pass through the airlock. For higher pressure differentials, multiple doors also allow the pressure break to be shared between doors.

Mechanized doors, opened by pneumatic or electrical means are particularly convenient for the passage of vehicular traffic or where the size of the door or air pressure would make manual operation difficult. Mechanically operated doors may, again, be side-hinged or take the form of roll-up or concertina devices. They may be activated manually by a pull-rope or automatic sensing of an approaching vehicle or person. Large doors may be fitted with smaller hinged openings for access by personnel. Man-doors exposed to the higher pressure differentials may be difficult to open manually. In such cases, a sliding panel may be fitted in order to reduce that pressure differential temporarily while the door is opened. Interlock devices can also be employed on an airlock to prevent all doors from being opened simultaneously.

*Regulators*

A passive regulator is simply a door fitted with one or more adjustable orifices. Its purpose is to reduce the airflow to a desired value in a given airway or section of the mine. The most elementary passive regulator is a rectangular orifice cut in the door and partially closed by a sliding panel. The airflow may be modified by adjusting the position of the sliding panel manually. Louvre regulators can also be employed. Another form of regulator is a rigid duct passing through an airlock. This may be fitted with a damper, louvres or butterfly valve to provide a passive regulator or a fan may be located within the duct to produce an active regulator. Passive regulators may be actuated by motors, either to facilitate their manual adjustment or to react automatically to monitored changes in the quantity or quality of any given airflow.

When the airflow in a section of the mine must be increased to a magnitude beyond that obtainable from the system then this may be achieved by active regulation. This implies the use of a booster fan to enhance the airflow through that part of the mine. Section 9.6 deals with the subject of booster fans in more detail. Where booster fans are employed, they should be designed into the system such that they help control leakage without causing undesired recirculation in either normal or emergency situations. In some countries, coal mine legislation prohibits the use of booster fans.

*Air crossings*

Where intake and return airways are required to cross over each other then leakage between the two must be controlled by the use of an air crossing. The sturdiest form is a natural air crossing in which the horizon of one of the airways is elevated above the other to leave a sill of strata between the two, perhaps reinforced by roof bolts, girders or timber boards. A more usual method is to intersect the two airways during construction, then to heighten the roof of one of them and/or excavate additional material from the floor of the other. The two airstreams can then be separated by horizontal girders and concrete blocks, or a steel structure with metal or timber shuttering. Sealants may be applied on the high pressure side. Control of the airway gradients approaching the air crossing reduces the shock losses caused by any sudden change of airflow direction. Man-doors can be fitted into the air-crossing for access.

Completely fabricated air crossings may be purchased or manufactured locally. These can take the form of a stiffened metal tunnel. Such devices may offer high resistance to airflow and should be sized for the flow they are required to pass. They are often employed for conveyor crossings. Another type of air crossing used mainly for lower airflows and which requires no additional excavation is to course one of the airstreams through one or more ducts that intersect a stopping on
either side of the junction. An advantage of this technique is that the ducted airflow may be further restricted by passive regulators or up-rated by fans in the ducts.

In all cases, the materials used in the construction of air crossings should be fireproof and capable of maintaining their integrity in case of fire. Neither aluminium nor any other low melting-point or combustible material should be employed in an air crossing.

4.2.2 Location of main fans

In the majority of the world’s mines, main fans are sited on surface. In the case of coal mines, this may be a mandatory requirement. A surface location facilitates installation, testing, access and maintenance while allowing better protection of the fan during an emergency situation. Siting main fans underground may be considered where fan noise is to be avoided on surface or when shafts must be made available for hoisting and free of airlocks. A problem associated with underground main fans arises from the additional doors, airlocks and leakage paths that then exist in the subsurface.

In designing the main ventilation infrastructure of a mine, a primary decision is whether to connect the main fans to the upcast shafts, i.e. an exhausting system or, alternatively, to connect the main fans to the downcast shaft in order to provide a forcing or blowing system. These choices are illustrated in Figure 4.2 (a and b).

From the time of the shaft bottom furnaces of the nineteenth century, the upcast shaft has, traditionally, been regarded as associated with the means of producing ventilation. Most mines are ventilated using the exhaust system. An examination of the alternatives continues to favour a primary exhaust system in the majority of cases. The choice may be based on the following four concerns.

Gas control

Figure 4.2 shows that air pressure in the subsurface is depressed by the operation of an exhausting main fan but is increased by a forcing fan. The difference is seldom more than a few kilopascals. As strata gases are, typically, held within the rock matrix at gauge pressures of 1000 kPa or more, it is evident that the choice of an exhausting or forcing system producing a few kilopascals will have little effect on the rate of gas production from the strata.

![Figure 4.2 Possible locations of main fans](image-url)

(a) Exhausting system  
(b) Forcing system  
(c) Push-pull system
Unfortunately, much of the gas is not emitted directly into the ventilating airstream but collects in worked out areas, relaxed strata or in voidage that is connected to, but not part of the main ventilation system. Such accumulations of gas are at near equilibrium pressure with the adjacent airways. Hence, any reduction in barometric pressure within the ventilation system will result in an isothermal expansion of the accumulated gases and produce a transient emission of those gases into the ventilation system. This occurs naturally during periods of falling barometric pressure on the surface. In some countries, a mandatory record of surface barometric readings is updated at the beginning of each shift at coal mines. During a period of falling barometric pressure, unusually heavy emissions of methane or deoxygenated air may be expected. At such times, it is not wise to sit down for lunch against a seal or stopping beyond which are old workings.

Steady state operation of either exhausting or forcing fans will produce no changes of air pressure in the subsurface. However, consider the situation of the stoppage of a forcing fan. The barometric pressure throughout the ventilation system will fall rapidly. Accumulations of voidage gas will expand and flood into the workings at the worst possible time i.e. when the airflow is considerably diminished, causing a peak concentration of gas. Conversely, when a main exhausting fan stops, air pressure in the system increases, compressing accumulations of gas. Hence, no peak of general body gas concentration occurs within the airstream. It is true that when the main exhaust fan restarts, the sudden reduction in barometric pressure will then cause expansion and emission of accumulated gas. However, this occurs at a time of full ventilation and the peak of gas concentration will be much less than that caused by stoppage of a forcing fan. A consideration of strata gas control favours a primary exhausting system.

Transportation
The choice between main forcing and exhausting systems should take into account the preferred routes for the transportation of mineral, personnel and materials. Ideally, conveyors, locomotives or other modes of moving broken rock should not be required to pass through airlocks. Hence, a mine design that has mineral or rock transportation routes in main intakes and rock hoisting in a downcast shaft will favour an exhausting system. Alternatively, if there are good reasons to transport mineral in the returns and upcast shaft then a forcing system may be preferred. This could be necessary, for example, in evaporite mines producing potash or halite where the hygroscopic nature of the mineral could give ore handling problems if transported within the variable humidity of intake airways.

In American coal mines, conveyors are normally required to be located in "neutral" airways, ventilated by air that will neither pass on to working faces nor is returning from work areas. This system has the advantage that smoke and gases produced by any conveyor fire will not pollute the working faces. The major disadvantages are the additional potential for leakage and difficulties in controlling the air quantity along the conveyor routes.

Fan maintenance
An exhausting fan will pass air that carries dust, water vapour, perhaps liquid water droplets, and is usually at a higher temperature than that of the air entering the mine. The combined effects of impact and corrosion on impeller blades are much greater on main exhaust fans. Forcing fans handle relatively clean air and require less maintenance for any given duty. On the other hand, corrosive air passing up through the headgear of an upcast shaft can cause much damage. This can be prevented by drawing the air out the shaft into a near surface fan drift by means of a low pressure - high volume fan.

Fan performance
A forcing fan normally handles air that is cooler and denser than that passing through an exhausting fan. For any given mass flow, the forcing fan will pass a lower volume flow at a reduced pressure. The corresponding power requirement is, therefore, also lower for a forcing fan. However, the effect is not great and unlikely to be of major significance.
A counteracting influence is that forcing fans must be fitted with inlet grilles to prevent the ingress of birds or other solid objects. These grilles necessarily absorb available energy and result in an additional frictional pressure drop. Furthermore, the expanding evasee fitted to a main exhaust fan recovers some of the kinetic energy that would otherwise be lost to the surface atmosphere.

When air is compressed through a fan its temperature is increased. If the air contains no liquid droplets and there is insignificant heat transfer through the fan casing then the temperature rise is given as

$$
\Delta T = \frac{0.286}{\eta} \frac{T_1}{P_1} \Delta P \quad ^\circ C
$$

(see derivation in Section 10.6.1.2)

where

- $\eta$ = fan isentropic efficiency (fractional)
- $T_1$ = Absolute temperature at inlet (K)
- $P_1$ = Barometric pressure at inlet (Pa) and
- $\Delta P$ = Increase in absolute pressure across the fan (Pa)

The rise in temperature through a forcing fan will be reflected by an increase in average temperature in the intake airways. However, heat exchange with the strata is likely to dampen the effect before the air reaches the work areas (Section 15.2.2.).

Figure 4.2(c) shows a combination of main forcing and exhausting fans, known descriptively as a push-pull system. A primary application of a main push-pull system is in metal mines practicing caving techniques and where the zone of fragmented rock has penetrated through to the surface. Maintaining a neutral pressure underground with respect to surface minimizes the degree of air leakage between the workings and the surface. This is particularly important if the rubbelized rock is subject to spontaneous combustion. In cold climates, drawing air through fragmented strata intentionally can help to smooth out extremes of temperature of the intake air entering the workings (Section 18.4.6.).

In the more general case of multi-shaft mines, the use of multiple main fans (whether exhausting, forcing or push-pull) offers the potential for an improved distribution of airflow, better control of both air pressures and leakage, greater flexibility and reduced operating costs. On the other hand, these advantages may not always be realized as a multi-fan system requires particularly skilled adjustment, balancing and planning.

4.2.3. Infrastructure of main ventilation routes.

Although the simplified sketch of Figure 4.1 depicts the main or trunk intakes and returns as single airways, this is seldom the case in practice other than for small mines. In designing or examining the underground layout that comprises a subsurface ventilation system, the following matters should be addressed:

Mine resistance
For any given total airflow requirement, the operational cost of ventilation is proportional to the resistance offered to the passage of air (Section 9.5.5.2.). This resistance, in turn, depends upon the size and number of the openings and the manner in which they are interconnected. Problems of ground stability, air velocity and economics limit the sizes of airways. Hence, multiple main intakes and returns are widely employed.

The mine resistance is greatly reduced and environmental conditions improved by providing a separate split of air to each working panel. The advantages of parallel circuits over series ventilation were realized early in the nineteenth century (Section 1.2).
Leakage control

The volumetric efficiency of a mine is defined as

\[ VE = \frac{\text{Airflow usefully employed}}{\text{Total airflow through main fans}} \times 100 \text{ per cent} \]  \hspace{1cm} (4.2)

where the 'Airflow usefully employed' is the sum of the airflows reaching the working faces and those used to ventilate equipment such as workshops, electrical gear, pumps or battery charging stations. The volumetric efficiency of mines may vary from 75 down to less than 10 per cent. The latter value indicates the large and, often, expensive amount of air leakage that can occur in a mine. It is, therefore, important to design a subsurface ventilation system to minimize leakage potential and to maintain the system in order to control that leakage. Whenever possible, intake and return airways, or groups of airways, should be separated geographically or by barrier pillars with a minimum of interconnections.

A prerequisite is that all doors, stoppings, seals and air crossings should be constructed and maintained to a good standard. A stopping between a main intake and return that has been carelessly holed in order to insert a pipe or cable, or one that has been subject to roadway convergence without the necessary repairs may be a source of excessive leakage. Unfortunately, if a large number of stoppings exist between an intake and adjacent return then the leakage may become untenable even when each individual stopping is of good quality. This can occur in workings that have been developed by room and pillar methods. The reason for this is the dramatic decrease in effective resistance to airflow when the flow paths are connected in parallel. For \( n \) stoppings constructed between two adjacent airways, their combined (effective) resistance becomes

\[ R_{\text{eff}} = \frac{R}{n^2} \]  \hspace{1cm} (see section 7.3.1.1)

where \( R \) is the resistance of a single stopping.

Figure 4.3 shows the dramatic reduction in effective resistance that occurs as the number of stoppings increases. In such cases, it becomes important not only to maintain good quality stoppings but also to design the system such that pressure differentials between the airways are minimized.

Air pressure management is a powerful tool in controlling leakage and, hence, the effectiveness, volumetric efficiency and costs of a ventilation system. It is particularly important for mines that are liable to spontaneous combustion. Ideally, resistance to airflow should be distributed equitably between intakes and returns. In practice, one often observes return airways of smaller cross section than intakes and that have been allowed to deteriorate because they are less frequently used for travelling or transportation. This will increase the pressure differentials between intakes and returns. Similarly, local obstructions caused by falls of roof, stacked materials, equipment or parked vehicles will affect the pressure distribution and may exacerbate leakage. The positions and settings of booster fans or regulators also have a marked influence on leakage patterns and should be investigated thoroughly by network analysis (Chapter 7) during design procedures.
Direction of airflow
There are two considerations regarding the direction of the airstream - first with respect to the transportation of the mined material. An antitropical system is one in which the airflow and transported rock move in opposite directions. Other than the "neutral" airways of American conveyor roads, this implies mineral transportation in intake airways. Conversely, a homotropical system is one in which the airflow and the fragmented rock move in the same direction. This implies mineral transportation in the return airways and is often associated with a main forcing system. The homotropical system ensures that any pollution generated from the fragmented rock along the transportation route passes directly out of the mine without affecting working faces. Such pollution may include dust, heat, humidity and gases issuing from the broken rock or equipment. The higher relative velocity between conveyed material and the airflow in an antitropical system can result in a greater entrainment of dust particles within the airstream. Furthermore, a homotropical system is preferable in the event of a fire occurring along the mineral transportation route. On the other hand, siting electrical or other equipment capable of igniting a methane-air mixture in a return airway may be inadvisable or, indeed, illegal for gassy mines.

The second concern in the matter of airflow direction is the inclination of the airway. An ascensional ventilation system implies that the airflow moves upwards through inclined workings. This takes
advantage of the natural ventilating effects caused by the addition of heat to the air (Section 8.3.1). In open stoping or mining layouts that involve multiple connections in inclined workings, ascensional ventilation may be the only technique capable of either controlling or utilising natural ventilating effects. Descensional ventilation may be employed on more compact mining systems such as longwall faces and normally then becomes also a homotropal system with both air and conveyed mineral moving downhill. However, this may cause difficulties in controlling the natural buoyancy effects of methane in waste areas. The advantage claimed for descensional ventilation is that because the air enters the workings at a higher elevation it is then cooler and drier than if it were first coursed to the lower end of the workings.

**Escapeways**

Except for blind headings, there should always be at least two means of egress from each working place in an underground mine or facility. Preferably, there should be two separate intake routes designated as escapeways in case of a fire or other emergency. Within this context the term "separate" is taken to imply that those airways have different and identifiable sources of intake air such that a source of pollution in one of them will not affect the other - either through leakage or series ventilation. Nevertheless, at least one return air route must always remain open and travellable without undue discomfort, to allow for an emergency situation where the working face itself becomes impossible to traverse.

Escapeways should be marked clearly on maps and by signs underground. Personnel should be made familiar with those routes through regular travel or organized escape drills. Mining legislation may dictate minimum sizes for escapeways and the frequency of their inspection.

**Airflow travel distance and use of old workings**

The routes utilized for main intake and return airflows should be reviewed from the viewpoint of travel distance and corresponding time taken for a complete traverse by the air. For high strata temperatures, it is advantageous for intake air to reach the workings as quickly as possible in order to minimize the gain of heat and humidity. However, this is tempered by air velocity constraints and ventilation operating costs.

In mines located in **cold climates**, it may be preferable to encourage natural heating of the intake air by allowing it to take a circuitous and slow route in order to maximize its exposure to rock surfaces. Another situation occurs when variations in air humidity or temperature cause problems of slaking (sloughing) of strata from the roof or sides of the airways or workings. Here again, a case may be made for the natural air conditioning gained by passing the intake air through a network of older airways prior to reaching the current work areas (Section 18.4.6.).

The employment of old workings as an integral part of a ventilation system can result in significant reductions in mine resistance and, hence, the operating costs of ventilation. Furthermore, return air passing through abandoned areas will help to prevent buildup of toxic, asphyxiating or flammable gases. However, using old workings in this way must be treated with caution. It is inadvisable to rely upon such routes as they may be subject to sudden closure from falls of roof. Secondly, travellable intake and return airways must always be maintained for reasons of safety and, third, old workings liable to spontaneous combustion must be sealed off and the pressure differentials across them reduced to a minimum.

From a practical viewpoint where old workings can be employed safely for airflow then it is sensible to use them. However, during system design exercises they should not be relied upon to provide continuous airflow routes but, rather, as a bonus in reducing the costs of ventilation. In any event, as a mine develops, it becomes advisable to seal off old areas that are remote from current workings. Unless this is done, then overall management and control of the airflow distribution will become increasingly difficult.
4.3. DISTRICT SYSTEMS

4.3.1. Basics of district system design

Underground ventilation layouts serving one or more districts of a mine may be divided into two broad classifications, U-tube and through-flow ventilation. Each of these takes on a diversity of physical configurations depending upon the type of mine and disposition of the local geology.

As illustrated on Figure 4.4 the basic feature of U-tube ventilation is that air flows towards and through the workings, then returns along airways separated from the intakes by stoppings and doors. Room and pillar layouts and advancing longwalls tend to be of this type.

Figure 4.5 illustrates the alternative through-flow ventilation system. In this layout, primary intakes and returns are separated geographically. Adjacent airways are either all (or mainly) intakes or returns and, hence, reducing the number of leakage paths. There are far fewer stoppings and air crossings but additional regulation (regulators or booster fans) is required to control the flow of air through the work area. Practical examples of through-flow ventilation are the parallel flows from downcast to upcast shafts across the multilevels of a metal mine, or the back-bleeder system of a retreating longwall.

The simplest possible application of the U-tube system is for a set of twin development headings. Indeed, the U-tube method is the only one capable of ventilating pilot workings that are advancing into an unmined area. Through-ventilation requires the prior establishment of one or more connections between main intake and return airways. Once that has been accomplished then through-ventilation has several significant advantages. First, leakage of air from intake to return is greatly reduced. Hence, lower total airflows are required to provide any required ventilation at the
working face. Secondly, the parallel airways and, often, shorter total travel distance of the airstream give a lower district resistance - particularly for workings distant from the main shafts. This permits reduced ventilating pressures. The combination of lower total airflows and lower ventilating pressures leads to large reductions in ventilation operational costs. Furthermore, the fan duties will remain much more stable in a through-flow system than the escalating demands of an advancing U-tube layout.

Figure 4.5 Through-flow ventilation system

4.3.2. Stratified deposits

The vast majority of underground mines extracting coal, evaporites or other tabular forms of mineral deposits normally do so by one of two techniques, longwall or room and pillar (bord and pillar) mining. While the actual layouts can vary quite significantly from country to country and according to geological conditions, this Section highlights the corresponding modes of airflow distribution that may be employed.
Longwall systems

The two major features of longwall mining that have influenced the design of their ventilation systems are, first, the control of methane or other gases that accumulate in the waste (gob) areas and, second, the high rate of rock breakage on heavily mechanized longwalls that exacerbates the production of dust, gas, heat and humidity.

Figure 4.6 illustrates some of the ventilation layouts used on longwall districts. Single entry systems are employed primarily in European coal mines. Figures 4.6 (a and b) show the application of the U-tube principle to advancing and retreating longwalls respectively. With the advancing system, leakage of some of the intake air occurs through the waste area, controlled by the resistance offered by the roadside packing material and the distribution of resistance and, hence, air pressure around the district. This can give rise to problems of gob fires in mines liable to spontaneous combustion.
Gases from the waste may also flush onto the face leading to unacceptable concentrations toward the return end. The same difficulty may arise to a lesser extent when the U-tube principle is applied to a retreating face, the abandoned airways being stopped off as the face retreats.

Figure 4.6(c) shows a single entry longwall with the back (or bleeder) return held open in order to constrain the gas fringe safely back in the waste area and, hence, prevent flushes of waste gas onto the face. The system illustrated in (c) is a combination of U-tube and through-flow ventilation.

Figure 4.6(d) illustrates the longwall system more often used in coal mining countries that have a tradition of room and pillar mining such as the United States, Australia or South Africa. Two or more entries are driven initially using room and pillar mining, these serving as the lateral boundaries of retreat longwall panels. Again, back bleeders are used to control waste gas.

Figures 4.6(e), (f) and (g) illustrate a classification of systems for longwall faces where the make of gas from the face itself is particularly heavy. The Y system provides an additional feed of fresh air at the return end of the face. This helps to maintain gas concentrations at safe levels along the back return airway(s). Figure 4.5(d) is, in fact, a double entry through-flow Y system. The double-Z layout is also a through-flow system and effectively halves the length of face ventilated by each airstream. The W system accomplishes the same end but is based on the U-tube principle. Both the double-Z and W systems may be applied to advancing or retreating faces, depending upon the ability of the centre return to withstand front abutment and waste area strata stresses. Again, in both the double-Z and W systems, the directions of airflow may be reversed to give a single intake and two returns (or two sets of multiple returns). This may be preferred if heavy emissions of gas are experienced from solid rib sides.

Room (bord) and pillar systems

Figure 4.7 shows two methods of ventilating a room and pillar development panel; (a) a bidirectional or W system in which intake air passes through one or more central airways with return airways on both sides, and (b) a unidirectional or U-tube system with intakes and returns on opposite sides of the panel. In both cases the conveyor is shown to occupy the central roadway with a brattice curtain to regulate the airflow through it. It is still common practice in room and pillar mines to course air around the face ends by means of line brattices pinned to roof and floor but hung loosely in the cross-cuts to allow the passage of equipment. An advantage of the bidirectional system is that the air splits at the end of the panel with each airstream ventilating the operational rooms sequentially over one half of the panel only. Conversely, in the unidirectional or U-tube system the air flows in series around all of the faces in turn. A second advantage of the bidirectional system arises from the fact that ribside gas emission is likely to be heavier in the outer airways. This can become the dominant factor in gassy coal seams of relatively high permeability necessitating that the outer airways be returns. In most coal mining countries, legislation requires that gas concentration in intake airways be maintained at very low levels.

Unfortunately, the bidirectional system suffers from one significant disadvantage. The number of stoppings required to be built, and the number of leakage paths created between intakes and returns, are both doubled. In long development panels, the amount of leakage can become excessive allowing insufficient air to reach the last open cross-cuts (Section 4.2.3). In such circumstances, attempts to increase the pressure differential across the outbye ends of the panel exacerbate the leakage and give a disappointing effect at the faces.

The unidirectional system has a higher volumetric efficiency because of the reduced number of leakage paths. However, in both cases, the line brattices in the rooms offer a high resistance to airflow compared with an open cross-cut. This is particularly so in the case of the unidirectional system where the useful airflow is required to pass around all of these high resistance line brattices in series.
The imposition of line brattice resistance at the most inbye areas of a mine ventilation system forces more air to be lost to return airways at all leakage points throughout the entire system. An analogy may be drawn with a leaky hosepipe. If the end of the pipe is unobstructed then water will flow out of it freely and dribble from the leakage points. If, however, the end of the pipe is partially covered then the flow from it will decrease but water will now spurt out of the leakage points.

The problem can be overcome by employing auxiliary fans and ducts either to force air into the rooms or exhaust air from them (Section 4.4). Figure 4.8 illustrates a room and pillar panel equipped with exhausting auxiliary ventilation. With such a system, the fans provide the energy to overcome frictional resistance in the ducts. The effective resistance of the whole face area becomes zero. Smaller pressure differentials are required between intakes and returns for any given face airflows and, hence, there is a greatly reduced loss of air through leakage. The electrical power taken by the auxiliary fans is more than offset by the savings in main fan duties.

A further advantage of employing auxiliary fans is that each room is supplied with its own separate and controllable supply of air. However, the fans must be sized or ducts regulated such that no undesired recirculation occurs.
The choice between auxiliary fan and duct systems and line brattices in room and pillar workings should also take into account the height and width of the airways, the size and required mobility of equipment, the placement of ducts or brattices, the extent of pollution from dust, gas and heat, fan noise, and visibility within the workings.

![Figure 4.8 U-tube room and pillar development panel with auxiliary fans (zero face resistance).](image)

The systems shown on Figure 4.6 for longwalls each have their counterparts in room and pillar mining. An example of a retreating double-Z (through-flow) system applied to a room and pillar section is shown on Figure 4.9. There are, however, significant differences in the ventilation strategy between the two mining methods. The larger number of interconnected airways and higher leakage result in room and pillar layouts having lower resistance to airflow than longwall mines. It follows that room and pillar mines tend to require higher volume flows at lower fan pressures than longwall systems. Similarly, because of the increased number of airways and leakage paths it is particularly important to maintain control of airflow distribution paths as a room and pillar mine develops. It is vital that barrier pillars be left between adjacent panels and to separate the panels from trunk airway routes. Such barriers are important not only to protect the integrity of the mine in case of pillar failure but also to provide ventilation control points and to allow sealing of the panel in cases of emergency or when mining has been completed.
4.3.3. Orebody deposits

Metalliferous orebodies rarely occur in deposits of regular geometry. Zones of mineralization appear naturally in forms varying from tortuous veins to massive irregularly shaped deposits of finely disseminated metal and highly variable concentration. The mining layouts necessarily appear less ordered than those for stratified deposits. Furthermore, the combination of grade variation and fluctuating market prices results in mine development that often seems to be chaotic. The same factors may also necessitate many more stopes or working places than would be usual in a modern coal mine, with perhaps only a fraction of them operating in any one shift. Hence, the ventilation system must be sufficiently flexible to allow airflow to be directed wherever it is needed on a day-by-day basis.

Figure 4.9 A retreating room and pillar district using a through-flow ventilation system
Ventilation networks for metal mines, therefore, tend to be more complex than for stratified deposits and are usually also three dimensional. Figure 4.10 illustrates the ventilation strategy of many metal mines although, again, the actual geometry will vary widely. Air moves in a through-flow manner from a downcast shaft or ramp, across the levels, sublevels and stopes towards return raises, ramps or upcast shaft. Airflow across each of the levels is controlled by regulators or booster fans.

Movement of air from level to level, whether through stopes or by leakage through ore passes or old workings tends to be ascentional in order to utilize natural ventilating effects and to avoid thermally induced and uncontrolled recirculation.

Airflow distribution systems for individual stopes are also subject to great variability depending upon the geometry and grade variations of the orebody. There are, however, certain guiding principles. These are illustrated in Figure 4.11 to 4.13 for three stoping methods. In the majority of cases, where controlled vertical movement of the air is required, stope airflow systems employ ascentional through-flow ventilation. Although auxiliary fans and ducts may be necessary at individual drawpoints, every effort should be made to utilize the mine ventilation system to maintain continuous.
airflow through the main infrastructure of the stope. Series ventilation between stopes should be minimized in order that blasting fumes may be cleared quickly and efficiently.

Figure 4.11 Simple ventilation system for shrinkage or cut-and-fill stopes.

Figure 4.12 Ventilation system for sub-level open stopes
Leakage through ore passes creates a problem in metal mines as the ore passes may often be emptied allowing a direct connection between levels. Airflows emerging from ore passes can also produce unacceptable dust concentrations. Closed ore chutes and instructions to maintain some rock within the passes at all times are both beneficial but are difficult to enforce in the necessarily production oriented activities of an operating mine. The design of the ventilating system and operation of regulators and booster fans should attempt to avoid significant pressure differences across ore passes. Maintaining an orepass at negative pressure by means of a filtered fan/duct arrangement can help to control dust at dumping or draw points. Attrition on the sides of ore passes often enlarges their cross-section and may produce fairly smooth surfaces. When no longer required for rock transportation, such openings may usefully be employed as low resistance ventilation raises.

Figure 4.13 for a block caving operation illustrates another guideline. Wherever practicable, each level or sublevel of a stope should be provided with its own through-flow of air between shafts, ventilation raises or ramps. While vertical leakage paths must be taken into account during planning exercises, maintaining an identifiable circuit on each level facilitates system design, ventilation management and control in case of emergency.
4.4. AUXILIARY SYSTEMS

Auxiliary ventilation refers to the systems that are used to supply air to the working faces of blind headings. Auxiliary ventilation may be classified into three basic types, line brattices, fan and duct systems, and "ductless" air movers. Ideally, auxiliary systems should have no impact on the distribution of airflows around the main ventilation infrastructure, allowing auxiliary ventilation to be planned independently from the full mine ventilation network. Unfortunately, this ideal is not always attained, particularly when line brattices are employed.

4.4.1. Line brattices and duct systems

The use of line brattices was introduced in Section 4.3.2 (Figure 4.7) in relation to room and pillar workings where they are most commonly employed. It was shown that a major disadvantage of line brattices is the resistance they add to the mine ventilation network at the most sensitive (inbye) points, resulting in increased leakage throughout the system. This resistance depends primarily upon the distance of the line brattice from the nearest side of the airway, and the condition of the flow path behind the brattice. This is sometimes obstructed by debris from sloughed sides, indented brattices or, even, items of equipment put out of sight and out of mind, despite legislative prohibitions of such obstructions. In this section we shall examine the further advantages and disadvantages of line brattices.

Figure 4.14 shows line brattices used in the (a) forcing and (b) exhausting modes. The flame-resistant brattice cloth is pinned between roof and floor, and supported by a framework at a position some one quarter to one third of the airway width from the nearest side. This allows access by continuous miners and other equipment. Even with carefully erected line brattices, leakage is high with often less than a third of the air that is available at the last open cross-cut actually reaching the face. This limits the length of heading that can be ventilated by a line brattice. The need for line brattices to be extended across the last "open" cross-cut inhibits visibility creating a hazard where moving vehicles are involved. The advantages of line brattices are that the capital costs are low in the short term, they require no power and produce no noise.
Chapter 4. Subsurface Ventilation Systems.

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Figure 4.15 shows the corresponding forcing and exhausting systems using auxiliary fans and ducting. In most cases, in-line axial fans are used although centrifugal fans are quieter and give higher pressures for the longer headings. The advantages of an auxiliary fan and duct are that they provide a more positive and controlled ventilating effect at the face, they cause no additional resistance to the mine ventilation system nor any consequent leakage throughout the network, and are much less liable to leakage in the heading itself. For headings longer than some 30 metres, auxiliary fans are the only practicable means of producing the required airflows. An exhausting duct also allows the air to be filtered, an advantage for dust control where series ventilation is practiced. The disadvantages involve the initial capital cost, the need for electrical power at the fans, the space required for ducts and the noise produced by the fans.

Care should be taken to ensure that the pressure-volume characteristics of the fan are commensurate with the resistance offered by the duct and the airflow to be passed. The latter is determined on the basis of the type and magnitude of pollutants to be removed (Chapter 9). The duct resistance is established as a combination of the wall losses within the duct, shock losses at any bend or change of cross section and at discharge. The equations employed are those derived for airway resistance in Section 5.4.

Example.
An airflow of 15 m³/s is to be passed through a 0.9m diameter fibreglass duct, 200m long, with one sharp right-angled bend. From manufacturer's literature, the friction factor for the duct is 0.0032 kg/m³. Calculate the total pressure to be developed by the fan and the fan power, assuming a fan efficiency of 60 per cent and an air density of 1.2 kg/m³.

Solution.

Duct area \( A = \pi \times 0.9^2 / 4 = 0.636 \ m^2 \)
Perimeter \( per = \pi \times 0.9 = 2.827 \ m \)

Let us first determine the shock loss \( X \) factors for the system from Appendix A5 (Chapter 5).

Entry: (Section A5.4) In the absence of any inlet fitting, the shock loss factor is given as \( X_{in} = 1.0 \). This is caused by turbulence as the air enters the duct and should not be confused with the conversion of static pressure to velocity pressure at entry.

Bend: (Figure A5.1) For a sharp right-angled bend, \( X_b = 1.2 \)

Exit: (Section A5.4) This is not really a shock loss but represents the kinetic energy of the air provided by the fan and lost to the receiving atmosphere. \( X_{ex} = 1.0 \)

Total shock loss factor: \( X_{sh} = 1.0 + 1.2 + 1.0 = 3.2 \)

Equivalent resistance of shock losses:
\[
R_{sh} = \frac{X_{sh} \rho}{2A^2} = \frac{3.2 \times 1.2}{2 \times (0.636)^2} = 4.744 \ m^8/Ns^2
\]

Duct resistance:
\[
R_d = \frac{kLper}{A^3} = \frac{Ns^2}{m^8} \quad \text{(see equation (5.4))}
\]
4.4.2. Forcing, exhausting, and overlap systems.

Figures 4.14 and 4.15 illustrate forcing and exhausting systems of auxiliary ventilation for line brattices and fan/duct systems respectively. The choice between forcing and exhausting arrangements depends mainly upon the pollutants of greatest concern, dust, gases or heat.

The higher velocity airstream emerging from the face-end of a forcing duct or, to a lesser extent, a forcing brattice gives a scouring effect as the air sweeps across the face. This assists in the turbulent mixing of any methane that may be emitted from fragmented rock or newly exposed surfaces. It also helps to prevent the formation of methane layers at roof level (Chapter 12). In hot mines, the forcing system provides cooler air at the face, even having taken the energy added by the fan into account. Furthermore, as the system is under positive gauge pressure, the cheaper type of flexible ducting may be used. This is also easier to transport and enables leaks to be detected more readily.

The major disadvantage of a forcing system is that pollutants added to the air at the face affect the full length of the heading as the air passes back, relatively slowly, along it.

Where dust is the main hazard, an exhausting system is preferred. The polluted air is drawn directly into the duct at the face-end allowing fresh air to flow through the length of the heading. However, the lack of a jet effect results in poor mixing of the air. Indeed, unless the end of the duct or brattice line is maintained close to the face then local pockets of sluggish and uncontrolled recirculation may occur. In all cases, it is important that the ducting or brattice line be extended regularly so that it remains within some three metres of the face. This distance may be prescribed by legislation.

A further advantage of a ducted exhaust system is that a dust filter may be included within the system. In this case, the additional pressure drop across the filter must be taken into account in choosing the fan, and the filter serviced regularly in order that its resistance does not become excessive. Exhaust ducts must necessarily employ the more expensive rigid ducting or reinforced flexible ducting. If the exit velocity from an exhaust duct is high then an induction effect (Section 4.3.3), akin to a low pressure booster fan, can result in unexpected consequences including recirculation in the main entries.
For long headings, the resistance of the duct may become so great that multiple fans connected in series must be employed. If these are grouped as a cluster at the outbye end of the ducting then the high (positive or negative) gauge pressure will exacerbate leakage. It is preferable to space the fans along the length of the ducting in order to avoid excessive gauge pressures. The use of hydraulic gradient diagrams assists in the optimum location of fans and to prevent uncontrolled recirculation of leakage air. Multiple fans must be interlinked electrically and airflow or pressure monitors employed to detect accidental severing or blockage of the duct, in which case all fans inbye the point of damage must be switched off - again, to prevent uncontrolled recirculation.

It is clear that forcing and exhausting systems both have their advantages and disadvantages. Two-way systems have been devised that can be switched from forcing mode to exhausting for cyclic mining operations. These may employ a reversible axial fan or, alternatively, both a forcing and an exhausting fan, only one of which is operated at any one time with an appropriate adjustment of valves or shutter doors within the duct arrangement.

The more common methods of combining the advantages of forcing and exhausting ducts are overlap systems. Examples are shown on Figure 4.16. The direction and mean velocity of the air in the heading within the overlap zone clearly depends upon the airflows in each of the ducts. These should be designed such that the general body airflow in this region does not become unacceptably low. Where permitted by law, controlled recirculation may be used to advantage in overlap systems (Section 4.5.2). Where continuous miners or tunnelling machines are employed, the overlap ventilator may be mounted on the machine. In all cases, it is important that the fans are interlinked so that the overlap system cannot operate when the primary duct fan is switched off.

(a) Forcing system with exhaust overlap  (b) Exhausting system with force overlap

Figure 4.16 Overlap systems of auxiliary ventilation
4.4.3. Air movers

In addition to conventional ducted systems of auxiliary ventilation, a number of other techniques may be employed to enhance or control the movement of air within localized areas of a mine or tunnel.

Jet fans, sometimes known as ductless, vortex or induction fans are free standing units that produce a relatively high velocity outlet airstream. The jet of air produces two effects. First, the reach and integrity of the air vortex depends upon the velocity at the fan outlet, the size of the heading and whether the airway is a blind heading or part of a throughflow system. Satisfactory ventilation and turbulent mixing at the face of a large heading can be obtained from a jet fan sited 100 m outbye. Secondly, an induction effect occurs at the outer boundaries of the expanding cone of projected air. This entrains additional air from the surroundings producing a forward moving flow that is greater than the airflow through the fan itself. The conversion of velocity pressure into static pressure as the plume of air decelerates also generates a true ventilating pressure. This is seldom greater than some 20 Pascals but is sufficient to create significant airflows in large, low resistance airways. The induction effect inhibits excessive recirculation provided that incoming intake air is provided at the fan inlet. Jet fans have particular application in large room and pillar operations and may also be used in series to promote airflow through vehicular tunnels.

An airflow can also be generated by a spray of water giving rise to spray fans. Inertia from the motion of the water droplets is transmitted to the air by viscous drag and turbulent induction. Spray fans may be used very effectively to control the local movement of air around rock-winning machines such as continuous miners or longwall shearsers. This assists in the rapid dilution of methane and in diverting dust-laden air away from operator positions. The effect depends upon the shape, velocity and fineness of the spray. Although dust suppression sprays also cause air induction it is usually necessary to add additional sprays if these are to be used for local airflow control. Provided that the service water is chilled, spray fans are also an efficient means of cooling the air in a work area.

Compressed air injectors are also induction devices. The compressed air is supplied through one or more forward pointing jets within a cylindrical or shaped tube. The best effect is obtained when the compressed air is supplied at the throat of a venturi. These devices are noisy and of much lower efficiency than fans. However, they have a role in areas where electrical power is unavailable. Mining law may proscribe the use of compressed air for the promotion of airflow in gassy mines as the high velocity flow of air through the jets can cause the build-up of an electrostatic charge at the nozzle. This produces the possibility of sparks that could ignite a methane-air mixture.

4.5. CONTROLLED PARTIAL RECIRCULATION

4.5.1. Background and principles of controlled partial recirculation

The idea of recirculating air in any part of a gassy mine has, traditionally, been an anathema to many mining engineers. Most legislation governing coal mines prohibits any ventilation system or device that causes air to recirculate. The background to such legislation is the intuitive fear that recirculation will cause concentrations of pollutants to rise to dangerous levels. A rational examination of controlled recirculation was carried out by Leach, Slack and Bakke during the 1960's at the Safety in Mines Research Establishment in England. Those investigators made a very simple and obvious statement but one that had, to that time, apparently been denied or ignored within the context of air recirculation. They argued that the general body gas concentration, \( C \), leaving any ventilated region of a mine is given by

\[
C = \frac{\text{Flow of gas into the region}}{\text{Flow of fresh air passing through the region}}
\]  

(4.3)
The value of C is quite independent of the flowpaths of the air within the region, including recirculation. It is true, of course, that if the through-flow of fresh air falls while the gas emission remains constant then the concentration of gas will rise. This would happen, for example, if the air duct serving a long gassy heading was dislocated while an inbye fan continued to run. This example illustrates a case of uncontrolled recirculation. The definition of a system of controlled partial recirculation is one in which a controlled fraction of the air returning from a work area is passed back into the intake while, at the same time, the volume flow of air passing through the region is monitored to ensure that it remains greater than a predetermined minimum value.

The advantages of controlled partial recirculation lie in the improved environmental conditions it can provide with respect to gases, dust and heat, as well as allowing mining to proceed in areas of a mine that are too distant from surface connections to be ventilated economically by conventional means.

As illustrated in Section 4.5.2., general body gas concentrations may actually be reduced by controlled recirculation. Furthermore, the higher air velocities that occur within a recirculation zone assist in the turbulent mixing of gas emissions, reducing the tendency to methane layering and diminishing the probability of accumulations of explosive methane-air mixtures.

As with gas, concentrations of respirable dust reach predictable maximum levels in a system of controlled recirculation and may be reduced significantly by the use of filters. The greater volume of air being filtered results in more dust being removed. The effect of controlled recirculation on climatic conditions is more difficult to predict. However, both simulation programming and practical observations have indicated the improvements in the cooling power of partially recirculated air for any given value of through-flow ventilation.

As workings proceed further away from shaft bottoms, the cost of passing air along the lengthening primary intakes and returns necessarily increases. Where new surface connections closer to the workings are impractical, perhaps because the workings lie beneath the sea or due to great depth then the cost of conventional ventilation will eventually become prohibitive - even when booster fans are employed. Using the air more efficiently through controlled partial recirculation then becomes an attractive proposition. If, for example, the methane concentration returning from a conventionally ventilated face is 0.3 per cent, and the safe mandatory limit is 1.0 per cent, then the through-flow provided from the main airways might be reduced to one half giving a methane concentration of 0.6 per cent while maintaining or increasing the face velocities by controlled recirculation.

During the 1970's the concept of controlled partial recirculation gained respectability and is now practiced by several of the world's mining industries operating, in some cases, by authorized exemptions from existing legislation.

The greatest disincentive against the introduction of controlled recirculation has been the risk of combustion gases from a fire being returned to working areas. Further potential problems arise from a consideration of transient phenomena such as blasting, or rapid changes in barometric pressure caused by the operation of doors or fans, and the possible resulting peak emissions of gases. The introduction of failsafe monitoring systems with continuous computer surveillance has revolutionized the situation (Section 9.6.3.). These self-checking systems involve monitoring the concentration of gases, air pressure differentials and airflows at strategic locations as well as the operating conditions of fans and other plant. Fans can be interlinked electrically to obviate the possibility of uncontrolled recirculation. Should any monitored parameter fall outside prescribed limits then the system will automatically revert to a conventional non-recirculating circuit. The introduction of reliable monitoring technology has allowed the advantages of controlled partial recirculation to be realized safely.

4.5.2. Controlled recirculation in headings

The most widespread application of controlled recirculation has been in headings. One of the disadvantages of the conventional overlap systems shown in Figure 4.16 is the reduction in general
body air velocity within the overlap zone. This can be overcome completely by arranging for the overlap fan to pass an airflow that is greater than that available within the heading, i.e. a system of controlled recirculation, accompanied by the corresponding monitoring system and electrical interlocks. This is particularly advantageous when applied to the scheme depicted in Figure 4.16(a) as filtered air is then available to machine operators as well as throughout the length of the heading.

Figure 4.17 shows two examples of a primary exhaust system configured for controlled recirculation. In both cases, an airflow \( Q_t \) (m\(^3\)/s) is available at the last open cross-cut and contains a gas flow of \( G_i \) (m\(^3\)/s). An airflow of \( Q_h \) passes up the heading where a gas emission of \( G_h \) is added.

Let us try to find the maximum general body gas concentrations that will occur in the systems. Referring to Figure 4.17(a) and using the locations A, B, C and D as identifying subscripts, the fractional gas concentration, \( C_{gD} \), at position D (leaving the system) must be

\[
C_{gD} = \frac{G_i + G_h}{Q_t} \quad \text{(see equation (4.3))}
\]

(In these relationships, it is assumed that the gas flow is much smaller than the airflow). However, inspection of the figure shows that this must also be the gas concentration at locations A and B. In particular,

\[
C_{gB} = \frac{G_i + G_h}{Q_t}
\]
But gas flow, $G = \text{Gas concentration} \times \text{airflow}$

\[
G_B = \frac{(G_i + G_h)}{Q_t} \times Q_h
\]

The gas flow in the duct, $G_C = G_B + G_h$, must then be

\[
G_C = \frac{(G_i + G_h)}{Q_t} \times Q_h + G_h
\]

Hence, the gas concentration in the duct,

\[
C_{gC} = \frac{G_C}{Q_h} = \frac{G_i + G_h}{Q_t} + \frac{G_h}{Q_h}
\]  \hspace{1cm} (4.4)

This is the highest general body gas concentration that can occur anywhere within the system shown on Figure 4.17(a).

Examination of equation (4.4) shows that if the gas flows, $G_i$ and $G_h$, are fixed and the fresh air supply, $Q_t$, remains unchanged then the maximum general body gas concentration, $C_{gC}$, must fall as $Q_h$ is increased - that is, as the degree of recirculation rises. In the limit, at very high $Q_h$, the gas concentration in the duct tends toward that leaving the system at position D.

In a conventional non-recirculating system, the airflow taken into a heading is often limited to no more than half of that available at the last open cross-cut, i.e. $Q_h = 0.5Q_t$. Applying these conditions to equation (4.4) gives

\[
C_{gC} \text{ (conventional)} = \frac{(G_i + G_h)}{Q_t} + \frac{G_h}{0.5Q_t}
\]

\[
= \frac{G_i + 3G_h}{Q_t}
\]  \hspace{1cm} (4.5)

Recirculation commences when $Q_h = Q_t$, giving

\[
G_{gC} \text{ (maximum, recirculating)} = \frac{(G_i + G_h)}{Q_t} + \frac{G_h}{Q_t}
\]

\[
= \frac{G_i + 2G_h}{Q_t}
\]  \hspace{1cm} (4.6)

$C_{gC}$ must be less than this at all greater values of $Q_h$, i.e. higher degrees of recirculation. Comparing equations (4.5) and (4.6) shows that in this configuration the maximum general body gas concentration is always less using controlled recirculation than with a conventional system. Turning to Figure 4.17(b), the analysis is even simpler. In this case, the maximum gas concentration (in the duct) must be the same as that leaving the system $(G_i + G_h)/Q_t$ provided that $Q_h$ is equal to or greater than $Q_t$, i.e. controlled recirculation must exist. Here again, this is always less than would be attainable with a conventional non-recirculating system.
A similar analysis for dust concentration, \( C_d \), on the system shown on Figure 4.17 but with no dust filter gives an analogous expression to that for gas.

\[
C_{dC} = \frac{D_i + D_h}{Q_t} + \frac{D_h}{Q_h} \quad \text{mg} \text{m}^{-3} \quad (4.7)
\]

Again it can be seen that the maximum concentration falls as \( Q_h \) is increased. For dust it is more pertinent to state the concentration at position B, i.e. in the main length of the heading: This becomes

\[
C_{dB} = \frac{D_i + D_h}{Q_t} \quad \text{mg} \text{m}^{-3} \quad (4.8)
\]

and is completely independent of the degree of recirculation.

If the filters shown in Figure 4.17(a) remove a fraction, \( \eta \), of the dust in the duct then it can be shown that the corresponding concentrations become

\[
C_{dC} = \frac{Q_h(D_i + D_h) + D_h Q_t}{Q_h(Q_t + \eta Q_h)} \quad \text{mg} \text{m}^{-3} \quad (4.9)
\]

and

\[
C_{dB} = \frac{D_i + (1 - \eta) D_h}{Q_t + \eta Q_h} \quad \text{mg} \text{m}^{-3} \quad (4.10)
\]

These equations show that when filters are used, dust concentrations fall throughout the system. It has been assumed in these analyses that there is no settlement of dust.

Similar relationships can be derived for other configurations of controlled recirculation in headings.

### 4.5.3. District systems

The extension of controlled partial recirculation to complete areas of a mine has particular benefits in decreasing the costs of heating or cooling the air and for workings distant from the surface connections.

**Positions of fans**

Figure 4.18 shows simplified schematics illustrating three configurations of fan locations in a district recirculation system. In each case, the throughflow ventilation in the mains is shown as \( Q_m \) with \( Q_c \) passing from return to intake in the recirculation cross-cut, to give an enhanced airflow of \( Q_m + Q_c \) in the workings. The ratio \( F = Q_c/(Q_m + Q_c) \) is known as the recirculation fraction. The fan that creates the recirculation develops a pressure of \( p_r \) while the pressure differentials applied across the outbye ends for the three systems shown are \( p_{o1} \), \( p_{o2} \) and \( p_{o3} \) respectively.

The simplest configuration is shown in Figure 4.18(a) with the recirculating fan sited in the cross-cut. This maintains the intakes and returns free for travel and unobstructed by airlocks. Locating the fan in this position will tend to decrease the throughflow, \( Q_m \). Hence, if the total flow is to be maintained, the applied pressure differential must be increased from

\[
p_{o1} = R_m Q_m^2 + R_w Q_m^2 \quad \text{(with no recirculation)}
\]
where $R_m$ is the combined resistance of the intake and return mains
to

$$p_{o1} = R_m \frac{Q_m^2}{Q_m + Q_c} + R_w (Q_m + Q_c)^2$$

with the recirculation shown on Figure 4.18(a). These equations are derived by summing the frictional pressure drops, $p$, around the path of the mains (subscript $m$) and workings (subscript $w$) - and by applying the square law $p = RQ^2$, where $R$ = airway resistance (Section 5.2).

Similarly, in all three cases (a), (b) and (c) the pressure required of the recirculating fan is given by summing the frictional pressure drops around the workings and cross-cut

$$p_r = R_c \frac{Q_c^2}{Q_m + Q_c} + R_w (Q_m + Q_c)^2$$

In system (b), the fan is located within either the intake or return inby the recirculation cross-cut. In this position, the fan acts as a district booster fan as well as creating the controlled recirculation. Hence, the throughflow, $Q_m$, will tend to increase. Alternatively, if the throughflow is to remain constant then either the outby pressure differential may be reduced to

$$p_{o2} = R_m \frac{Q_m^2}{Q_m + Q_c} - R_c \frac{Q_c^2}{Q_m + Q_c}$$

or a regulator can be introduced into either of the mains.
System (c) in Figure 4.18 combines a booster fan with a cross-cut fan and is the preferred configuration where recirculation is employed due to the workings being distant from the surface connections. In this system, the ventilating pressure applied across the district may be reduced by the magnitude of the booster fan pressure to maintain a constant \( Q_m \), i.e.

\[
\rho_{o3} = R_m Q_m^2 + R_w (Q_m + Q_c)^2 - \rho_b
\]

(4.14)

The total airpower consumed within the systems is given as the sum of the \( pQ \) products for all airways. If corresponding airflows and the resistances are the same in each of the three systems then the required total airpowers must also be equal, irrespective of the locations of the fans. In practice, differences in the efficiencies of the fans will cause variations in the required total electrical input to the fan motors. This may, however, be of minor significance.

A major consideration in all designs of controlled partial recirculation is that in conditions of an emergency or plant stoppage, the system must fail-safe and revert to a conventional non-recirculating configuration. The airflows must then remain sufficient to allow personnel to evacuate the area safely and for the necessary ameliorative measures to be taken. The detection of such conditions is provided by monitoring the environmental parameters and the operation of the fans.

If the recirculating cross-cut fan in system (a) fails, then doors in the cross-cut must close automatically. The throughflow ventilation will increase, reducing the general body gas concentration in the return airways. However, the reduced air velocity in the working area will increase the probability of local accumulations of gas to that of a conventional non-recirculating system.

Stoppage of the in-line fan of system (b) is more serious. Again, doors in the cross-cut must be closed but the throughflow of air will decrease, resulting both in diminished airflows in the workings and also higher general body gas concentrations in the returns. However, this system is capable of better airflow control than the cross-cut fan. The degree of recirculation may be varied by modifying the duty of the fan installation (fan speed, vane settings or number of fans operating), regulating the airflow in either the intake or return inbye the cross-cut or by adjustment of a bypass path around the fan.

System (c) gives the greatest degree of flexibility. Stoppage of the cross-cut fan and closure of the corresponding doors will increase the throughflow, \( Q_m \). However, should the booster fan fail, then electrical interlocks should close down the cross-cut fan. Reduced airflows throughout the system are maintained by the outbye pressure differential. Adjustment of the two fans allows a much greater degree of independent control of the airflow distribution than either of the systems (a) or (b).

**Pollution levels**

Although the general body gas concentration leaving any zone is independent of airflow distribution within the zone - recirculating or otherwise (Section 4.5.1), any airflow passed from a return to an intake airway may affect the quality as well as the quantity of the air in that intake.

Referring to Figure 4.18(b) suppose that the incoming intake air contains a gas flow of \( G_i \) (m³/s) and a constant gas emission of \( G_w \) occurs in the workings. Let us derive expressions for the general body gas concentrations in the face return (position 2) and intake (position 1).

The return concentration must be the same as that leaving the complete district and in the cross-cut, assuming no other sources of gas emission:

\[ C_{g2} = G_{gC} = C_g \text{(main return)} = \frac{G_i + G_w}{Q_m} \]  
(4.15)

(see equation (4.3))
This is independent of the degree of recirculation.

To determine the gas concentration in the intake at position 1, consider first the gas flow passing through the cross-cut (subscript c)

\[ G_c = Q_c \times C_{gC} \]

\[ = Q_c \left( \frac{G_i + G_w}{Q_m} \right) \quad \text{from equation (4.15)} \]

Now the gas flow at position 1 is

\[ G_1 = G_i + G_c \]

\[ = G_i + \frac{Q_c}{Q_m} (G_i + G_w) \]

The corresponding concentration is given by dividing by the corresponding airflow, \(Q_m + Q_c\), giving

\[ C_{g1} = \frac{G_1}{(Q_m + Q_c)} = \frac{1}{Q_m} \left( \frac{G_i Q_m}{(Q_m + Q_c)} + \frac{Q_c}{(Q_m + Q_c)} (G_i + G_w) \right) \]  
(4.16)

However, if we define the recirculation fraction as

\[ F = \frac{Q_c}{(Q_m + Q_c)} \quad \text{where, also,} \quad 1 - F = \frac{Q_m}{(Q_m + Q_c)} \]

Then equation (4.16) becomes

\[ C_{g1} = \frac{1}{Q_m} \left[ (1 - F) G_i + F (G_i + G_w) \right] \]

\[ = \frac{1}{Q_m} \left[ G_i + F G_w \right] \]  
(4.17)

This verifies the intuitive expectation that as the degree of recirculation, \(F\), increases then the gas concentration in the intake also increases. However, as \(F\) is never greater than 1, and comparing with equation (4.15), we can see that the intake, or face, gas concentration can never be greater than the return concentration. Hence, in a district recirculation system the general body gas concentration at no place is greater than the return general body concentration with or without recirculation. The maximum allowable methane concentrations in coal mine intakes may be prescribed by law at a low value such as 0.25 per cent. The value of \(F\) should be chosen such that this limit is not exceeded. Similar analyses may be carried out for dust concentrations. In this case, drop-out and the use of filters can result in significant reductions in the concentrations of dust in a system of controlled partial recirculation. However, the enhanced air velocities in the work area should not exceed some 4 m/s as the re-entrainment of settled particles within the airstream accelerates rapidly at greater velocities.
The climatic conditions within a system of controlled partial recirculation depend not only upon the airflows and positions/duties of the fans but also upon the highly interactive nature of heat transfer between the strata and the ventilating airstreams (Chapter 15). The locations, types and powers of other mechanized equipment, and the presence of free water also have significant effects. The only practicable means of handling the large numbers of variables is through a computer program to simulate the interacting physical processes (Chapter 16). Such analyses, together with practical observations, indicate that wet and dry bulb temperatures at any point may either increase or decrease when controlled partial recirculation is initiated without air cooling. The increased air velocities within the recirculation zone enhance the cooling power of the air on the human body for any given temperature and humidity. However, when controlled recirculation is practiced in hot mines, it is normally accompanied by cooling of the recirculated air. This is less expensive and more effective than bulk cooling the intake air in a conventional ventilation system and significant improvements in climatic conditions may be realized. Again, practical experience has shown that the higher airflows within a recirculation zone improves the effectiveness of existing refrigeration capacity.

In closing this section the reader is reminded, once again, that air recirculation may be prohibited by the governing legislation. The relevant statutes should be read, and/or enforcement agencies consulted before instituting a system of controlled partial recirculation.

4.6. UNDERGROUND REPOSITORIES

4.6.1. Types of repository

Underground space is increasingly being utilized for purposes other than the extraction of minerals or for transportation. The high cost of land, overcrowding and aesthetic considerations within urban areas encourages use of the subsurface for office accommodation, manufacturing, warehousing, entertainment facilities and many other purposes. The safety and stability of a well chosen geologic formation makes underground space particularly suitable for the storage of materials, varying from foodstuffs and liquid or gaseous fuels to toxic wastes. The design and operation of environmental systems in such repositories require the combined skills of mine ventilation engineers and HVAC (heating, ventilating and air-conditioning) personnel. The repositories must be constructed and operated in a manner that preserves the integrity of the stored material and also protects the public from hazardous emissions or effluents.

Perhaps the most demanding designs arise out of the perceived need to store radioactive waste in deep underground repositories. There are basically two types of this waste. First, there is the transuranic, or low level, radioactive waste such as contaminated clothing, cleaning materials or other consumable items that are produced routinely by establishments that handle radioactive materials. Such waste may be compressed into containers which may be stacked within excavated chambers underground. Secondly, there is the concentrated and highly radioactive waste produced from some defence establishments and as the plutonium-rich spent fuel rods from nuclear power stations. This waste may be packed into heavily shielded and corrosion resistant cylinders and emplaced within boreholes, about one metre in diameter, drilled from underground airways into the surrounding rock.

4.6.2. Ventilation circuits in repositories for nuclear waste

Figure 4.19 depicts the primary ventilation structure of a high level nuclear waste repository. As in the figures illustrating mining circuits shown earlier in the chapter, this sketch is conceptual in nature and is not intended to represent all airways.
During the operation of an underground repository two activities must proceed in phase with each other. One is the mining of the rooms or drifts where the material is to be placed, together with the excavation of transportation routes, ventilation airways and the other entire infrastructure required in an underground facility. This is referred to simply as the mining activity. Secondly, the hazardous waste material must be transported through the relevant shafts and airways to the selected rooms for emplacement. Accordingly, this is known as the emplacement activity.

For reasons of environmental safety, the ventilation circuits for mining and emplacement activities in a nuclear waste repository must be kept separate. Furthermore, any leakage of air through doors or bulkheads between the two systems must always leave the mining zone and flow into the emplacement zone - even in the event of the failure of any fan. Figure 4.19 shows how this is achieved. The mining circuit operates as a through flow forcing system with the main fan(s) sited at the top of the mining downcast shaft(s). On the other hand, the emplacement circuit operates as a throughflow exhaust system with the main fans located at the top of the waste upcast shafts. It should be remembered that within the nomenclature of underground repositories, the term waste refers to the hazardous waste to be emplaced and not waste rock produced by mining activities. With this design, any accidental release of radionuclides into the underground atmosphere is contained completely within the emplacement circuit and will not contaminate the mining zones.

Figure 4.19 Example of primary ventilation circuits for an underground nuclear waste repository.
The shaft or surface-connecting ramp used for transporting the nuclear waste underground is not shown on Figure 4.19. This shaft will normally not form part of the main ventilation system but will have a limited downcasting airflow which passes directly into a waste main return. Similar arrangements may be made for the waste transportation routes underground, thus limiting the potential dispersion of radioactive contamination in the event of a waste container being damaged during transportation. Separate maintenance and repair shops are provided in the mining and emplacement circuits.

When emplacement activities have been completed in any given room then the ends of that room may be sealed. In the case of high level nuclear waste, this may result in the envelope of rock surrounding the airway reaching temperatures in excess of 150 OC, depending upon the rate of heat emission from the waste, the distance of the canisters from the airway, the thermal properties of the rock, and thermal induction of water and vapour migration within the strata. If the drift is to be reopened for retrieval or inspection of any canister then a considerable period of cooldown by refrigerated air may be required before unprotected personnel can re-enter. To reduce the time and expense of the cooldown period, the emplaced room may not be completely sealed but allowed to pass a regulated airflow sufficient to maintain the rock surface temperature at a controlled level.

4.6.3. Additional safety features

Before any repository for hazardous waste is commissioned it must conform to the strictest standards of safety and quality assurance in order to protect both the workers and the general public from chemical or radioactive contamination. In the case of an underground repository, design safeguards commence with an extensive examination of the suitability of the geologic formation to act as a natural containment medium. This will involve the physical and chemical properties of the rock, the presence and natural migration rate of groundwater and the probability of seismic activity. Other factors that influence the choice of site include population density and public acceptance of the surface transportation of hazardous waste to the site.

In addition to continuous electronic surveillance of the quality of the atmosphere throughout the main ventilation routes of a nuclear waste repository, the fans, bulkheads and regulators must be monitored to ensure that they operate within design limits and that pressure differentials are maintained in the correct direction at all times. Regulators and doors may be fitted with electrical or pneumatic actuators suitable for both local and remote operation. Should airborne radioactivity be detected at any time and any location in the circuit then the air emerging from the top of the waste upcast shafts is diverted automatically through banks of high efficiency particulate (HEPA) filters. Separate or additional precautions should also be taken to protect surface buildings and shaft tops from tornadoes, floods or fall-out from volcanic activity.

Bibliography


