7 Ventilation

To inquire into . . .

(d) whether there was any defect in or about the Mine or the modes of working the Mine;

(f) whether there was compliance with applicable statutes, regulations, orders, rules, or directions

The ventilation system of any underground mine is an arterial network of interconnected roadways that are also used as transportation routes for personnel and vehicles and the products of mining. Fresh air is drawn from the surface atmosphere. As the air passes through the underground passages, its quality deteriorates as a result of pollutants produced from the strata and from the effects of machines and mining procedures. The contaminated air is returned to the surface, where it is rejected to the outside atmosphere.

A mine ventilation system has to deal with both gaseous and particulate (dust) pollutants. All mines produce dusts that may lead to long-term health problems for mine workers. Many mines are subject to emissions of gases from the strata. Diesel equipment, increasingly used in underground mines, produces a variety of gases and other emissions that can have adverse physiological effects. Some mines require temperature and humidity to be controlled so that personnel may perform their duties safely and without undue discomfort. *The primary objective of any mine ventilation system is to provide breathable airflows in sufficient quantity and quality to dilute airborne pollutants to safe concentrations in all areas where personnel are required to work or travel.*

Methane is the most prevalent strata gas in underground coal mines. Although non-toxic, methane is hazardous because of its flammability. It will explode when in concentrations of between 5 and about 15 per cent by volume in air, and it reaches maximum explosibility at about 9.6 per cent. The gas is emitted from the coal seams and, sometimes, adjacent strata when those formations are disturbed by mining activities.

The second most dangerous pollutant routinely present in coal mines is coal dust. When exposed to significant concentrations of such dust over a number of years, miners may develop coal workers' pneumoconiosis, known also as black lung disease, a debilitating reduction in lung function that can lead to serious heart disorders. Coal dust, like most finely divided organic materials, is also explosive when suspended in air.

Protection of the health and safety of personnel is at the foundation of mining legislation in all jurisdictions that have such laws. Additionally, experience, prudent regard for safety, and the need to safeguard the continuity of mineral production have resulted in guidelines and procedures for designing, planning, and maintaining effective systems of mine ventilation.

Ventilation in Underground Coal Mines[1]

Every underground mine has at least two systems of ventilation. The first, made up of the *main* structure, consists of a network of interconnected airways (also known as entries or openings) along which passes the throughflow ventilation. The movement of air is maintained by the main fan (or fans) and any booster fans that may be used to help promote that airflow.[2] As mining advances into previously unworked areas, there will inevitably be blind headings that cannot be part of the main throughflow system. Ventilation of those headings is accomplished by the second system – local, or *auxiliary*, ventilation.

The Structure of a Mine Ventilation System

Air enters the mine ventilation system by being drawn from the surface atmosphere into one or more vertical shafts, slopes, or level adits. The air flows through passages known as *intake* airways until it reaches the active work areas where the mineral is being mined. This is where most of the airborne contaminants are added. The air then proceeds along *return* airways until it re-enters the surface atmosphere.

Main Forcing and Main Exhaust Systems

In the vast majority of modern mines, the movement of air through the ventilation system is maintained by one or more main fans. In Nova Scotia, as in most jurisdictions, legislation requires that the main fans at coal mines be located on the surface.[3] This requirement is a precaution against damage to those fans in the event of an underground emergency condition. The main fans may blow air into the intake airway to form a *main forcing* (or blowing) system, or they may draw air from the return airway to form a *main exhaust* system. The two systems are illustrated in figure 7.1.

Larger mines may be equipped with main fans at more than one surface connection and operating in either the forcing or the exhaust mode. In a few mines, both forcing and exhaust fans may be used to provide a "push-pull" system.

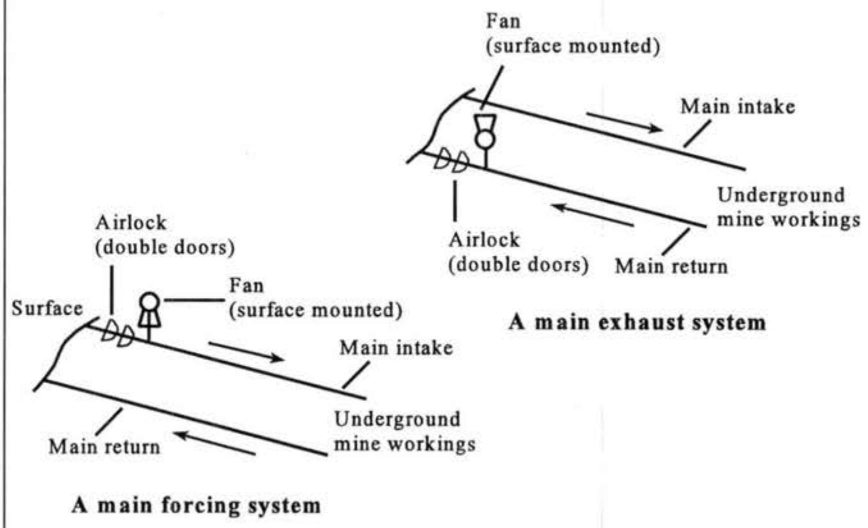
---

1 The Inquiry is indebted to Dr Malcolm J. McPherson, mining engineer and coal mine ventilation expert, for his assistance and advice in the preparation of Chapters 7, 8, and 9 of this Report relating to ventilation, methane, and coal dust. His testimony at the hearings, as well as his review and interpretation of other evidence, has been essential to an understanding of the complex interrelationships among these factors and the way they contributed to creating the explosive environment that led to the disaster of 9 May 1992. A more technical treatment of these subjects is found in McPherson's book *Subsurface Ventilation and Environmental Engineering* (London: Chapman and Hall, 1993) and in the papers he prepared for the Inquiry.

2 There were no booster fans in the Westray mine. In the United States, there is a general prohibition on the use of booster fans. (The United States seems to regard the requirement for booster fans as evidence of poor ventilation planning.) The booster fan is quite acceptable in many UK and other European mines, however, probably because these are older mines that have extended their mining areas beyond the capacities of the original main fans.

3 *Coal Mines Regulation Act*, RSNS 1989, c. 73, s. 71(3).

Figure 7.1 Main Ventilation for Underground Mines



Source: Prepared by Malcolm J. McPherson for the Westray Mine Public Inquiry.

A main exhaust system is the more common ventilation design in coal mines. This system allows intake airways to remain unencumbered by airlocks or ventilation control doors, thus facilitating transportation along the intake routes. More importantly, however, should a main exhaust fan stop, the atmospheric (barometric) pressure will rise throughout the mine and temporarily inhibit the release of gases from old workings or other zones where such gases may have collected. It is inadvisable to design a system in which belt conveyors pass through airlocks, because increased leakage and dust dissemination occur at such points. Where there is good reason to locate belt conveyors in return airways, a main forcing system may be preferred to a main exhaust one.

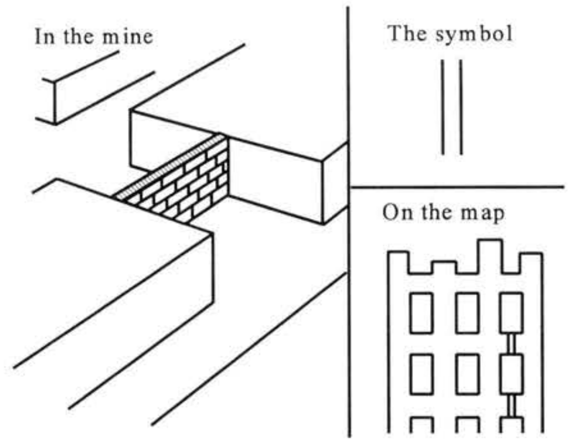
Ventilation Controls

As mine workings are developed – and to alleviate the need for excessively long blind headings (open at one end only) – connections known as *cross-cuts* are driven between intake and return airways. Cross-cuts allow the main ventilation system to advance. However, they must subsequently be blocked by *stoppings* built of masonry, concrete, or other substantial material in order to prevent excessive leakage from the intake to the return airways.[4] Access between intakes and returns must still be provided at strategic points, where the stoppings are equipped with *ventilation doors*. Ventilation doors vary in size from about 0.6 m^2 for personnel access to doors large enough to allow vehicles to pass through. For vehicles, Nova Scotia law requires that two doors be used so that at least one remains closed while vehicles are passing through.[5] The two-

[4] *Coal Mines Regulation Act*, s. 71(11). See figures 7.2 and 7.3 for illustrations of stoppings.

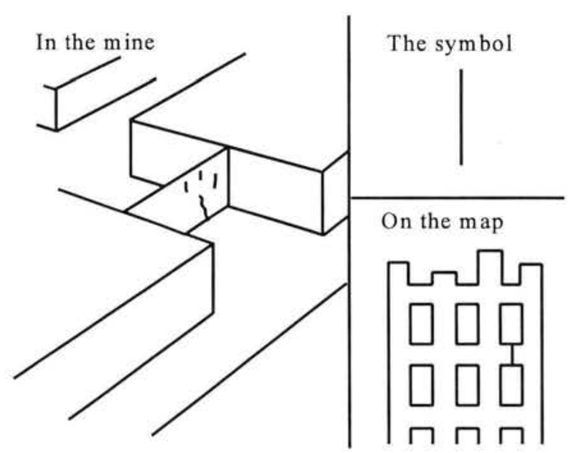
[5] *Coal Mines Regulation Act*, s. 71(12).

Figure 7.2 Permanent Stopping



Source: United States, Department of Labor, Mine Safety and Health Administration, *Mine Ventilation*, Safety Manual No. 20 (Washington, DC: MSHA, 1991).

Figure 7.3 Temporary Stopping

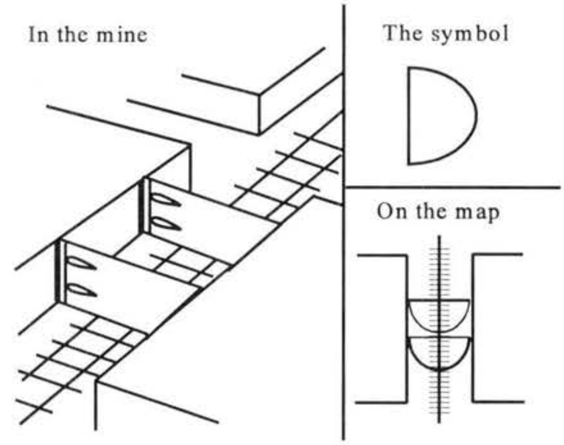


Source: United States, Department of Labor, Mine Safety and Health Administration, *Mine Ventilation*, Safety Manual No. 20 (Washington, DC: MSHA, 1991).

door arrangement is known as an *airlock* (see figure 7.4). All doors should remain closed except when they are used for access, because short-circuiting of the air can result in insufficient airflows to the areas of active mining. It is crucial that this requirement be observed at access points between a main intake and an adjoining main return.

If the air were allowed to flow freely between the sections of a mine, some sections would receive excessive airflows while other sections, more distant from the surface, could suffer from insufficient ventilation. It is therefore necessary to balance the airflows to their required values by

Figure 7.4 Mine Doors Forming an Airlock



Source: United States Department of Labor, Mine Safety and Health Administration, *Mine Ventilation*, Safety Manual No. 20 (Washington, DC: MSHA, 1991).

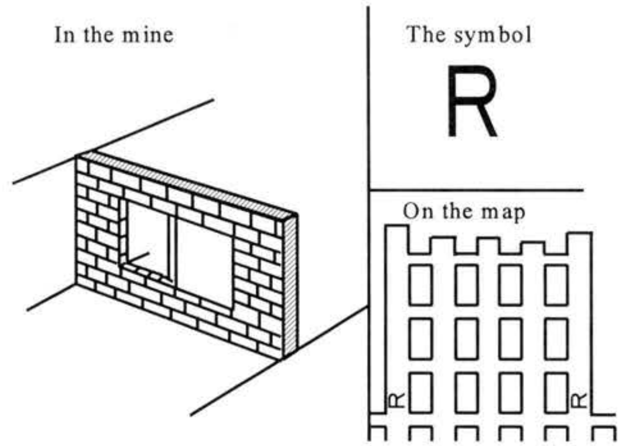
deliberately placing an obstruction within the air courses (intakes or returns) of the otherwise overventilated sections. Such a deliberate obstruction normally takes the form of a ventilation door with an adjustable rectangular opening cut into it. Sliding panels partially cover the opening and are used to regulate the airflow down to the required value. These devices are known as *regulators* (see figure 7.5).

Where the layout of the mine requires intake and return airways to cross each other, steps must be taken to prevent direct short-circuiting of the ventilation at the intersection. The most common way to construct an *air crossing*, or overcast, is to excavate additional material from the roof or floor of the entries and to build a horizontal platform across the intersection to separate the two airways. That platform may take the form of girders with concrete slabs cemented into place. Additional sealant material may be added to the intake side to minimize leakage. For less substantial types of air crossings, air ducts or metal sheeting may be used to separate the two airstreams (see figure 7.6).

Methods of Auxiliary Ventilation

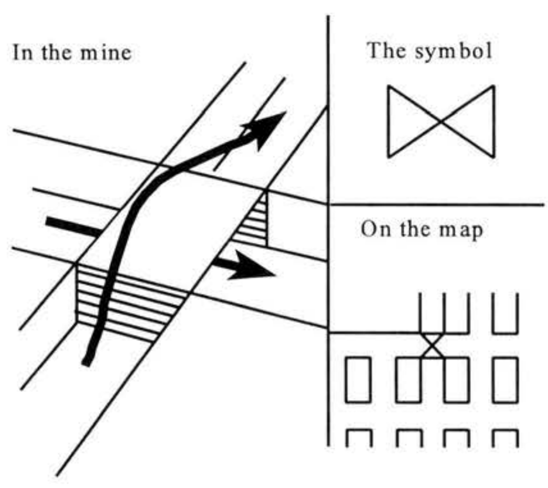
In addition to the mine ventilation system through which airflow is induced by the main fan(s), there may be headings or rooms open at one end only. These cannot be ventilated as part of the main throughflow system, so ventilation can be accomplished only through a local, or *auxiliary*, method. In particular, the room-and-pillar method of mining requires many such headings. Because the major emissions of gas and dust occur at the advancing ends, or faces, of those headings, it is particularly important that the airflows available at the faces are sufficient to remove pollutants safely and efficiently, ensuring that legal limits of their concentrations are not exceeded.

Figure 7.5 Regulator



Source: United States Department of Labor, Mine Safety and Health Administration, *Mine Ventilation*, Safety Manual No. 20 (Washington, DC: MSHA, 1991).

Figure 7.6 Air Crossing (Overcast)

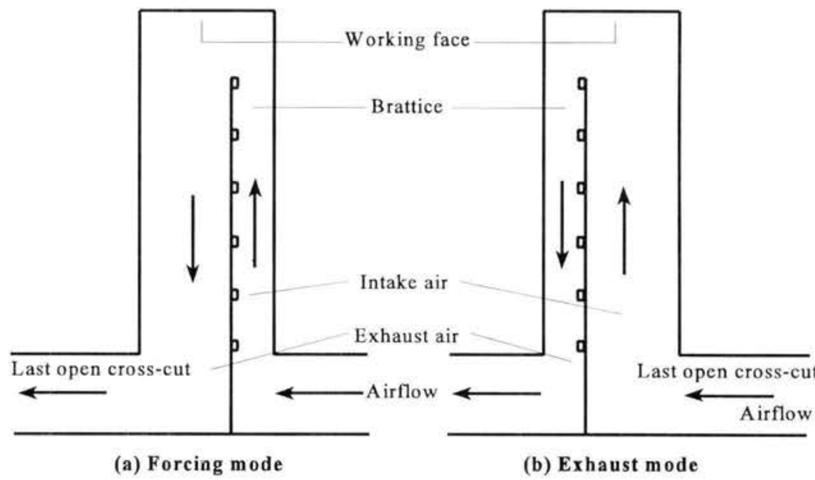


Source: United States, Department of Labor, Mine Safety and Health Administration, *Mine Ventilation*, Safety Manual No. 20 (Washington, DC: MSHA, 1991).

Two methods of auxiliary ventilation are widely practised: the line brattice method, and the auxiliary fan-and-duct method.

The *line brattice* method is favoured in the United States. The principle is illustrated in figure 7.7. In this method, brattice cloth, a heavy woven fabric coated in flame-resistant plastic, is used as a local and temporary means of controlling airflows in underground mines. A continuous line, or curtain, of brattice cloth, reaching from floor to roof along the heading, extends across the last open cross-cut, allowing the airflow to be diverted towards the face of the heading. In the forcing mode,

Figure: 7.7 The Line Brattice Method of Auxiliary Ventilation



Source: Malcolm J. McPherson, *Subsurface Ventilation and Environmental Engineering* (London: Chapman & Hall, 1993), 114.

illustrated in figure 7.7, the line brattice is located so that the corridor carrying air towards the face is narrower than the return path. (Typically, the brattice is placed about one-quarter of the width of the heading from the rib.) The air thus passes relatively quickly to the face. The return path is used for access of personnel and equipment, while the narrower intake passage is only for ventilation. The opposite effect occurs in the *exhaust* mode (figure 7.7). Line brattice can be used with the kind of continuous miner used at the Westray Mine. With an extendable curtain, the airflow can be concentrated to within 2 m of the working face.[6]

The advantages of the line brattice method are:

- It uses the main ventilation system of the mine and, hence, the main fans. It follows that auxiliary ventilation will be maintained as long as the main ventilation structure is operational and the line brattice remains in place.
- It does not require local fans.
- It does not impede access for moving equipment, allowing adequate headroom.
- Vehicular access along the last open cross-cut is facilitated by overlapping sheets of brattice cloth. A vehicle will push aside the cloth, which falls back into place once the vehicle has passed through.
- Capital costs are low in the short term.
- Line brattice requires no power and emits no noise.

---

6 This type of installation is used by Jim Walter Resources, Inc. in Brookwood, Alabama, to ventilate the mine face while driving entries on either side of the longwall panel. It is also used in the exhaust mode.

The disadvantages of the line brattice method are:

- One of the ribs in the heading is hidden from view. The narrower passage can become obstructed by debris that has sloughed from the rib, or even by materials stacked in that inappropriate location.
- Vision is restricted in the last open cross-cut, producing a potential hazard for moving vehicles.
- The ventilating efficiency of the system is low. A significant fraction of the air (usually the majority) will leak across the curtain before reaching the face. The section of brattice in the last open cross-cut often fails to fall back into place after it has been disturbed by vehicles or personnel.
- Line brattices in high workings are more difficult to erect and maintain. The larger surface area is subjected to greater force arising from the air pressure differential across it.
- The additional resistance offered by the line brattice increases the pressure differentials across stoppings and other ventilation controls all the way back to the main fan(s). This causes additional leakage at those points and greater power costs for operating the main fans.

In the *fan-and-duct* system of auxiliary ventilation, the method used at Westray, a fan is located in-line with lengths of ducting. The principle is shown in figure 7.8. In the design of such a system, it is important that the fan-duct combination is able to provide the required airflow at the face of the heading. The resistance offered by the ducting depends upon its length, size, type, and restrictions or shock losses caused by bends, fittings, and configuration of entry and exit. Here again, the technique can be used in either forcing or exhaust mode. In the former, the fan and entrance to the duct are located in the last open cross-cut, upstream from the heading. The relatively fresh air passes through the duct to emerge within the heading with sufficient momentum to project it as a jet further towards the face. The forcing mode ensures the duct is under positive gauge pressure; thus, an unreinforced type of flexible ducting may be used. This type of ducting is less expensive and easier to transport than other varieties.

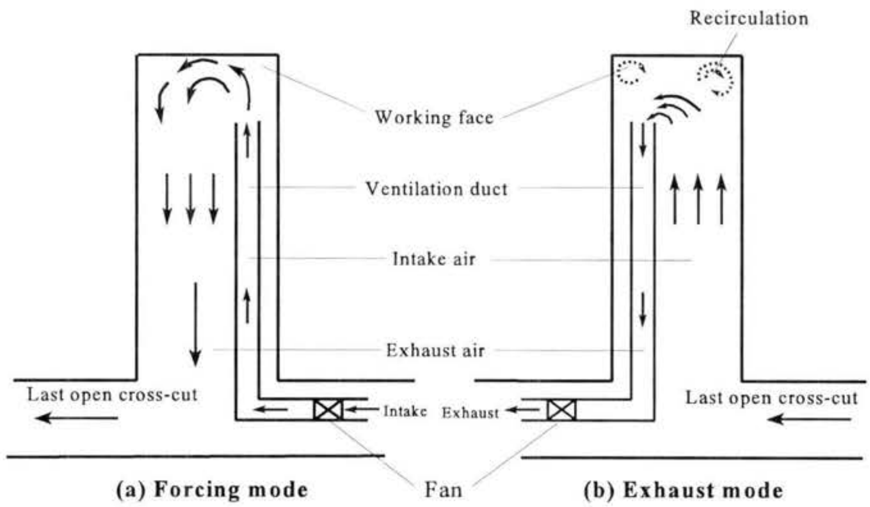
In the exhaust mode, the fan is located in the last open cross-cut, downstream from the heading.[7] The fresh air is drawn up the main body of the heading and returns through the duct. In this case, the air in the duct is at negative gauge pressure, causing suction. Therefore, the ducting must either be constructed from a rigid material (fibreglass or steel) or, if it is flexible, be reinforced against inward collapse. Internal steel spirals normally provide the reinforcement.

In both forcing and exhaust modes, the ducting should be hung close to the roof and, if the roof is laterally inclined, at the higher side. This placement reduces obstruction to equipment and helps prevent methane layering at the face end of gassy headings.

---

[7] This is the system principally used at Westray.

Figure 7.8 The Fan-and-Duct Method of Auxiliary Ventilation



Source: Malcolm J. McPherson, *Subsurface Ventilation and Environmental Engineering* (London: Chapman & Hall, 1993), 115.

The advantages of the fan-and-duct method are:

- It provides a positive and more controllable method of providing airflow to a heading than does the line brattice method. (Line brattice is passive in that it deflects the existing airflow.)
- The velocity of air emerging from the duct in the forcing mode is considerably higher than in the corresponding line brattice application. The jet's longer reach provides improved airflow at the working face.
- Ducting is less liable to leakage than is line brattice.
- Visibility is improved, both within the heading and in the last open cross-cut.
- It causes no additional resistance to the main ventilation system of the mine, thus reducing pressure differentials and leakage across outbye stoppings or doors.
- In the exhaust mode, the momentum of the air emerging from the fan into the last open cross-cut produces a small ventilating pressure, which helps to promote airflow through the main ventilation structure.[8]
- Filters can be located within the ducting to remove dust from the air. Cooling units can similarly be employed in hot mines.
- For headings longer than about 30 m, the fan-and-duct system is the only technique that will provide acceptable airflows to the face area.

The disadvantages of the fan-and-duct method are:

- Fans are noisy.

[8] Note that, in poorly designed layouts, particularly where entries are large, the resulting turbulence can cause undesirable recirculation. This phenomenon occurred at the Westray mine, as will become clear later in this chapter.

- Electrical power is required at the fans.
- Capital costs for auxiliary fans will be incurred as well as the operational expenses of ducting and maintenance.
- Headroom for the passage of vehicles is reduced even with the flatter elliptical ducting. This disadvantage may preclude the use of any ducting when thinner seams are mined.
- Ventilation in the heading is lost when the fan is switched off or loses power.
- Both the electric motors and the high-speed impellers of auxiliary fans may produce sparking in a potentially gassy atmosphere.

If partial recirculation of air through the ducting is to be avoided, the auxiliary fan must pass a volume of air less than that available in the last open cross-cut. Section 71(9c) of the *Coal Mines Regulation Act* requires that the auxiliary fan take no more than 40 per cent of the air passing the fan. A properly designed and monitored system of controlled partial recirculation can improve the mixing and dilution of gases and reduce airborne dust concentrations, but the regulations of most coal mining jurisdictions do not allow the technique because of the fear of its being misapplied. Some jurisdictions, however, grant special permission case-by-case.

Modes of Auxiliary Ventilation

As illustrated in figures 7.7 and 7.8, both the line brattice and the fan-and-duct methods can be used in either forcing or exhaust modes, but there is an essential difference. In the forcing mode, the fresher air is transported and delivered relatively rapidly to the face area, while the return air progresses relatively slowly back along the main body of the heading where the equipment and personnel are located. Conversely, in the exhaust mode, fresher air passes through the main body of the heading, while the polluted air is drawn into the exhaust duct or behind the return brattice.

In specific cases, arguments can be made for either system. The preferred mode depends largely on the pollutant of greatest concern: gas, dust, or heat. In gassy headings, the scouring effect of an air jet issuing from a forcing duct can assist greatly in mixing and diluting gas emitted at the face (figure 7.8a). The same effect is observed, although to a lesser extent, in a forcing line-brattice system. Current Nova Scotia regulations mandate that when an auxiliary fan is used, it must be in the forcing mode.[9] In an exhaust system, no such air jet is available to flush the face. The air is drawn directly into the duct or into the narrower passage behind the brattice, leaving local and sluggish pockets of uncontrolled recirculation near the face of the heading (figure 7.8b).

Where little gas is being produced in the heading, leaving dust as the primary concern, an exhaust system may be preferred. The airborne dust is drawn behind the return brattice or into the exhaust duct rather than

---

9 *Coal Mines Regulation Act*, s. 71(9d).

progressing through the main body of the heading. Whether the forcing or exhaust mode of auxiliary ventilation is used, neither will be effective if the inbye end of the line brattice or the duct is not maintained close to the face. This positioning is particularly important in the exhaust system because of the absence of a jet effect.

Abandoned Areas

All but the newest mines have abandoned areas from which the coal has been extracted. Although nobody enters them, these zones are a potential source of danger.

Methane emissions do not cease when coal mining stops. The gas continues to issue from any remaining coal.[10] If the stress on pillars and on ribs remains constant within the abandoned area, the gas emission will decay with time. The rate of that decay depends on a number of factors, including the extent of the old workings, the initial gas content of the emitting strata, and the permeability of those strata. If strata movement continues within the abandoned section because of crushing of pillars or ribs, or because of roof collapses and the subsequent subsiding of overlying strata, the gas emissions will continue at a higher rate than would be the case if the ground had reached equilibrium.

If the abandoned area is not ventilated adequately, the composition of the atmosphere will change. In addition to an increase in methane, a reduction of the oxygen content and an increase in carbon dioxide will result from oxidation. Accumulations of gases in old workings present several hazards. Unless the mine layout has been properly designed, the accumulated gases may emerge at high concentrations into active sections of the mine. The amount of gas that emerges from an abandoned area over a two-week period (for example) will be approximately the same as the amount of gas emitted from the strata into that area over the same period of time. What goes in must come out. However, variations in the surface barometric pressure cause short-term variations underground. Falling barometric pressure will cause gases held within all worked-out areas to expand, increasing their rate of emission into the mine's ventilation system. For this reason, a barometer should be kept at the surface of the mine and read at the start of each shift.[11]

A sudden large collapse of roof within the old workings can cause a more violent emission of gases from an abandoned area. If the collapse is over a sufficiently large area, a windblast capable of destroying strong stoppings may then occur. Explosions of methane and coal dust have also

---

10 **Comment** The mined-out areas of the Jim Walter Resources mines at Brookwood, Alabama, are a source of methane for Black Warrior Methane Corp. During a tour of methane extraction facilities, Black Warrior's president, Gerry Sanders, told me that the company can profitably drain the gob (the mined-out area) for as long as two years after active mining has ceased. Presumably, some of the hazards associated with the mined-out areas of the mine will be alleviated by this process.

11 *Coal Mines Regulation Act*, ss. 36(2), 38(4), 92(1). The importance of the barometer in underground coal mining is discussed in detail in the section on the barometer in Chapter 6, The Explosion.

been initiated within abandoned areas, these resulting from at least three possible sources of ignition within old workings: friction between blocks of quartzitic rock (sandstones) or between rock and steel during falls of roof; spontaneous combustion of fragmented coal; and, more rarely, a phenomenon know as adiabatic compression, in which the pressure of an explosive atmosphere is so great that it explodes without an outside source of ignition or increase in temperature.

Spontaneous heating of broken coal occurs because of complex physical and chemical reactions taking place on the surface of the material when exposed to air. If insufficient air leaks through the fragmented coal to maintain oxidation, the temperature of the coal will stabilize at a safe level. If the flow of air is sufficiently great to remove heat as quickly as it is being produced, then the temperature will stabilize. However, a dangerous situation arises when there is sufficient air to encourage the oxidation, but not enough to carry away all the heat produced. In such circumstances, the temperature will escalate, further encouraging the rate of oxidation, into a runaway condition. The coal will become incandescent. Spontaneous combustion of this type will produce the highly toxic gas carbon monoxide, as well as large quantities of carbon dioxide.

Two approaches can be taken to minimize the dangers associated with abandoned parts of mines: the old workings must be either ventilated or sealed off, with provisions made in either case to direct emerging gases into return airways. The former method, favoured in the United States, must be used with caution if the coal is susceptible to spontaneous combustion. Figure 7.9 illustrates the back-bleeder system, which may be used both during and after mining has taken place within the section. A regulated flow of air is allowed to move continuously through the section and into the back-bleeder returns, which connect into a main return.

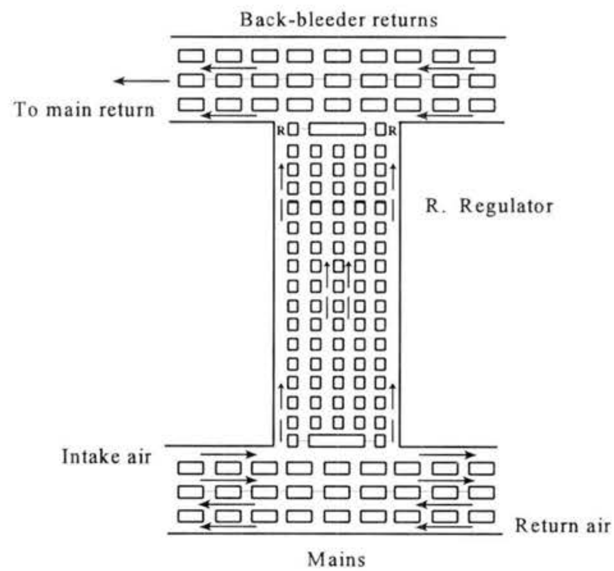
Several precautions must be taken if the abandoned section is to be sealed. First, the seals should be capable of maintaining their integrity in the event of an explosion within the sealed section. This implies double stoppings, keyed into roof, sides, and floor, with the intervening space completely filled with an inert material. Second, the ventilated entry adjacent to the seals should not be an intake airway.[12] In some cases, one or more air crossings may have to be built to meet this requirement.

The difference in air pressure across a sealed area should be as low as possible to minimize leakage through or around the seals. A skilled mine ventilation engineer will use the data gathered through well-managed pressure-volume surveys to plan for the control of pressure differentials across worked-out areas. This planning is particularly important when spontaneous combustion is a possibility.

---

[12] *Coal Mines Regulation Act*, s. 71(6).

Figure 7.9 The Back-bleeder System of Ventilating a Room-and-Pillar Section



Source: Prepared by Malcolm J. McPherson for the Westray Mine Public Inquiry.

The Main Ventilation System at Westray

The ventilation system in place at the time of the explosion at the Westray Mine is shown in maps 5, 6, and 7 in Reference.[13] The intake airways were generally used to transport personnel and materials on diesel vehicles, while the belt conveyors were located in the return airways. This arrangement is known as *homotropal ventilation,* since both the airflow and the transported coal travel in the same direction. It would tend to minimize freezing problems on the main conveyor during the winter months.

This section of the chapter, which describes the development of the throughflow ventilation system at Westray, is divided into three parts. The first part deals with the main access roadways, the main fan, and the surface recirculation duct. The second part describes, in some detail, the condition of the ventilation system in the North and Southeast sections of the mine and reviews how it changed during the three months leading up to the explosion. The third part repeats the process for the Southwest sections of the mine. The auxiliary ventilation of headings is dealt with separately.

Access Mains

As shown on map 5 in Reference, the intake slope known as No. 1 Main served for access of personnel and materials as well as for fresh-air entry

[13] These three maps reconstruct the ventilation picture at the time of a survey taken 8 May 1992 (Exhibit 45.01.15).

into the mine. No. 2 Main, the parallel exhaust slope, carried the main belt conveyor and return air back to the surface. Despite the locations of the conveyors, the mine operated on a main exhaust system of ventilation. This was contrary to recommendations given in feasibility studies carried out by Norwest Resource Consultants Ltd in 1986, by Placer Devėlopment Limited in 1987, and by Kilborn Limited in 1989. Although, as indicated earlier in this chapter, there are good reasons for preferring the main exhaust system in gassy mines, it would seem that Westray chose a main exhaust and homotropal system primarily so that it could use non-permissible diesel equipment for transporting personnel and materials in the intake airways.

Two main fans, one operating and one as a standby, were on the surface above, and connected to, No. 2 Main (the return slope). The fans indicated in the drawings used by the installer, Alphair, were Joy Axivane units, Model M72-43-1200. The model numbers indicate that the fan-casing diameter was 72 inches,[14] the fan hub diameter was 43 inches, and the nominal speed was 1200 rpm. The same model number is indicated on the ventilation maps produced by Westray. Although this model seems to have been installed, the application for permission to use the main fans gives the model number as 72-50-1180.[15]

An airlock, through which the belt conveyor passed, was located near the portal of No. 2 Main. Current regulations require that a pressure differential gauge, commonly known as a water gauge, be connected to the casing of the main fan to indicate that a suitable ventilating pressure is applied to the mine.[16] In addition to providing a continuous record of the pressure developed by the fan, a recording pressure differential gauge also yields invaluable information about explosions, fires, or any other emergency situation that affects the ventilation system of a mine. As discussed in Chapter 6, The Explosion, no such instrument was available at Westray. Neither was a barometer – also a statutory requirement – provided.

An additional duct, fitted with a butterfly valve, was connected to the outlet of the main fan.[17] This duct, which ran across the mine surface to No. 1 Main to recirculate a fraction of the return air back into the main intake, was used in the winter months of 1991–92 in an attempt to increase the temperature of the intake air and to alleviate problems such as icing of the roadway caused by cold temperatures in the main intake. Although the regulations specifically disallow recirculation of auxiliary ventilation, they

---

14 All major nations have converted, or are in the process of converting, to Système Internationale (metric) units. In the United States, federal agencies are required to use SI units, but the vast majority of industry and commerce still retains the old British imperial (foot-pound-second) units. Canada is further advanced in its adoption of SI, but its speed of conversion is inhibited by its proximity to the large U.S. market. The old units are still used widely in the Canadian mining industry. In this Report, both systems of units are used, with conversions given where appropriate.

15 Exhibit 69b.153.

16 *Coal Mines Regulation Act,* ss. 36(2), 38(4). The importance of the water gauge to underground ventilation control is discussed in detail in Chapter 6, The Explosion.

17 Exhibit 73.2, photo 17.

make no mention of recirculation in the primary air circuits of a mine.[18] Controlled partial recirculation of air in mines remains a matter of some controversy. It is practised at some mines in Canada and other countries for various reasons, including the alleviation of low temperatures and the reduction of air-heating costs in cold climates. However, where it is so used, continuous monitoring of the quantity and quality of the air – particularly for carbon monoxide as an indication of fire – should be carried out. No continuous monitoring of either the quantity or the quality of the recirculated air appears to have been carried out at Westray.

Permanent stoppings were constructed between the main slopes in Nos. 1, 2, 4, 6–8, and 10 Cross-cuts. No. 5 Cross-cut had a single stopping with a large steel door for vehicle access. The lack of an airlock (at least two sets of doors) was in contravention of section 71(12) of the act. On at least one occasion, this door was held open, short-circuiting the main ventilation system for about half an hour while mining continued.[19] Double sets of doors large enough to permit vehicle access were installed on No. 3 and No. 11 Cross-cuts. At No. 10 Cross-cut, the intake air split into roughly equal proportions between the North mains and the Southwest. The air returning from the Southwest passed over an air crossing to connect, via No. 9 Cross-cut, into No. 2 Main.

The splitting of the airflows between the North mains and the Southwest was controlled by a regulator in No. 2 Main between No. 9 and No. 10 Cross-cuts (see map 6). This regulator, through which the main conveyor passed, consisted of vertical timber posts with plywood sheets nailed to them. Contrary to the construction of a properly built regulator, there was no way of adjusting it other than by physically removing or adding plywood sheeting.[20]

Finding

Generally, the regulating, control, and monitoring of the main airflow was inadequate and poorly planned. In some cases, the regulating devices contravened the requirements of the Coal Mines Regulation Act. In other cases, these devices were simply improperly constructed, as in the regulator in No. 2 Main between No. 9 and No. 10 Cross-cuts.

Throughflow Ventilation

The changes that took place in the throughflow ventilation arrangements within the sections of the mine in 1992 can be traced from the airflow measurements taken and recorded in ventilation "survey" reports,

18 *Coal Mines Regulation Act*, s. 71(9b).

19 Clive Bardauskas (Hearing transcript, vol. 23, p. 4632).

20 Trevor Eagles (Hearing transcript, vol. 76, pp. 16429–30).

summarized below.[21] The airflows measured between 12 February and 8 May 1992 are given in table 7.1.

At Westray, the measurements of airflow were made at weekly intervals, from February onward, by Trevor Eagles, an engineer-in-training. Routine check measurements and ventilation surveys will be discussed later in this chapter, in the section on ventilation planning.

North and Southeast Sections

12 February 1992 The first of the 1992 reports, dated 12 February 1992, indicates that, at that time, mining in the North workings had ceased because of a major fall of ground in 3 North Main on 9 February. All auxiliary fans in the section were switched off. However, the primary ventilation system maintained throughflow around North 4 Cross-cut. Temporary plastic stoppings in North 1 and 2 Cross-cuts were in need of repair or replacement. More seriously, the stopping in No. 11 Cross-cut between No. 1 Main and No. 2 Main was a temporary arrangement of plastic that was leaking 28.4 kcfm (thousand cubic feet per minute). The report indicates that a permanent bulkhead was required in this location.

19 February 1992 The following week, the stopping in No. 11 Cross-cut was still constructed from plastic. However, steel doors in the stopping were recorded. Five headings – 1 North Main, 2 North Main, North 5 Cross-cut, 3 North Main, and what was to become 1 East – were ventilated by three auxiliary fans operating in a series ventilation arrangement.[22] Tee-jointed ducts were used in the 2 North Main and North 5 Cross-cut headings and in the 3 North Main and 1 East headings.[23] The stoppings in orth 1 and North 2 Cross-cuts were still the temporary plastic ones, with an air recirculation from 2 North Main (main return) into 1 North Main (main intake) of between 10 and 15 kcfm.[24]

26 February 1992 On 21 February, work in the North was again suspended by large falls of ground, this time in North 4 and North 3 Cross-cuts. The blade angle of the main fan was adjusted from 12 to 17 degrees, resulting in a significant increase in the airflow in the main slopes (see table 7.1). Some 36 kcfm passed over the fall in North 4 Cross-cut. The

---

21 Exhibit 37a.044–96. This exhibit includes Eagles's reports to management on his weekly ventilation surveys. Much of the following narrative (North and Southeast Sections; Southwest Sections) is based on these reports and the accompanying maps in Exhibit 45.01.07–15.

22 Series ventilation occurs, in this context, when air issuing from a heading is returned into a throughflow airstream, which is then used, further downstream, to provide air for the auxiliary ventilation arrangements of one or more further headings. Since each heading adds airborne pollutants, it follows that the air will progressively suffer a loss of quality. For this reason, series ventilation should be avoided wherever possible. (See also *Coal Mines Regulation Act*, s. 71(5).)

23 For an explanation of tee-jointed ducts, see the section on use and maintenance of ducting later in this chapter.

24 Recirculation in the North and Southeast sections was a recurring theme throughout the records in 1992. It was caused by a combination of the air jets issuing from exhausting auxiliary fans into the throughflow ventilation system, and natural ventilating effects resulting from warmer air in a rising return airway.

Table 7.1 1992 Measured Airflows (kcfm)

Location	12 Feb 92	19 Feb 92	26 Feb 92	4 Mar 92	11 Mar 92	18 Mar[b]	2 Apr 92[c]	8 Apr 92	15 Apr 92	23 Apr 92	29 Apr 92	8 May 92
No. 1 Main, outbye 1 Cross-cut	166.3	163.5	182.7[a]			181.4	223.6	218.4				
No. 1 Main, inbye 8 Cross-cut	155.5	144.0	176.8	160.2	172.0	176.0	207.7	204.1	201.6	197.3	203.4	190.8
No. 1 Main, inbye 10 Cross-cut	88.8				76.1		103.9	99.9	102.4	103.3	101.1	100.8
SW1-C1 Road	68.6	65.6	71.2	80.5	87.3	89.3	97.6	99.0		94.2	96.8	93.5
No. 1 Main, inbye 11 Cross-cut	60.4	73.8	68.4	60.9	61.4		78.8	74.1	76.1	95.9	91.5	89.5
1 North Main, outbye North 4 Cross-	51.8	81.2	36.4		27.7		74.5		75.8	79.5		
SW1-6 Cross-cut	30.6	29.3	41.8	48.6								
No. 2 Main, at 9 Cross-cut		132.0	157.0	157.2	155.7	158.8				(d)		
No. 1 Main, outbye 11 Cross-cut		76.7	83.7	73.7			98.8	98.2	100.2		98.8	
1 North Main, outbye North 2 Cross-		70.8	64.0	62.3	65.8				75.9			
SW1-B Road, outbye SW1-1 Cross-cut				86.4	86.4	84.5	100.4	103.6	98.3	95.0	93.7	88.4
SW1-3 Cross-cut				11.2								
SW1-B Road, inbye SW1-6 Cross-cut					48.1							
2 North Main, inbye North 2 Cross-cut					35.1							
SW1-B Road, outbye SW1-8 Cross-cut						49.1						
SW1-C1 Road, inbye SW1-3 Cross-cut							20.9					
1 North Main, outbye North 4 Cross-								76.1				
2 North Main, outbye North 4 Cross-										77.2	83.2	77.5
1 East								31.9	32.4	62.1		75.5
1 North Main, inbye North 4 Cross-cut								87.5				63.8
SW2- B Road, outbye SW2-1 Cross-									78.4	81.5		
2 East											66.8	
SW1-C1, outbye SW1-3 Cross-cut											65.8	
SW2-A Road, inbye SW2-1 Cross-cut											38.2	59.8
SW1-A Road (face)			11.7	7.0	5.2							
SW1-A1 Road (face)			13.9	10.4	7.2							
SW1-A2 Road (face)			10.4	17.0	13.0							
SW1-A3 Road (face)				13.6	16.8							
SW2-B Road (face)									7.8			
SW2-A Road (face)									6.4			
SW2-1 (fan)												5.2
SW2 "C" (fan)												5.3
Southeast (fan)												7.0
Northwest 1 Cross-cut (fan)										4.7		
2 North Main (fan)										5.2		
SW2-B Road (fan)										4.7		

Source: Exhibit 37a 044–96.

a Blade angle changed from 12° to 17°
b Anemometer damaged, could not complete survey
c Southwest 1 production stopped 25 March, abandoned by 28 March
d Regulator in No. 2 Main opened

plastic stopping in North 2 Cross-cut was opened to short-circuit the obstructed areas and to further assist in maintaining acceptable ventilation as far as that cross-cut in the North Mains.

4 March 1992 A limited rate of production in the North did not resume until 4 March. The airflow measurements taken on that date indicate that the auxiliary fans remained switched off. The plastic stopping with steel doors in No. 11 Cross-cut, still not replaced by a bulkhead, was leaking some 12.7 kcfm.

11 March 1992 By 11 March, limited production was taking place in 2 North Main and 2 East, with those two headings ventilated in series. The North 2 Cross-cut remained open, short-circuiting 33.7 kcfm and leaving only 27.7 kcfm to find its way over the fall in North 4 Cross-cut to provide air for the 2 North Main active heading. It is unclear why this direct short-circuit was allowed to continue. The plastic stopping in No. 11 Cross-cut continued to leak at a rate of 12.2 kcfm.

18 March 1992 The airflow measurements taken on 18 March were cut short because of a damaged anemometer. No airflow measurements were made in the North or Southeast sections during this week.

Mining continued in the 2 North Main, North A Road, and 2 East headings, with series ventilation. The fan providing air to 2 North Main and the North A heading was located in North 5 Cross-cut, requiring an excessive length of ducting. The section of ducting adjacent to the fan serving 2 East needed replacing. The stopping in No. 11 Cross-cut remained as a temporary construction, with an air leakage of 12 kcfm. Although the North A heading had now joined up with 2 North Main, advancing the main throughflow system, North 2 Cross-cut remained open as a direct short-circuit between the main intake and return.

2 April 1992 There are no records of airflow measurements having been taken between 18 March and 2 April. The temporary plastic stopping in North 2 Cross-cut had been removed on 21 February because of the fall in North 4 Cross-cut. A properly constructed permanent stopping should have been built in North 2 Cross-cut as soon as the fall was bypassed by the 2 North Main to North A Road interconnection. That connection occurred during the week prior to 18 March. However, the stopping was not reported as being replaced until 2 April. During that period, there had been a large loss of air across the short-circuit. When the stopping in North 2 Cross-cut was eventually replaced, it was by yet another temporary plastic one instead of a permanent structure and was leaking 7.5 kcfm. Air pressure increased across the plastic and steel door stopping in No. 11 Cross-cut, where the air loss had risen to 20 kcfm. On 2 April, auxiliary fans were ventilating the North A and North B headings in the North and 1 East and 2 East in the Southeast, again effectively in series. A section of the ducting in the North A heading had collapsed. A recirculation of approximately 15 kcfm occurred over the fall in North 4 Cross-cut. Concern over the growing rate of recirculation may have been the reason for an additional adjustment of the main surface fan, as reflected by another increase in the main slopes' airflow measured during this week (see table 7.1).

8 April 1992 Once again, ground control problems were occurring in the Southeast area, and the auxiliary fans in 1 East and 2 East were switched off. Ducts remained operating in the North A, North B, and 2 North Main headings. Despite the adjustment to the main fan in the previous week, the recirculation situation was growing worse. Although an airflow of 87.5 kcfm was measured approaching the last open cross-cut (North A Road), only 74.1 kcfm progressed inbye along 1 North Main (measured outbye North 4 Cross-cut). There was, therefore, a recirculation of at least 13.4 kcfm in the North section. (The measurements indicate 11.4 kcfm recirculating over the fall in North 4 Cross-cut, and a further 2 kcfm recirculating across North 2 Cross-cut). The leakage across the plastic and steel door stopping in No. 11 Cross-cut had increased further, to 24 kcfm.

15 April 1992 The damaged anemometer had been repaired, recalibrated, and returned to the mine by this date. However, the only work in progress in the North and Southeast sections was the setting of steel arches. No mining was going on; the only auxiliary fan operating was that serving the North A heading. Although reported two weeks earlier, a section of the ducting in that heading still needed to be replaced. Construction of a permanent concrete-block stopping had, at last, been started in No. 11 Cross-cut and was now half-complete – nine weeks after the engineer responsible for the airflow measurements had first requested that a proper bulkhead be built in this location. Recirculation estimated at 10 kcfm continued over the fall in North 4 Cross-cut. An airflow of 5 kcfm occurred along 3 North Main between 1 East and 2 East. The 15 April report recommended that a stopping be constructed in this location.

23 April 1992 No ventilation map is available for the 23 April measurements. The permanent stopping in No. 11 Cross-cut had now been completed and, coupled with the widening of the regulator setting in No. 2 Main between No. 9 and No. 10 Cross-cuts, resulted in a significant increase in the airflow supplied to the North sections. The stopping in North 2 Cross-cut was reported to be constructed from strips of conveyor belt and leaking at a rate of approximately 10 kcfm. The 2 North Main and North A headings were being ventilated. Once again, the ducting in the North A heading was reported as needing replacement. An auxiliary fan was located in 2 East to ventilate the 1 Southeast heading. This was causing a recirculation of 13.3 kcfm from 2 East to 1 East, the stopping requested the previous week for 3 North Main not having been constructed.

29 April 1992 No ventilation map is available for the 29 April airflow measurements. However, Eagles reported an auxiliary fan exhausting from the North A heading. The fan located in 2 East continued to exhaust air from the 1 Southeast heading, but a plastic stopping had now been erected in 3 North Main to control the recirculation from 2 East back to 1 East. As had been the case since March, the ventilation of the Southeast section continued downstream from, and in series with, that in the North section. Eagles indicated further dissatisfaction with the belt-strip "stopping" in North 2 Cross-cut.

8 May 1992 This final set of airflow measurements was recorded less than 24 hours before the explosion and may therefore be considered fairly representative of the state of the ventilation system at the time of the explosion. The situation is illustrated on map 7. An auxiliary fan was located in Northwest 1 Cross-cut, exhausting from the North A heading.[25] A second fan in Northwest 1 Cross-cut was attached to ducting in the North B heading. The operational status of this fan on 8 May is unknown. A fan in 2 North Main drew air via a tee-jointed duct from the 2 North Main heading and North 6 Cross-cut. There were, therefore, four headings with series ventilation in the North section. These, in turn, were in series with the auxiliary ventilation of the 1 Southeast heading, giving a total of five headings with series ventilation.

The ventilation arrangements for the 1 Southeast heading can only be described as strange. Mining had been severely inhibited in this area because of methane emissions.[26] The length, type, and size of ducting in 1 Southeast restricted the exhaust ventilation to 7.1 kcfm – insufficient to remove the gas effectively from the heading, even with a 30 kW (40-horsepower) fan. An attempt was made to increase the airflow in the heading by attaching a short length of ducting to a fan located on the downstream side of the heading entrance and using it in the forcing mode. The map for 8 May shows this ducting protruding only a few metres into 1 Southeast. The effect would be to increase the airflow within that short distance, yet it would have minimal influence on the flow drawn from the inbye end of the heading by the primary exhaust duct.

Observations

Throughout the life of the North and Southeast sections of the mine, production was interrupted frequently by falls of roof. These had a severe impact on the ventilation structure and continuity of airflows to the headings. Despite the proximity of the main intake and return slopes and the commensurate need for vigilance against leakage, only one permanent stopping was built inbye No. 10 Cross-cut in the main slopes. This was in No. 11 Cross-cut, and it was not built until 15 April 1992 – nine weeks after it had been requested by Eagles. All other stoppings were constructed from plastic or, in the case of North 2 Cross-cut after 15 April, strips of conveyor belting. Such structures were out of compliance with section 71(11) of the *Coal Mines Regulation Act.*

---

25 This cross-cut is not labelled on the maps. It connects North A, B, and D Roads inbye the North mains.

26 Don Dooley mentioned this in his testimony (Hearing transcript, vol. 35, p. 7837).

Air leakage rates were high, and uncontrolled recirculation of air was prevalent. It would appear that the mine management was ignorant of the reason for the recirculation. The main cause was the induction of additional air motion by a jet of relatively high velocity air from each auxiliary fan exhausting into throughflow airstreams. This would have been avoided by employing forcing auxiliary fans, as required by section 71(9d) of the act.

No airflow measurements were made in the North or Southeast sections during the week of 18 March 1992 because of a damaged anemometer. Spare anemometers should have been available on site because airflow measurements are the primary means for checking the operation of a mine's ventilation system.

Throughout the three-month period described above, headings were ventilated in series, to a total of five headings in series by 8 May 1992. In conjunction with the uncontrolled recirculation and inadequate auxiliary ventilation, this situation led to dangerous mining conditions, particularly in the Southeast section, with respect to methane.

Finding

The ventilation system in the North Mains and Southeast sections of the mine was haphazard, reflecting little or no planning. Plastic stoppings were generally in a state of disrepair – increasing the leakage of air, promoting the recirculation of air, and decreasing the quality and flow of ventilation air. Faulty placement of auxiliary fans further decreased the flow and caused problems such as collapsed ducting, which remained in that state for unduly long periods. The placement of the auxiliary fans in these sections further diminished the airflow – to the extent that it was incapable of flushing liberated methane from the headings. The combined effect of all these deficiencies was to perpetuate poor air quality, the air circulating or recirculating within the sections at velocities too low to remove dangerous contaminants. Significantly, these conditions appear to have been tolerated, or even ignored, by a complacent or careless management.

Especially appalling is the thought that these dangerous conditions were not even recognized by an ill-trained and incompetent management.

Southwest Sections

12 February 1992 In early February, mining was taking place in the Southwest 1 (SW1-B, SW1-A, and SW1-A1) headings. The SW1-A and SW1-A1 headings were ventilated by a common exhaust fan and a tee-jointed duct, in series with the SW1-B heading. Of the 68.6 kcfm that entered the Southwest area, at least 38 kcfm (55 per cent) were lost by leakage through the stoppings in SW1-1 Cross-cut (wooden construction), SW1-2 Cross-cut (wood), and SW1-3 Cross-cut (plastic). There was a slight recirculation through SW1-5 Cross-cut (wood). No measurements of airflow in the auxiliary ducts were recorded. However, the report for this date indicated that the flows were very low in the SW1-A and SW1-A1 headings.

19 February 1992 Little change in the throughflow ventilation had been made from the previous week, the leakage remaining at 55 per cent. The wooden stopping at SW1-1 Cross-cut was reported as leaking 15 kcfm. The plastic stopping at SW1-5 Cross-cut was in poor condition and recirculating approximately 5 kcfm. Three auxiliary fans were now in operation, one exhausting from the SW1-B heading, and one from the SW1-A heading. The duct serving the SW1-B heading was reported as being 75 per cent closed off and, unsurprisingly, yielding poor ventilation. The third fan operated in the forcing mode, providing air through a tee-jointed duct to the SW1-A1 heading and the developing SW1-4 Cross-cut. During the three-month period preceding the explosion, this is the only auxiliary fan-and-duct arrangement that complied with the mandatory requirement for a forcing system.

26 February 1992 An adjustment of the blade angle from 12 to 17 degrees on the impeller of the main fan had resulted in an increase from 65.6 to 71.2 kcfm in SW1-C1 Road (supplying air to the Southwest section). The leakage across the stoppings between SW1-C1 and SW1-B Roads had been reduced to 29.4 kcfm (41 per cent), indicating that those stoppings had received some attention. Exhaust fan-and-duct arrangements continued to serve the SW1-A and SW1-A1 headings, in series with the forcing fan that ventilated the SW1-A2 and SW1-4 Cross-cut headings. For the first time, airflows were measured in the auxiliary ducts (see table 7.1). All ducting was reported to be in good condition.

4 March 1992 The airflow entering the Southwest section had risen to 80.5 kcfm, resulting from a reduction in the open area of the regulator in No. 2 Main following the 26 February measurements. Additional repair work to the Southwest stoppings had been carried out. Nevertheless, the leakage between SW1-C1 and SW1-B Roads remained high at 32 kcfm (40 per cent). A plastic stopping, erected in SW1-A1 Road, was recirculating approximately 7.5 kcfm. The duct exhausting air from the SW1-A heading was reported at 50 per cent closed, resulting in poor ventilation in this heading. The SW1-A2 heading had connected into, and advanced beyond, SW1-8 Cross-cut and was now ventilated by a dedicated exhaust fan and duct. The forcing fan had been moved to the outbye end of SW1-A2 Road and ventilated the new SW1-A3 heading. This gave four headings ventilated in series.

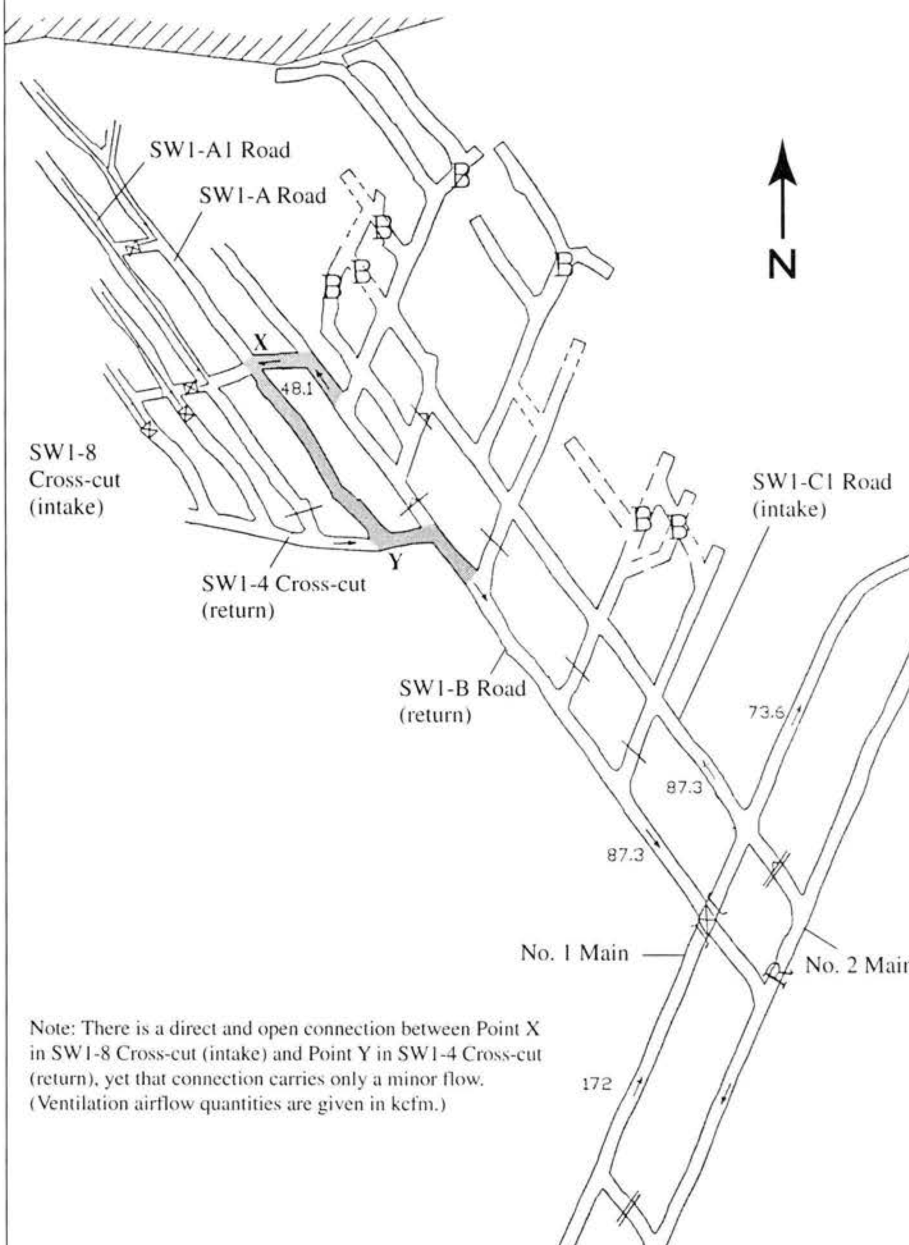
11 March 1992 On 11 March, depillaring (pillar recovery) began in Southwest 1. Leakage across the stoppings separating SW1-C1 and SW1-B Roads remained high at 39.2 kcfm (45 per cent). Exhaust fan-and-duct auxiliary systems ventilated the SW1-A, A1, A2, and A3 headings, all in series. The duct serving the SW1-A heading was tee-jointed, drawing air from both the SW1-A heading and SW1-10 Cross-cut. This duct was in a particularly poor state, with the section of ducting adjacent to the fan 80 per cent closed, resulting in a total airflow of only 5.2 kcfm serving the SW1-A heading and SW1-10 Cross-cut (see table 7.1). The ducting in the SW1-A1 heading was also in poor condition, passing an airflow of 7.2 kcfm. Control of the throughflow ventilation in Southwest 1 was being lost at this stage. Figure 7.10 shows that the active areas were all west of SW1-B Road and were supplied by a total airflow of 48.1 kcfm, via SW1-8 Cross-cut and returning through SW1-4 Cross-cut. The SW1-A Road contained the belt conveyor, but no stopping. Therefore, there was a direct and open short- circuit between the intake (point X on figure 7.10) and the return (point Y). Nevertheless, Eagles commented in his notes that only minor flow was measured in this airway. This observation indicates that the main ventilation system was incapable of providing a positive ventilating pressure differential across the active working area in Southwest 1. The airflow that proceeded through this area was induced primarily by the air jets issuing from the auxiliary fans and resulting in a recirculation of 7.5 kcfm through the plastic stopping in SW1-A1 Road.

18 March 1992 The ventilation system in Southwest 1 had changed little since the previous week. Depillaring continued in the SW1-A heading and the nearby SW1-B extension. Both were supplied via a tee-joint from the same exhaust duct. No airflow measurements in the ducts were recorded, since the high-speed anemometer used for this purpose had been damaged. It is unlikely that satisfactory ventilation could have been provided to depillaring operations by a tee-jointed duct. As reported in the previous week, the ducting drawing air from the SW1-A1 heading was again in a poor condition, with several sections crushed. As well, a section of the ducting in SW1-A2 Road needed to be replaced. The inbye end of this duct was an excessive 30 m from the face, indicating that there was no effective ventilation at the face of this heading. At the time of these observations, the fan serving the SW1-A3 heading was switched off, resulting in an unquantified recirculation in the completed section of SW1-A3 Road. A plastic stopping had now been erected in the SW1-A belt road. The wooden stopping in SW1-3 Cross-cut between SW1-C1 and SW1-B Roads had deteriorated and was passing a leakage of approximately 15 kcfm. Eagles reported that it had holes in it and was in need of repair.

2 April 1992 There are no quantified reports of airflow measurements between 18 March and 2 April. On 25 March, one week after the 18 March observations, production ceased completely in the Southwest 1 section because of crushing of the finger pillars. Equipment was withdrawn, and the area was abandoned by 28 March. Of the numerous hazardous situations that occurred at Westray, the withdrawal from Southwest 1 was the most dangerous operation prior to the actual explosion.[27] The

[27] A number of witnesses attested to the hazards of this operation. Trevor Eagles was familiar with the inadequate ventilation and high levels of methane in Southwest 1 (Hearing transcript, vol. 76, pp. 16472–78, 16505–06, 16517–20). Wyman Gosbee discussed the poor ventilation and high methane (vol. 25, pp. 4983, 4987–89). Lenny Bonner spoke of operating equipment

Figure 7.10 Short-circuiting in Southwest 1, 11 March 1992



Note: There is a direct and open connection between Point X in SW1-8 Cross-cut (intake) and Point Y in SW1-4 Cross-cut (return), yet that connection carries only a minor flow. (Ventilation airflow quantities are given in kcfm.)

Source: Prepared for the Westray Mine Public Inquiry by Dr Malcolm J. McPherson from ventilation map of Southwest 1 on 11 March 1992 (Exhibit 45.01).

depillaring operations of the preceding week and crushing of the remaining pillars would have created emissions of gas probably greater

---

under extremely dangerous conditions (vol. 24, pp. 4753–58). The withdrawal from Southwest 1 is covered in Chapter 10, Ground Control, and Chapter 5, Working Underground.

than those experienced in the headings. Damage to stoppings and further loss of control of the already inadequate ventilation would have resulted in methane remaining inadequately diluted and failing to be removed from the area. The methane would accumulate – initially at roof level, especially in roof cavities created by the falls of roof during the withdrawal – and continue to fill the entries after abandonment of the area.

No ventilation map is available for this date, although the situation is approximated by the map for the following week. An increase in the total airflow supplied to the mine suggests that the main fan had been adjusted during the preceding week (see table 7.1). All work had been abandoned in the Southwest 1 section. A wood-and-plastic stopping had been erected in SW1-B Road inbye SW1-4 Cross-cut. The SW2-A and SW2-B headings had been started and were ventilated by exhaust fan-and-duct arrangements in series. An airflow of 97.6 kcfm entered the Southwest area via SW1-C1 Road. There were wood or wood-and-plastic stoppings in SW1-1 and SW1-2 Cross-cuts. The three entrances to the abandoned Southwest 1 section immediately inbye SW1-3 Cross-cut remained open. These were SW1-C1 and SW1-B Roads, and the old 2 North Main A entry that was obstructed by a roof fall. A belt-strip "stopping" had been erected in SW1-3 Cross-cut in an attempt to induce airflow around Southwest 1. This allowed vehicles to access the developing Southwest 2 section via the intakes – SW1-C1 Road and SW1-3 Cross-cut. The belt-strip stopping was ineffective and allowed the passage of approximately 70 kcfm. A measurement taken in SW1-C1 Road inbye SW1-3 Cross-cut indicated that the airflow progressing into the Southwest 1 section was limited to approximately 21 kcfm.

From this time onward, the new Southwest 2 section was ventilated by intake air that had been routed past the entrances to abandoned workings. This contravened section 71(6) of the *Coal Mines Regulation Act*. To compound the problem, an airflow was being deliberately diverted through those old workings – insufficient to deal with the methane that was being produced, but enough to carry dangerous concentrations of the explosive gas towards the Southwest 2 developments. Methanometer measurements taken in SW1-B Road outbye SW1-4 Cross-cut showed that the air returning from Southwest 1 contained 2.5 per cent methane in the general airstream and 9.0 per cent near the roof.

8 April 1992 SW2-A and SW2-B Roads had advanced to the point where they could connect through SW2-1 Cross-cut. Nevertheless, the auxiliary exhaust fans had not yet been moved forward, remaining in SW1-B Road. The wood-and-plastic stopping in SW1-B Road inbye SW1-4 Cross-cut, and the belt-strip "stopping" in SW1-3 Cross-cut, had both been dismantled. An airflow estimated at 15 kcfm and a methane concentration of 4.0 per cent (near the roof) were reported for the former location.

15 April 1992 Two days before the 15 April measurements were taken, plywood stoppings had been erected in SW1-C1 and SW1-B Roads, both inbye SW1-3 Cross-cut. These stoppings prevented access to the abandoned Southwest 1 section and reduced the airflow supplied to that section to leakage values. Ground control difficulties had been experienced in those same locations and wooden chocks had been built for roof support.[28] One of the miners building the chock in SW1-B Road became dizzy while working near the roof. His dizziness was probably caused by displacement of oxygen by methane.[29] Plywood sheets, ¼-inch thick, were nailed to the chocks to form the stoppings intended to isolate the abandoned workings.[30] The construction of such stoppings did not follow with prudent practice. Old workings should either be ventilated adequately and directly into return airways or, if not so ventilated, be isolated by explosion-proof seals. Unfortunately, the flimsiness of the plywood stoppings was not their only weakness. They were built in disturbed ground; in particular, the stopping in SW1-B Road was incapable of preventing high concentrations of methane from issuing out into the intake air supplying the Southwest 2 section. Leakage air moved inbye across the stopping in SW1-C1 Road and re-emerged, contaminated by methane, back into the intake route through the stopping in SW1-B Road.

The auxiliary fan exhausting from SW2-B Road had been moved up to SW2-1 Cross-cut. The fan serving SW2-A Road was located at the outbye end of the road, requiring an unnecessarily long length of ducting. The two headings were ventilated in series. Airflow measurements indicated low flows of 7.8 kcfm for SW2-B Road and 6.4 kcfm for SW2-A Road. The ventilation map for this date indicates that a stopping had been erected in the belt road (SW1-B Road) between the entrances to SW2-A and SW2-B Roads.

23 April 1992 No ventilation map is available for the 23 April measurements. Upward movement of the underlying strata (floor heave) was causing the plywood stoppings in SW1-C1 and SW1-B Roads to buckle. The ducting exhausting air from SW2-B Road was reported as being in good condition but not hung straight. Nevertheless, the measured airflow for the corresponding fan was only 4.7 kcfm, indicative of a significant obstruction in the duct. No airflow measurement was recorded for the SW2-A heading.

29 April 1992 No ventilation map is available for 29 April. Floor heave and buckling of both Southwest 1 stoppings in SW1-C1 and SW1-B Roads were noted. Apertures had developed in those stoppings, allowing an estimated leakage of 5 kcfm through the Southwest 1 section.[31]

---

28 Don Dooley (Hearing transcript, vol. 36, pp. 7956–57).

29 Harvey Martin (Hearing transcript, vol. 23, pp. 4538–39).

30 Jonathan Knock (Hearing transcript, vol. 26, pp. 5285–86).

31 The ventilation survey report of 29 April refers to the estimated leakage (Exhibit 37a.074). Eagles said in testimony that "[the stoppings] were closed off, but you could, if you really wanted to, probably stick your head through and take a look in some of the buckles" (Hearing transcript, vol. 76, p. 16599). Mick Franks, in his testimony, said, "I don't know if the floor was heaving or if the roof was coming in, but it was all busted. The plywood was all busted" (vol. 21, p. 4150).

The total airflow entering the Southwest section via the outbye end of SW1-C1 Road was 96.8 kcfm. Unfortunately, 58.6 kcfm (61 per cent) of this airflow was lost to leakage. The wood-and-plastic stopping in SW1-2 Cross-cut had now been replaced by conveyor belt strips and was suffering from a large leakage of some 31 kcfm. The background to this change was that the previous access route around SW1-3 Cross-cut had become inaccessible for vehicles because of poor roof conditions at the junction of SW1-C1 Road and SW1-3 Cross-cut. Vehicles were now required to travel inbye from SW1-C1 Road, through the belt-strip stopping in SW1-2 Cross-cut, across SW1-B Road, and up SW2-A Road to the working areas. The initial location of the Southwest 2 conveyor had been in SW2-A Road (return), with the conveyor drive at the junction with SW1-B Road. Because this would have made that junction unsuitable for the passage of vehicles, the conveyor for the Southwest 2 section had been moved to SW2-B Road (intake).[32] A consequence of these changes was that vehicles now travelled in a return entry, which is not suitable for non-permissible vehicles. An alternative plan would have been to construct an air crossing at the SW1-B Road and SW2-A Road intersection and reverse the ventilation around the Southwest 2 section. This would have allowed intake access for vehicles and would also have eliminated the need for the Southwest 2 intake route to pass the stopped entrances of the abandoned Southwest 1 section.

In addition to the large loss of air at SW1-2 Cross-cut, the plastic stopping in SW2-1 Cross-cut was passing an excessive leakage of 22.6 kcfm. A further 5 kcfm was lost through the conveyor stopping in SW1-B Road. Two exhaust auxiliary fans were now located side by side in SW2-2 Cross-cut – one ventilating the SW2-B heading and the other drawing air from SW2-1 Road. In series with those was the fan exhausting air from the SW2-A heading. There are no records of airflows having been measured in the auxiliary systems.

8 May 1992 The Southwest airflows measured the day before the explosion are shown on map 6. Two exhaust fans remain side by side in SW2-2 Cross-cut. The corresponding ducts are both tee-jointed, one drawing air from SW2-3 Road and the advancing SW2-B heading, the other exhausting from SW2-1 Road and the Lefthander. Only two of these airflows were recorded: 5.2 kcfm in SW2-1 Road, and 5.3 kcfm in the SW2-B heading. A third fan exhausted air from the SW2-A heading, giving an effective five headings ventilated in series. The belt-strip stopping in SW1-2 Cross-cut continued to leak approximately 17 kcfm, with an additional 16 kcfm lost across SW2-1 Cross-cut and the conveyor belt stopping in SW1-B Road. These combined leakages represented 36 per cent of the air entering the Southwest section. The conveyor belt stopping was constructed from wood and plastic, and it had a personnel door.

---

32 Bryce Capstick, in his testimony, explained these changes (Hearing transcript, vol. 42, pp. 9364–67). Eagles also discussed the situation in his testimony (vol. 76, pp. 16575–76).

Observations
The weaknesses of the main ventilation system in the North and Southeast sections not only appeared in the Southwest sections, but were also compounded by further defects. Not one permanent stopping was built anywhere in Southwest 1 or Southwest 2. All stoppings were flimsy constructions of wood, plastic, or, even worse, strips of conveyor belting. As in the northern sections, such structures were out of compliance with section 71(11) of the *Coal Mines Regulation Act*. The result was that high leakages occurred. Westray practised series ventilation throughout the lives of both Southwest 1 and Southwest 2 sections, with up to five headings ventilated in series.

Lack of planning or of forethought led to a loss of control of ventilation in Southwest 1 even before mining had to be terminated because of failing pillars. Recirculation was common and out of control. Given the ground conditions and heavy emissions of methane, the withdrawal of equipment from Southwest 1 was an extremely hazardous procedure. For more than two weeks – from 28 March to 13 April 1992 – the abandoned Southwest 1 section, although known to be filling with an explosive gas, was left with neither stoppings nor ventilation adequate to dilute and remove the gas safely. When the stoppings were erected, they were built from ¼-inch plywood in highly disturbed ground that continued to be unstable. Such structures were completely incapable of withstanding either strata pressures or any sudden air movement that might be caused by large roof falls or an ignition of gas within the abandoned area. Neither were they capable of preventing emissions of gas into the main ventilation system. Contrary to both the law and common sense, intake air for the Southwest 2 section was routed past those inadequate stoppings, beyond which the old workings were still actively producing methane.

Deteriorating ground conditions at the junction of SW1-C1 Road and SW1-3 Cross-cut led to a relocation of the Southwest 2 conveyor into an intake airway and necessitated the use of a return airway for access of vehicles, some of which were not designed for use in potentially gassy atmospheres.

Finding

The ventilation system in the Southwest section was consistently defective and inadequate. The ventilation system in the North Mains and the Southeast sections was also defective and inadequate. The litany of defects includes:

- **poorly constructed plastic stoppings, permitting air leakage of up to 55 per cent of the total airflow;**
- **the broken anemometer (with no replacement on site), which prevented the taking of airflow measurements for two weeks;**
- **low ventilation pressures and low airflows, which provided little or no air movement at the working faces where required to clear methane;**
- **intake air directed past the two plastic stoppings inbye the SW1-3 Cross-cut, which were leaking quantities of methane from the abandoned areas**

into the active workings of the Southwest 2 1 section and contributing to the methane-layering problem; and

- **placement of conveyors in an intake airway, necessitating the movement of non-permissible vehicles in the return airways.**

All these factors lead inexorably to the conclusion that Westray's management was either apathetic or, through incompetence, unaware of the implications of its actions and decisions in these crucial matters.

Auxiliary Ventilation at Westray

Section 71(9c) of the *Coal Mines Regulation Act* stipulates that an auxiliary fan must take not more than 40 per cent of the air passing the fan. This regulation is a safeguard against recirculation within the heading itself and was complied with at Westray. However, the intent of the regulation was circumvented by the excessive number of headings ventilated in series and by uncontrolled recirculation within the ventilation structure. Notwithstanding the weaknesses of the ventilation structure, the major cause of the difficulties experienced in the headings at Westray was a completely inadequate system of auxiliary ventilation. The method of auxiliary ventilation chosen was the fan-and-duct system, in itself an acceptable choice. However, problems arose not only from the overuse of series ventilation, but also from a combination of low airflows, ducting that was too small, incompatibility between the auxiliary fans and the choice of ducting, and ventilation ducts that were often split to service two headings at once, inadequately maintained, and, on frequent occasions, deliberately obstructed.

Airflow Requirements in Headings

The initial step in designing any system of ventilation in an underground coal mine is to assess the amount of air that will provide a safe and reasonably comfortable environment for mine workers. In the case of coal mines with significant emissions of methane, the airflow must be sufficient to dilute the gas at least to concentrations below the threshold limit values specified within the relevant regulations. Owing to the uncertainty involved in predicting rates of methane emission in specific work areas, it is prudent practice to design for gas concentrations well below the legal threshold limit values. At the points of emission, the atmosphere contains a very high concentration of methane that will therefore pass through the explosive range as it is diluted into the general body of air. Hence, a second consideration is that the air velocity should be sufficient to cause efficient mixing of the gas and to ensure that it does not accumulate as pockets or layers. Since it is lighter than air, methane tends to form layers along and under the roof of mine entries if not mixed into the air.[33]

[33] See the section on buoyancy effects and methane layers in Chapter 8, Methane.

On the basis of seam gas content studies, an average methane content of the coal in the Pictou coalfield was assessed to be 4.25 m^3/t.[34] If one assumes that 10 per cent of this is emitted before the coal leaves a heading, and that coal is mined at a rate of 6 tonnes per minute, the gas emission rate would be 0.0425 m^3/s.[35] If this is to be diluted to a general body concentration of 0.625 per cent (half the threshold limit value at which electrical power must be switched off), the required airflow is 0.0425/0.00625, or 6.8 m^3/s (14.4 kcfm).

Methods are available for calculating the velocity of the general airstream to prevent methane from layering along the roof, as is discussed in the section on methane layering in Chapter 8. However, considerable uncertainty exists about the proportion of methane that remains unmixed by the motion and turbulence caused by a mining machine and other face equipment. Furthermore, gas will be emitted not only from the coal actually being fragmented, but also from ribs and standing faces. Indeed, gas from these latter sources is more likely to stream upward to form a roof layer.[36] For these reasons, a pragmatic, though inexact, approach is to select an air velocity that experience has shown to minimize the formation of methane layers. In the United States, the minimum air velocity to be maintained in exhausting face ventilation systems is set at 60 feet per minute (0.3 m/s):

> **75.326 Mean entry air velocity.**
> In exhausting face ventilation systems, the mean entry air velocity shall be at least 60 feet per minute reaching each working face where coal is being cut, mined, drilled for blasting, or loaded, and to any other working places as required in the approved ventilation plan. A lower mean entry air velocity may be approved in the ventilation plan if the lower velocity will maintain methane and respirable dust concentrations in accordance with the applicable levels. Mean entry air velocity shall be determined at or near the inbye end of the line curtain, ventilation tubing, or other face ventilation control devices.[37]

There is no minimum air velocity mandated in the regulations governing coal mining in Nova Scotia.

For the large rectangular entries (6 × 3.5 m nominal size) that were driven at Westray, layering of air and gas streams occurs more readily. Hence, it would have been appropriate to employ the higher value of 0.4 m/s.[38] This gives the airflow required to inhibit methane layering to be 6 × 3.5 × 0.4, or 8.4 m^3/s (17.8 kcfm), indicating that it was the inhibition of methane layering, rather than gas dilution, that was the dominant factor in arriving at an appropriate airflow for headings at Westray. *None of the*

---

34 Jacques, Whitford and Associates Limited, 10 August 1984 (Exhibit 73.01); Suncor, 11 September 1984 (Exhibit 73.01); Algas Resources Ltd, March 1981 (Exhibit 73.03).

35 Norwest Resource Consultants Ltd, July 1986 (Exhibit 8, s. 13).

36 See figure 8.2 in Chapter 8, Methane.

37 *Code of Federal Regulations,* Title 30, Mineral Resources [30 CFR], Part 75, Mandatory Safety Standards – Underground Coal Mines (Washington, DC: Office of the Federal Register, National Archives and Records Administration, 1 July 1996).

38 Exhibit 8, s. 13.1.3.

measurements of auxiliary ventilation during the three months before the explosion reached 8.4 m^3/s.

In the Westray Coal Inc. Manager's Safe Working Procedures (G. Phillips, 15 December 1988), a minimum airflow of 2.5 m^3/s (5.3 kcfm) was required within 8.5 m of the face in coal driveages when no diesel equipment was in the heading.[39] Measurements of airflows in the ducts of the auxiliary ventilation systems were not undertaken each week or in all ducts. Those measurements that were reported in Eagles's weekly reports are shown in table 7.1. There appears to have been a distinct deterioration in airflows in auxiliary ventilation ducts during the 2½ months prior to the explosion. The limited frequency of sampling and the sparsity of sampling points make rigorous analysis impossible. However, we note that the 11 readings taken in February and March 1992 range from 5.2 to 17.0 kcfm, with an average airflow of 11.5 kcfm. The eight (different) points sampled in April and May had airflow ranging from 4.7 to 7.8 kcfm, with an average of 5.8 kcfm.

If the effects of leakage into the ducting are not taken into account, half of those airflows in the month preceding the explosion comply with the 2.5 m^3/s (5.3 kcfm) specified in the Manager's Safe Working Procedures of 1988. However, a comparison with the 8.4 m^3/s (17.8 kcfm), shown to be advisable in order to inhibit methane layering, indicates that the value given in the Manager's Safe Working Procedures was totally inadequate and that the measured airflows in the headings at Westray were only 26 to 44 per cent of the airflow required to inhibit methane layering. An airflow of 2.5 m^3/s distributed over a cross-section of 6×3.5 m gives an average air velocity of only 0.119 m/s, or 24 feet per minute. Such a velocity would be imperceptible to personnel, and the air would appear to be effectively stagnant. Dave Matthews, a Westray miner, described working conditions in the North A, B, and D Road headings: "There was air movement, but very little . . . the dust would be pretty stagnant, would stay around for a long while."[40]

Specifications for Auxiliary Ventilation

On 19 February 1992, Robert Parry, the maintenance superintendent, applied to the Nova Scotia Department of Labour (Mine Safety Division) for permission to use five auxiliary fans of motor power 37 kW (50 horsepower) and 18 auxiliary fans of motor power 18.5 kW (25 horsepower).[41] All these fans were described as being manufactured by Engart, used, and in good condition. The fans were stated as having flameproof motors designed for use in underground coal mines and had been certified by the British Certifying Authority in Buxton, England. Letters of approval for the fan specifications were issued by director of

[39] Exhibit 37a.118. This is page 1 of the Manager's Auxiliary Ventilation Plan – Coal Driveage.

[40] Hearing transcript, vol. 31, pp. 6525–26.

[41] The fans were denoted on the ventilation reports and accompanying maps as having motor powers of 40 and 20 horsepower, respectively.

mine safety Claude White on 4 March 1992.[42] The documentation did not indicate either the electrical frequency for which the motors had been designed or the rotational speed of the impellers. The manufacturer's specification data indicated that all the fans ran at a speed of 2,850 rpm.[43] Because this was a British specification, it is implied that this speed was attained at an electrical frequency of 50 Hz (current alternating at 50 cycles per second).

These fans had been purchased as used equipment from British Coal (formerly the National Coal Board) in the United Kingdom. The motors had been designed to run on an electrical frequency of 50 Hz. The standard frequency of electrical power supplied by utilities in North America is 60 Hz. The speed of rotation of a fan is proportional to the electrical frequency; hence, when supplied with 60 Hz electrical power, it would run at 20 per cent above its design speed.[44] This would have several consequences. First, when connected to a duct of a given (fixed) resistance, the fan would pass 20 per cent more air (60/50). Second, the pressure developed by the fan would increase by 44 per cent ($(60/50)^2$). Finally, the air power developed by the fan would rise by some 73 per cent ($(60/50)^3$). The last effect would place an abnormally heavy load on the motor and reduce its life considerably. Auxiliary fans burned out frequently at Westray, resulting in the motors' having to be rewound.[45]

Section 71(9d) of the *Coal Mines Regulation Act* states:

> An auxiliary fan may be installed or operated in a mine only on the written permission of an inspector and, after such fan has been installed it shall be situated on the intake side and at least 20 feet out by the last open cross-cut or entrance to the place being ventilated.

The location of the fan specified in this regulation clearly indicates that a forcing system should be employed. Similarly, forcing fan-and-duct systems of auxiliary ventilation were specified to be used at Westray in the Manager's Safe Working Procedures.[46] Notwithstanding those legal and company mandates, a letter sent by Kevin Atherton, senior mine engineer at Westray, to Albert McLean of the Department of Labour on 15 October 1991, requested permission to use auxiliary fans in both forcing and exhaust configurations.[47] The letter indicated a number of limitations that would be imposed on the use of auxiliary fans. Three of these were that fans would be operated continuously, that they would be operated such that recirculation did not take place, and that ducting would be maintained to "within 10 m of the face, or as required to deliver an adequate supply of air to within 5 m of the face." The weekly ventilation reports indicate

---

42 Exhibit 69b.155–62.

43 Exhibit 38.12.

44 John Bossert discussed this point in his testimony (Hearing transcript, vol. 12, pp. 2103–04).

45 Mick Franks testified to this effect (Hearing transcript, vol. 21, pp. 4153–54; vol. 22, p. 4199). Harvey Martin also noted the frequency of auxiliary fan burnout (vol. 23, pp. 4496–97).

46 Exhibit 37a.126, 129–32. Diagrams 1, 4–7, show the forcing system only.

47 Exhibit 37a.136–7.

that, in practice, these conditions were not fulfilled. The Inquiry has been unable to trace a written response to that letter. During the three months preceding the explosion, there was only one location in which a forcing system of auxiliary ventilation was used (see the earlier section in this chapter, Throughflow Ventilation in the Southwest Sections (19 February, 26 February, and 4 March 1992)). All others employed an exhaust arrangement.

The relative merits of forcing and exhaust arrangements of auxiliary ventilation and of fan-and-duct systems were discussed earlier in this chapter. The inbye end of a line of ducting should be maintained as close as practicable to the face. As noted in his letter to McLean, Atherton specified that the duct would be maintained "as required to deliver an adequate supply of air to within 5 m of the face." It is impractical, however, to locate the inbye end of the duct so far forward that it interferes with the operation of the continuous miner. The distinct advantage of a forcing system in gassy conditions is the jet of air projected forward beyond the end of the duct. Conversely, an exhaust system gives poor control of airflow at the face (see figure 7.8). Therefore, in addition to remaining in compliance with the law, a forcing system of auxiliary ventilation would have been preferable at Westray. Because exhaust ventilation was used in all but one of the auxiliary systems at Westray, it became even more important to maintain the end of the ducting close to the face. In one of the airflow measurement reports, a duct was reported as being approximately 30 m back from the face.[48]

In the design of a fan-and-duct system, the size and configuration of the ducting must be compatible with the fan capacity. There are several approaches. Each begins with specifying the airflow. To diminish methane layering in the headings at Westray, an airflow of 8.4 m^3/s (17.8 kcfm) has been suggested earlier in this chapter as appropriate. One guideline, used to provide acceptable pressures within the ducting, is to assume an air velocity of 10 m/s in the duct. The standard diameter of ducting that will produce that airflow and velocity is 1,050 mm (42 inches). The 37 kw fans were capable of passing the required airflow of 8.4 m^3/s.[49] However, the ducting used at Westray was 750 mm (30-inch) diameter spiral-reinforced flexible tubing. That gave a resistance to airflow of more than five times greater than the larger-diameter duct of the same construction ($(42/30)^5$).

The smaller-diameter ducting would fail to allow the required airflows when the fans that were provided were used. Furthermore, the higher-powered auxiliary fans would have given static pressures in excess of 2 kPa (8 inches water gauge) for duct resistances that resulted in airflows of less than 6.6 m^3/s (14 kcfm).[50] The majority of auxiliary airflow measurements were considerably below this figure and resulted in excessive suction pressures being applied to those sections of ducting

48 18 March 1992 (Exhibit 37a.067).

49 Performance curve for Model B70 (Exhibit 38.12).

50 Performance curve for Model B70 (Exhibit 38.12).

closest to the fan. Furthermore, if a fan had been running at a speed greater than its design value of 2,850 rpm owing to its being supplied with 60 Hz electrical power, the situation would have worsened. Not surprisingly, inward collapses of ducting were common at Westray. The matter of the type and size of ducting was raised as a concern by the engineer responsible for airflow measurements. Eagles had talked to the underground manager "about using three-inch pitch next to the fan to try to prevent the problem from happening."[51] He commented to the Inquiry that "they could have gone to a rigid duct . . . constructed out of an anti-static fibreglass . . . which would have been the ideal solution."[52]

Use and Maintenance of Ducting

The spiral-reinforced ducting of the type used at Westray is manufactured from a flame resistant material sewn around a spring-steel-wire spiral. The spring steel keeps the ducting open when it is bent around corners and when it is subjected to a negative gauge pressure, as in an exhausting auxiliary ventilation system. The pitch (spacing between spirals when fully stretched) is normally 150 mm (6 inches) for ducting in a forcing system and 50–100 mm (2 to 4 inches) for exhaust applications. At Westray, the ducting was hung from roof screen by hooks attached to grommets in a reinforced webbing running along the top of the tubing. The ducting was supplied in 7.6 m (25-foot) lengths. The couplings between sections of ducting were made of circles of wire rope sealed into the end of the material. Those circular bands could be inserted one inside the other and reinforced by metal clamps.

Damaged ducting, common at Westray, was mentioned in "ventilation survey" reports and during the testimony of mine personnel.[53] The damage would have inevitably resulted in leakage of air into the duct, leaving the headings, already with dangerously low ventilation, with even further reductions in their airflows. It was also in contravention of section 71(9e) of the *Coal Mines Regulation Act*.[54]

The incompatibility between the ducting and the higher-duty fans is also reflected by the number of examples of ducting damaged by inward collapse. For example, the night-shift foreman for Southwest 2 section reported on 7 May 1992: "v-tube collapse on new fan."[55] Such a collapse would flatten and effectively close the ducting against passage of air. The reaction of the fan would be to increase the suction pressure and seal the collapsed section even more tightly. The fans could possibly operate in a

---

51 Hearing transcript, vol. 76, p. 16510.

52 Hearing transcript, vol. 76, p. 16544.

53 Trevor Eagles (Hearing transcript, vol. 76, pp. 16510–13, 16543); Don Dooley (vol. 36, pp. 7836–37).

54 ". . . air ducts or tubing shall be maintained in such condition as to minimize air leakage and to ensure an adequate supply of air being delivered within fifteen feet of the face."

55 Exhibit 42h.0061. Buddy Robinson also confirmed collapsed ducting (Hearing transcript, vol. 30, p. 6444). Eagles discussed the matter in some detail (vol. 76, pp. 16422, 16496, 16507, 16509–11, 16531).

stalled condition in these circumstances, resulting in increased noise levels and vibration. The effect of collapsed ducting would be to reduce the ventilation of the headings to near zero.

As indicated on the ventilation maps produced to illustrate the weekly airflow measurements,[56] tee-jointed ducts were used frequently. In this arrangement, a single duct, connected at one end to an exhausting fan, was divided further inbye into two ducts in order to draw air from a pair of adjacent headings. A prefabricated flexible tee-shaped (or "y"-shaped) piece was used at the junction of the ducts. This is an acceptable practice, provided the single duct and the fan to which it is connected both have the capacity required to ventilate two headings adequately. This was not the case at Westray. Indeed, no duct passed the 8.4 m^3/s deemed appropriate to ventilate even a single heading properly. While all fan-and-duct systems should be well engineered, a careful design is particularly important when a tee-jointed arrangement is to be used.

The most disturbing treatment of ventilation ducting in the Westray headings was the deliberate obstruction of one side of a tee-jointed duct system. In the pairs of headings that were ventilated in this manner, it was common for roof bolting to be in progress in one heading while active mining was carried out in the other. Because of the inadequate auxiliary ventilation, methane concentrations often rose to unacceptable levels. While such levels were dangerous in both headings, they were particularly so in the heading where mining was taking place, on account of the concentration of equipment and friction at the pick points of the continuous miner. Furthermore, when properly fitted to the continuous miner, a methanometer would cut off electrical power from the machine at a preset level of gas concentration. This caused frequent interruptions to the production of coal. In those circumstances, a common practice at Westray was to restrict airflow to the duct in the roof bolting heading in an attempt to improve the ventilation in the adjacent mined heading. Wyman Gosbee recalled roof bolting in the SW2-1 heading on 7 May 1992 – less than 48 hours before the fatal explosion – when his foreman, Arnie Smith, came into the heading and "he took the piece of rope and he wrapped it around the vent tube, and he just cinched it up, choked it right off." When Gosbee asked him what he was doing, Smith replied that " 'we need the air over there to mine.' "[57]

Restrictions were applied in one of two ways. The first was to apply a wire screen to the inlet end of the duct and to cover it with plastic sheeting, effectively sealing the duct. The second was to restrict the ducting partially, by tightening a length of wire around it in the manner of a tourniquet. In either case, the practice can be described only as extremely

56 Exhibit 45.01.

57 Hearing transcript, vol. 25, p. 5022. A number of other witnesses also testified about blocking vent ducts: Eagles (vol. 76, p. 16511); Wayne Cheverie (vol. 21, pp. 4046–47); Lenny Bonner (vol. 24, pp. 4785–89); Doug MacLeod (vol. 27, pp. 5647–51); Randy Facette (vol. 33, p. 7241); Don Dooley (vol. 36, pp. 7862–66); Jay Dooley (vol. 39, pp. 8657–61 and vol. 41, p. 9129); Bryce Capstick (vol. 42, pp. 9382–83).

hazardous and foolish. The total, or partial, restriction of ventilation to a heading would have been imprudent even in an inactive heading at Westray, because of gas emissions. To restrict ventilation while operations were in progress was patently reckless.

Finding

The auxiliary ventilation system at the Westray mine was defective in several ways. Some of the more hazardous defects were:

- **It was ineffective in removing the methane from the working face.**
- **The exhaust system of auxiliary ventilation (used in all but one location) was contrary to the *Coal Mines Regulation Act* and Westray's own Manager's Safe Working Procedures.**
- **In most cases, the ventilation ducting was too small for the size of the auxiliary fans. This situation resulted in high resistance in the ducts and excessive suction, which caused collapsing of the ducts and loss of ventilating air to the working faces.**
- **Poor airflow to the face permitted the accumulation of high levels of methane, which, in turn, caused the continuous miner to shut down until the methane was cleared and safe operating levels attained. To alleviate this gas accumulation and direct more intake air to the working face, miners would, on occasion, block the ventilation ducting serving the roof bolters – a reckless and foolhardy practice.**

Concluding Comments on the Ventilation at Westray

In his commentary on the evidence following the hearings, Inquiry ventilation expert Malcolm McPherson made the following observations:

> Minimal thought seems to have been given to the planning and control of ventilation in any section of the mine. Indeed, little was done in accordance with either the law or prudent practice. Decisions appear to have been made in an ad hoc manner to produce a temporary alleviation of an immediate problem. The result was a series of "band-aid" fixes, each of which led to an even greater difficulty. The problems were self-accumulating and created a very high probability that the mine would suffer a major hazardous incident early in its life.
>
> Recommendations made by the engineer who took the airflow measurements were either ignored or delayed for inordinate periods of time. The weakness of mixing engineers' responsibilities between mine production and matters relating to safety, and in not engaging persons totally dedicated to ventilation and safety, was demonstrated all too tragically at Westray.

Ventilation Planning

The planning of ventilation for an underground mine is a vital component of the overall mine-planning procedure. Unless the airflow system is carefully engineered, it will lack efficiency and effectiveness throughout the life of the mine, reacting adversely on mine productivity and perhaps causing premature cessation of mining. A poorly planned system may result in the underground environment's not meeting minimum legal requirements. It may also lead to both short- and long-term health problems for mine workers and, at worst, result in the tragedy of a mine disaster involving multiple fatalities. Ventilation planning should therefore be carried out by people knowledgeable about the appropriate legislation and skilled in the discipline of underground mine ventilation engineering. Planning of the subsurface airflow systems is an essential part of the total mine-planning procedures and should be integrated into matters relating to mining layouts, production targets, ground control, and types and sizes of equipment to be employed.

The need for ventilation planning does not stop when a new mine becomes operational. The condition of the ventilation system must be checked regularly and, when appropriate, the long-term plans must be updated to take account of variations in the mining layout, in airway conditions, or in emission rates of airborne contaminants. In this section of the chapter we discuss, first, the initial ventilation planning of a proposed new mine and, then, the ongoing procedures of ventilation planning once the mine has entered into production.

Ventilation Planning for a New Mine

A proposed mining operation begins with feasibility studies before a decision is made to proceed. The next phase, the engineering phase, will involve a detailed investigation of all the technical aspects of a modern mining undertaking. During this phase, ventilation planners should work closely with others who are designing alternative mining layouts and the step-by-step development of the mine for as many years as can sensibly be foreseen, given proven reserves and established markets. The ventilation planners are particularly concerned with the length, number, size, and interconnectivity of the underground openings. These are also important matters to ground control engineers, who are responsible for the stability of those same openings, and to operations planners, who must ensure that the entries satisfy the requirements of moving equipment, traffic control, and transportation of the mined material. A number of iterations are usually undertaken before all requirements for the underground structure are satisfied.

An initial task for ventilation planners is to assess the airflow requirements at working faces as well as at other areas and facilities requiring ventilation, including underground workshops, electrical equipment, transportation routes, and pump stations. The heaviest demands for airflow occur at the working faces. Using data that may have been obtained from borehole samples or neighbouring mines, engineers estimate probable methane emission rates. They also assess dust, heat, and humidity. Diesel exhaust emission rates can be calculated from machine specifications. Calculations can then be carried out to determine the airflows required to dilute and remove those contaminants in a safe and effective manner.

A number of time phases projecting through the predictable future of the mine should be selected. These will typically include the early stages of development, the initial production, and then two-year intervals, extending to five-year intervals, throughout the planned life of the mine. The ventilation planners will concentrate on periods of particularly heavy demand on the ventilation system. These will include periods when faces are at their greatest distance from main airways, and times of transition – when additional districts may require ventilation simultaneously. For each of the time phases, the airflow requirements should be determined and indicated on a corresponding map. The next step is to determine the optimum location and duties of main fans and whether modifications will be necessary to the initial estimates of the number and sizes of airways. In the case of a relatively simple mining layout, such as the one at Westray, the location of the main fan(s) is often prescribed.

The methods used for planning mine ventilation layouts have changed radically since the mid-1960s. Earlier methods relied on summing pressure differentials along the longest intake and return routes, based on assumed airflows, to arrive at a rough assessment of required fan pressures. Since the 1960s, computer simulation packages have become readily available, enabling ventilation planning to be carried out with unprecedented precision and speed. Mine ventilation network analysis by computer has been conventional textbook material since 1982, and it is now routinely practised throughout the world.

Such analysis concentrates initially on the main ventilation structure rather than on auxiliary ventilation systems. Identifying numbers are allocated to junctions of airways in the layout under investigation. Data relating to all branches represented in the ventilation network are then entered into the computer. For a proposed new mine, the data involve the length and dimensions of each branch, together with a measure of the roughness of the airway lining and matters relating to bends or other factors that introduce additional resistance to airflow. Fans of selected pressure-volume characteristics may be entered at locations throughout the network. The computer can then rapidly produce tabulated listings and graphic depictions of the network, showing the distributions of airflows, pressure differentials, airway resistances, operating costs, and other parameters that may suggest improvements to the system. The ventilation planner can make modifications and rerun the simulation in seconds. Through such means, alternative ventilation layouts can be investigated rapidly to arrive at an optimum system.

Each simulated ventilation system that appears to be practicable and efficient is subjected to a number of acceptability checks. The predicted system is examined to ensure that it complies with all legal requirements. Air velocities must lie within specific ranges. If they are too low, problems may arise from inefficient mixing and delays in the removal of gases or airborne dust. Air velocities close to the normal travel speeds of diesel vehicles should be avoided so that drivers do not remain within the exhaust fumes when travelling in the same direction as the airflow. Air velocities that are too high will give rise to dusty conditions, causing discomfort and creating health hazards. Considerations of air velocity may necessitate changes in the number and sizes of mine entries. If the airflow requirements initially established for a particular time phase have been met, the dilution of contaminants to safe and legally acceptable levels will have been satisfied. These levels are subject to an additional check, however, in light of predicted airflow patterns.

Checks are carried out on alternative escapeways from the mine in the event that normal travel routes become inaccessible. Ventilation network analysis exercises should be carried out to investigate the travel paths of smoke and toxic gases from fires that might occur at critical locations within the system. These exercises are valuable for designating selected routes as escapeways and for choosing locations for refuge chambers underground.

The economics of the system should also be analysed in order to optimize between capital and operating costs – whether, for example, it makes sense to pay for a larger main entry, with consequent lower fan-operating costs. The main fans should be selected so that they can handle the wide range of duties that may be required over their life span. Similarly, the number and size of main entries to be driven should suit an economically acceptable period of time, such as the 15-year projected lifespan of the Westray mine. The series of analyses conducted for each period should be monitored for continuity between time phases. For example, if one knows that a new surface connection will be required at some future date, then it may be more cost effective to drive it before it becomes absolutely necessary. The speed and versatility of ventilation simulation software have enabled this level of detail in planning to be used routinely.

Initial Ventilation Planning for the Westray Mine

The ventilation planning carried out before construction of the Westray mine was limited to the relevant sections in three feasibility studies: The first of these, entitled "Suncor Inc. Pictou County Coal Project Feasibility Study," was prepared by Norwest in 1986.[58] Section 13 of that study dealt with ventilation of the mine. Although Norwest used the older, traditional techniques of mine ventilation planning, its treatment of ventilation was the fullest of the three studies.

The Norwest study refers to methane desorption tests conducted by Suncor on borehole samples of coal seams in the Pictou area. A large-scale study had been carried out by Novacorp Engineering (formerly Algas Resources Ltd) on the Pictou coalfield in 1980. While the maximum value of total gas was recorded as 6.6 m^3/t (cubic metres per tonne), the average value was 2.4 m^3/t, and only 6 per cent of the samples showed gas content

[58] Exhibit 8. An earlier report prepared by Norwest for Suncor, submitted in January 1985 (Exhibit 12), involved access by vertical shafts and shortwall mining methods. Section 9.6 of that report gives a two-page outline of the ventilation arrangements envisaged for that mining layout.

in excess of 5 m^3/t. Norwest chose an average gas content of 4.25 m^3/t for the purposes of assessing ventilation needs. This seems a reasonable assumption in the absence of empirical values obtained from mining experience in the area. Norwest assumed that 10 per cent of the total gas content would be emitted into the working face, based on a residence time of 10 minutes before fragmented coal was removed from the face area. No separate account was taken of emissions from standing faces or ribs.

For room-and-pillar faces, Norwest assessed a required airflow of 4.8 m^3/s (10.2 kcfm) to dilute methane to a concentration of 0.9 per cent. A preferred technique would have been to base airflow requirements for methane emissions on a dilution to one-half the mandatory threshold limit value (TLV) at which electrical power must be shut off. In the case of Nova Scotia, this TLV is 1.25 per cent methane. The corresponding airflow in a room-and-pillar heading (using Norwest's assumed gas emission rate) becomes 6.8 m^3/s, or 14.4 kcfm.

The Norwest report was the only study that considered air velocity. To prevent the layering of methane along the roofs of headings, Norwest proposed a minimum air velocity of 0.4 m/s, which was entirely reasonable, although a higher velocity may have been required in some areas to prevent the formation of methane layers. Unfortunately, the study did not take the next logical step of recommending the corresponding airflow, by multiplying the minimum velocity by the cross-sectional areas of the entries.

Norwest commented on tests that had previously been conducted on the propensity to spontaneous combustion of the Foord and Cage seams. These tests seem to have been based entirely on experiments carried out on samples of the coal and appear to ignore the many other mining and atmospheric factors that also influence the susceptibility of any given coal mine to spontaneous combustion. The authors of the Norwest study appear unimpressed with the reliability of those tests.[59]

In designing a mine ventilation system, it is inadvisable to locate a conveyor where it will pass through a main airlock, because this passage can raise coal dust in addition to being a source of air leakage. Since the Westray slope conveyor was to be located in the return airway, Norwest recommended a forcing system of ventilation, with the main fan located near the top of the intake slope. Personnel and materials would have to pass through an airlock, but the conveyor would be left unimpeded.

The Norwest study projected total airflow requirements for each of the first four years in the life of the mine. These forecasts were based on assumed air leakages – a traditional but imprecise method. Similarly, the old technique of summing pressure drops was used to estimate the corresponding pressures to be developed by the main fan. In the fourth year, the fan duty was projected to rise to an airflow of 155 m^3/s at a pressure of 2145 Pa. The cost of heating the intake air by propane during the winter months was also addressed.

---

59 Exhibit 8, s. 13.2.

The Norwest study included a calculation on the fans and ducting required for the auxiliary ventilation of headings. To accommodate a medium-sized supply vehicle in a heading, the recommended minimum airflow at the face was increased from 4.8 to 6.6 m^3/s. For that airflow, Norwest suggested a duct diameter of 915 mm (36 inches). If we ignore the effects of leakage, this size implies an air velocity in the duct of 10 m/s, which agrees with the design value often used as a first approximation for duct air velocity. Using an airflow of 8.4 m^3/s to reduce the risk of methane layering (as discussed earlier in this chapter, in the section on Auxiliary Ventilation Systems Used at Westray), and employing a nominal duct velocity of 10 m/s, we arrive at a required minimum diameter of ducting of 1,034 mm. In practice, this would be rounded up to the nearest standard duct size of 1,050 mm (42 inches).

Section 71(9c) of the *Coal Mines Regulation Act* requires that auxiliary fans not draw more than 40 per cent of the airflow available at the inlet to the duct. Taking this figure and duct leakage into account, the Norwest study arrived at an airflow of 23.75 m^3/s to be available at the outbye end of a heading. If the value of 8.4 m^3/s had been used for the heading airflow, and allowing a duct leakage of 20 per cent, the corresponding estimate of available airflow would become $8.4 \times 1.2/0.4 = 25.2$ m^3/s, or 53.4 kcfm.

A second feasibility study was assembled by Placer in July 1987.[60] The section on ventilation was limited to a single paragraph and a "general ventilation flow" figure that shows a few airflows without background justification. Again, reference was made to the tests for gas content of the coal. A main forcing system of ventilation was chosen, with a fan duty of 175 m^3/s at a pressure of 2,080 Pa. Each of the mine sections was to receive an airflow of 19 m^3/s. Here again, there is no indication of how these values were determined. The treatment given to ventilation in this study can only be described as trivial.

The third study, undertaken for Westray Coal Inc. by Kilborn, was submitted in 1989.[61] Section 3.5 of that report deals with ventilation of the mine and is limited to one page. The treatment is simplistic. It assumed a coal production of 450 t per section, producing gas at a rate of 6.2 m^3/t. This gave a gas emission rate that necessitated an airflow of 20 m^3/s to dilute it to a concentration of 0.5 per cent. For five mining units, the required airflow was, therefore, 100 m^3/s. A volumetric efficiency of 55 per cent was assumed (airflow usefully employed divided by airflow at the main fan), giving a main fan airflow of 180 m^3/s. The corresponding fan pressure was stated to be 2,100 Pa, based on the summation of frictional pressure drops of 800 Pa in each main slope and 500 Pa across

60 Exhibit 10.2, "Pictou Project, Feasibility Study: Volume 2 – Mining." This document appears to have been put together from contributions by Placer US, Suncor Inc., and Associated Mining Consultants Ltd. (AMCL), with information drawn from previous documentation by Golder Associates (1984–86), Norwest Mining Consultants (1986), AMCL (1987), Nova Scotia Mines Inspection Reports (1873–1951), the Geological Survey of Canada (1987), and a Suncor geological report (1986).

61 Exhibit 4, "Technical and Cost Review of the Pictou County Coal Project: Nova Scotia."

the split of maximum pressure drop. These values were not backed up by calculations shown in the report. As in the Norwest study, a forcing system of ventilation was recommended, with the main fan sited at the top of the intake slope.

In the case of headings, an airflow of 5 m^3/s was stated to be required at each working face, with ducting sizes of 600 mm diameter (approximately 24 inches). Again, no justification was given for these values. This study recommended the installation of an underground environmental monitoring system to monitor concentrations of carbon monoxide and methane at strategic locations and to record those parameters continuously at the mine surface.

In summary, the treatment of ventilation in the 1986 Norwest study was of the nature of an initial overview that might be produced in preparation of a full ventilation-planning study. The single paragraph on ventilation in the 1987 Placer report indicates that ventilation was not treated as a serious part of that study. And the Kilborn treatment of 1989 was elementary and indicates that little effort was made to analyse ventilation. Feasibility studies for any proposed new mine should, as the term suggests, entail investigations into whether the project under consideration is feasible from every consideration – technical, financial, human resources, marketing, and environmental. The results of feasibility studies provide sufficient information for a decision on whether to proceed with the project. If that decision is positive, a comprehensive engineering study should be initiated to provide the detailed specifications for every technical aspect of the work.

There appears to have been no organized and documented engineering study carried out for Westray. Hence, there is no record of properly constituted ventilation plans having been produced. In the absence of such plans, it is almost ludicrous that the Westray mine could be approved either by the financing agencies or by the regulators. We reviewed an example of a ventilation plan submitted for approval under the U.S. *Code of Federal Regulations*. The plan is a 75-page document of impressive scope and detail. It comprises monitoring and degasification procedures; detailed drawings of typical seals, doors, dust and methane control methods, and bleeder systems; drawings of typical mining and ventilation procedures; a detailed cutting sequence plan, including bolting and line brattice detail; equipment lists; dust sampling plan; and a procedure for maintenance of underground workings during monthly scheduled fan stoppages.[62]

Such plans are a necessary component in ensuring that the mine can develop safely, efficiently, and productively over its predicted life.

---

62 Jim Walter Resources, Inc., Blue Creek No. 3 Mine, "Ventilation System and Methane and Dust Control Plan," (Adger, Ala.: JWR, Inc. 1995). This six-month update was approved by MSHA under 30 CFR 75.370 in January 1996. Charles Dixon, senior vice-president of engineering with JWR, provided the Inquiry with a copy of the complete plan.

Finding

Ventilation planning for the Westray mine did not address the requirements for a comprehensive system of fresh-air circulation and methane removal. The plan on which the ventilation was based was merely a brief outline in a feasibility study. A comprehensive engineering study by competent ventilation experts was not completed and documented before approvals were requested. The regulating agency, in this case the Department of Natural Resources, could not assess the efficiency or the safety of the ventilation system of the proposed Westray mine.

Ongoing Ventilation Planning and Control

The essential difference between a mine ventilation system and the ducts that may appear in a surface building is that a mine is a dynamic entity, continually changing. Individual entries change their shape and cross-section as a result of strata stresses as well as movement of equipment and stored material. Stoppings, doors, and other ventilation controls have their leakage characteristics modified through usage and, again, movements of the strata. As the mine progresses, sections become depleted of mineral reserves and are abandoned while new sections are opened up. Throughout the life of an underground mine, it is necessary to maintain ongoing vigilance about the changing nature and geometry of the airflow network. Two separate sets of procedures should be undertaken by engineers responsible for the ventilation of a mine. First, measurements of airflows should be made at relatively short time intervals and at strategic locations. Second, at greater time spans it becomes necessary to undertake full and detailed ventilation surveys of frictional pressure drops and corresponding airflows in order to provide accurate data for the longer-term planning of the mine ventilation system.

Routine Measurements

At Westray, routine measurements of airflow were made at weekly intervals, from February 1992 onward. That should be an acceptable frequency in most circumstances. Additionally, it is prudent to require section supervisors to take airflow measurements at the beginning of each shift. This is a mandatory part of preshift examinations in the United States.[63] The purpose of these weekly and daily airflow measurements is to ensure that all places in which personnel work or travel receive enough air to provide a safe and legal atmosphere. A secondary reason for the weekly measurements is to check that the actual ventilation of the mine follows, and is in reasonable agreement with, the prescribed ventilation plans.

While taking routine measurements, the mine ventilation engineer should also note and make observations on fan pressures, pressure differentials between intake and return entries, and such measurements of

[63] 30 CFR 75.360.

air quality that may be of concern. In a coal mine, the air-quality measurements will include concentrations of methane and, perhaps, airborne respirable dust. Notes should also be made on the conditions of doors, stoppings, regulators, and air crossings. Necessary repair work should then be carried out expeditiously. Those control devices are subject not only to strata stresses, but also to penetration for pipes and cables. Air leakage in such circumstances can reach serious proportions. Auxiliary ventilation systems should also be inspected for damaged and restricted ducts, and for excessive leaks. While such checks and observations were recorded at Westray by Eagles, little action was taken in response to the problems that he reported.

Ventilation Surveys

We have outlined the modern procedures of ventilation planning for a proposed but yet unconstructed new mine. In the absence of hard data, it was necessary to describe the condition of each proposed airway in terms of length, cross-section, and an estimated friction factor that relates to the surface roughness of the airway lining. Friction factors for the various airways are selected from empirically derived values listed in the literature of mine ventilation. All of those geometric parameters are used to assess a *predicted* resistance that each airway will offer to the passage of air. As the main entries of the mine come into existence, it is prudent to measure their *actual* resistance. This is accomplished by conducting a *ventilation survey*.

At Westray, the weekly routine measurements of airflows were misnamed "ventilation surveys." A true ventilation survey is a carefully organized procedure, well managed and subject to quality assurance checks to minimize the chances of error. The procedure involves selecting one or more routes around the structure of throughflow airways, each route commencing and finishing at the same junction (that is, following a closed loop). Measurements are taken with newly calibrated instruments that allow accurate values of air volume flow and frictional pressure drop to be established. A network of the actual mine ventilation system is established as a computer model. The purpose of ventilation surveys is to provide and update the data required for ongoing ventilation planning.

Ventilation surveys should be conducted by personnel who are well trained and experienced in mine ventilation. New surveys are required throughout the life of the mine for a number of reasons. First, as discussed, the geometry and, hence, the resistances of individual entries vary with time and from diverse causes. Second, new airways are added and older ones removed from the ventilation infrastructure as the mine progresses through its life. Third, ventilation controls also change with time and location: doors and stoppings alter their leakage characteristics, and fan performance changes as a result of impeller wear, accretions, and erosion. The interval between surveys depends on the rate at which physical changes occur in the mine, but is typically six months to one year. One further factor that necessitates periodic review of ventilation plans is that mine development may deviate from the layouts initially intended and on which the original ventilation plans were based.

There were no ventilation surveys, if one uses the term correctly, carried out at Westray. Such surveys should have been conducted at the time the main slopes were completed down to No. 9 or No. 10 Cross-cut, and again following the development of the main airways into the Southwest and North sections. Westray did acquire ventilation network analysis software, and it employed it, between August and December 1991, in an attempt at long-term planning. However, the input was based on geometric and literature values rather than on real measured data. In the event, the haphazard development of the mine rendered such efforts fruitless.

Splitting

Generally, more than one mining crew is working in the mine at any one time. At Westray, at least two crews were working at different areas or working faces. This introduces an added complication in ventilation planning since it is essential that each crew get a constant supply of fresh air. This process of providing a separate air supply to each working face is called "splitting":

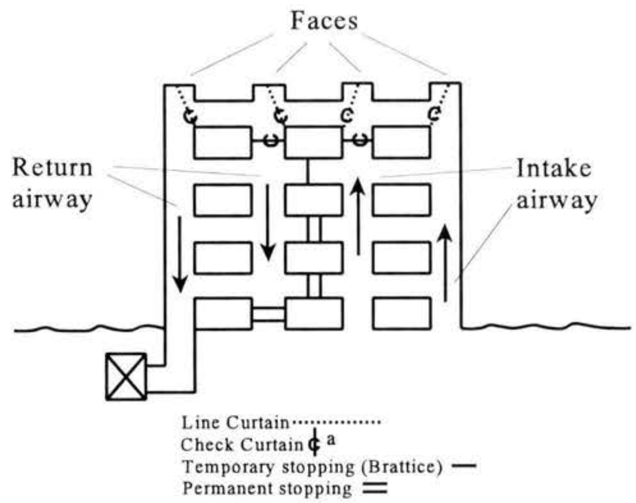
> Splitting the airflow is necessary for safety as well as to minimize power costs. Placing each working section on a separate parallel split insures that each crew will have a fresh air supply, uncontaminated by dust and gas accumulated on a previously ventilated section. . . . [B]y regulating splits, control of the local ambient conditions is possible. Without regulation, the constantly moving sections offer a resistance to the flow of air too variable to allow for reliable ventilation. Within a split, if both development and pillar line must be ventilated, mining should be planned in such a manner that the development is ventilated first and the pillar line last so the return air passes directly into the gob.[64]

Intersections or cross-cuts along the roadway must be controlled both to increase the efficiency of the airflow to the face and to avoid mixing the fresh intake air with the return air. Figure 7.11 shows several methods of controlling air flow to room-and-pillar faces.

All these factors and techniques have as an objective the provision of an adequate fresh air supply to the mine, especially to the working face where dust and methane constitute the most immediate hazard. One might assume that the higher the velocity of the air current, the more air gets into the mine and the better it is for the health and safety of the miner, because the higher velocity will quickly remove methane and thus increase safety. This is not the case. The velocity of the air must be carefully balanced so that it does not unduly interfere with the settling of the coal dust. If coal dust is agitated by the airflow, it will increase the amount of airborne dust and thus degrade the respirable quality of the air. In addition, the more

64 Robert Stefanko, *Coal Mining Technology: Theory and Practice* (New York: Society of Mining Engineers, 1983), 59.

Figure 7.11 Mine with Controls



Source: United States Department of Labor, Mine Safety and Health Administration, *Mine Ventilation*, Safety Manual No. 20 (Washington, DC: MSHA, 1991).

[a] A check curtain is a temporary stopping (such as the conveyor-belt stoppings used at Westray) that allows the passage of equipment and personnel.

dust that is held in suspension in the air, the more danger of dust-propagated explosions.

Finally, splitting "also has the desirable effect of separating the various portions of the mine into sections with regard to airflow and thereby minimizing the likelihood of an explosion propagating from one section to another."[65] Obviously, this effect did not operate at the time of the Westray explosion since all crews, in all sections, were killed instantly by the blast.

Responsibility for Mine Ventilation and Safety

In some jurisdictions, including the United Kingdom and South Africa, ventilation and safety departments are a recognized, accepted, and separate part of the engineering staff of underground mines. Following a series of mine explosions, similar arrangements are being contemplated in Australia. In North America, with a few notable exceptions, responsibilities for mine ventilation are allocated to engineers who also have duties relating to mine production. This assignment often gives rise to a conflict between those two charges. Recurring inadequate ventilation will inevitably result in loss of production. However, until conditions become untenable, that loss may not be immediate. Mining can continue for a time, and at increased risk, with deteriorating ventilation. Conversely,

65 Stefanko, *Coal Mining Technology*, 59.

problems with equipment or roof control are likely to result in immediate cessation of mining. Hence, when a conflict between ventilation and production responsibilities arises, it is far more likely that the production-related problems will receive priority. This was certainly the case at Westray.

In those companies that have structured mine ventilation departments, there should be at least one professional ventilation engineer who is well educated and trained in the discipline. The ventilation engineer should be responsible for the control and maintenance of the mine ventilation system, as well as longer-term planning, and should have assistance as necessary to conduct routine measurements and periodic ventilation surveys. Furthermore, the ventilation engineer should have a labour force sufficient for the routine construction and maintenance of doors, stoppings, and regulators. More labour-intensive but less frequent operations such as the building of air crossings or explosion-proof seals may require additional intermittent assistance from the general mine workforce.

The mine ventilation engineer should report directly to the mine manager or, at large operations, to the senior underground manager. These managers carry the overriding responsibility for the safety and health of all employees at mine level.

Westray was a classic case of the situation that can arise where no dedicated ventilation expertise is available and where ventilation matters are given a low priority in the face of immediate mining difficulties. It shows how such a philosophy can lead to tragic consequences. Airflow measurements and related inspections were carried out by a graduate engineer-in-training from February 1992 onward. He had been employed at Westray since May 1991.[66] His knowledge of mine ventilation was limited to a theoretical course at university and airflow measurement in a gold mine during student summer employment in 1990. He had not previously worked in a coal mine and was not, for example, initially made aware of the phenomena associated with methane layering. Trevor Eagles was conscious of the lack of an experienced ventilation engineer with whom he might discuss such matters at Westray.[67] He had very little authority, and his recommendations were largely ignored.[68] His experience at Westray underlines firmly the need to separate matters relating to safety from the direct control of those whose prime responsibility is for mine production. Eagles concluded: "Safety personnel underground, which includes your mine examiners, . . . ventilation people and . . . rock mechanics people, must be given the authority to shut down workplaces if they see something that's not right."[69]

---

66 Eagles (Hearing transcript, vol. 76, pp. 16413, 16415).

67 Hearing transcript, vol. 76, pp. 16427–28.

68 Hearing transcript, vol. 76, pp. 16464, 16515, 16555, 16567, 16641, 16646.

69 Hearing transcript, vol. 76, pp. 16593.

The employment of people wholly engaged in mine ventilation or safety-related matters is not widespread practice in North American mines because those personnel appear not to contribute, directly and in the short-term, to mine productivity. They may be mistakenly regarded as non-essential "overhead" staff. In reality, they are no less essential than aircraft safety personnel employed by an airline. Since many mines in the private sector have shown little inclination to engage full-time ventilation or safety staff, this change will come about only by legislative action directed at mines with labour forces above a specified size.

RECOMMENDATIONS

10 The overriding principle in mine ventilation must be that the mine is properly ventilated at all working times. It is the primary duty of the mine manager to ensure this proper ventilation.

(a) All active working places should be ventilated by a current of fresh air containing not less than 19.5 per cent by volume of oxygen and not more than 0.5 per cent by volume of carbon dioxide.
(b) Each working face should receive fresh air of sufficient volume and velocity to dilute and render harmless all noxious or flammable gases and maintain all working and travelling areas in a safe and fit condition.

11 No mine should start up without a comprehensive ventilation plan approved by the regulator. The ventilation plan should be subject to at least an annual update, and any changes in the interim should be subject to approval by the regulator.

12 The ventilation plan should contain details of the system proposed, or of amendments to the existing approved system, and should indicate:

(a) the limits of the mine property and any adjacent workings, as well as any abnormal conditions;
(b) the location and detailed specifications of all surface fans and all surface openings;
(c) the direction, velocity, and volume of air at each mine opening;
(d) all underground workings, including location of all stoppings, overcasts, undercasts, regulators, doors, and seals;
(e) the method of sealing worked-out areas, provisions for air sampling behind any such seals, and the manner in which such sealed areas will be vented into return air passages (ensuring that no intake air is or could be passing any sealed-off area);
(f) the location of all splits and the volume of fresh air entering each split and of return air at each cross-cut in a room-and-pillar mine and at each working face; and
(g) the locations for the measurement of air in the mine to ensure the proper ventilation at all times.

13 The mine operator should employ or retain the services of a qualified ventilation engineer to assist in the preparation of all ventilation plans or amendments to such plans. The ventilation engineer should sign any ventilation plans or amendments before they are submitted to the regulator.

14 The regulator may submit plans or amendments to a qualified mine ventilation engineer for review, and any fee for such review should be the responsibility of the mine operator. The regulator may require modifications to the plan in the interests of safety.

15 The regulator, in consultation with a qualified ventilation engineer, should draft regulations dealing with main fans and auxiliary fans. These regulations should include:

(a) details of the design, installation, operation, maintenance, and inspections of such fans; and
(b) requirements for instrumentation, the recording of data from such instrumentation, and the filing of this data with the regulator.

16 No booster fan should be installed underground without the approval of the regulator.

17 Every main ventilating fan should be mounted above ground in a fireproof fan house located at a safe distance from any mine opening and offset from any such openings or connections. The fan house should be equipped with a weak wall or explosion door located in a direct line with any possible explosion forces. Every main fan should be equipped with an audible alarm that sounds automatically if the fan stops or slows down.

18 Where any fan used in ventilating a mine stops for any reason, the area affected should be immediately evacuated. No auxiliary fan should be restarted until a qualified person has inspected the area and found it to be safe and free of gas. The area should not be re-entered until the ventilation has been restored to the required level and the area has been found to be safe and free of gas by a qualified person. If any fan remains stopped for more than 30 minutes, the mine operator should report the relevant circumstances to the regulator.

19 The regulator, in consultation with a qualified ventilation engineer, should draft regulations dealing with requirements for ducting, brattice, stoppings, locations of measuring devices, and sealing of abandoned sections of the mine. All brattice cloth, ducting, and materials used for constructing stoppings should be of fire-resistant material.

20 Equipment used to ventilate an underground coal mine should be of a type approved by the regulator and should be installed in an approved manner. Equipment, materials, or procedures not previously approved may be approved if the regulator is satisfied that the same measure of protection is provided to the underground worker.

21 Unless specifically approved in writing by the regulator, no more than one mechanized coal mining unit should operate in each ventilation split. Each split should be provided with a separate supply of fresh air.

22 Ventilating air should not be recirculated without the written consent of the regulator.

23 The mine operator should employ a qualified mine ventilation technician to be responsible for the operation and maintenance of the ventilation system. The ventilation technician should measure the airflow and sample the air quality in the mine at approved intervals of at least once a month for the whole mine and weekly for working areas. The results of ventilation and air quality tests should be recorded and a copy of such record should be filed with the regulator.

24 Workers should be removed from any area in a mine where the concentration of dust or noxious gases in the air exceeds the standards set out by the American Conference of Governmental Industrial Hygienists (ACGIH).

25 Devices used for testing air quality, velocity, and volume should be of a type certified and approved for such use by the Canada Centre for Mineral and Energy Technology (CANMET), the Approval and Certification Center of the Mine Safety and Health Administration (MSHA), the Canadian Standards Association (CSA), or other such equivalent testing body.