Chapter 9. Ventilation Planning

9.1. SYSTEMS ANALYSIS OF THE PLANNING PROCEDURE .......................... 1

9.2. ESTABLISHMENT OF THE BASIC NETWORK .................................. 3
  9.2.1. New mines or other subsurface facilities ........................................... 3
  9.2.2. Existing mines .................................................................................. 3
  9.2.3. Correlation study ............................................................................. 5

9.3. AIRFLOW REQUIREMENTS ................................................................. 6
  9.3.1. Strata gas .......................................................................................... 7
  9.3.2. Diesel exhaust fumes ................................................................. 7
  9.3.3. Dust ............................................................................................... 8
  9.3.4. Heat ......................................................................................... 9
  9.3.5. Workshops and other ancillary areas ............................................ 12
  9.3.6. Air velocity limits ........................................................................ 13

9.4. PLANNING EXERCISES AND TIME PHASES ................................... 13
  9.4.1. Network planning exercises .......................................................... 13
  9.4.2. Time phases ............................................................................... 14
  9.4.3. Selection of main fans ............................................................... 15
  9.4.4. Optimization of airflow systems ............................................. 15
  9.4.5. Short term planning and updating the basic network ............. 15

9.5. VENTILATION ECONOMICS AND AIRWAY SIZING ....................... 16
  9.5.1. Interest payments ...................................................................... 16
  9.5.2. Time value of money, present value ........................................ 18
    9.5.2.1. Present value of a lump sum ............................................. 18
    9.5.2.2. Present value of regular payments ................................ 19
  9.5.3. Equivalent annual cost ............................................................... 21
  9.5.4. Ventilation operating costs ........................................................... 22
  9.5.5. Optimum size of airway ............................................................... 23
    9.5.5.1. Capital cost function ......................................................... 23
    9.5.5.2. Operating cost function .................................................. 24
    9.5.5.3. Case Study ...................................................................... 25
      Task 1: Establish the capital cost function: .................................... 25
      Task 2: Establish the operating cost function: ................................ 26
      Task 3: Establish the total cost function: ....................................... 26
      Task 4: Determine the optimum diameter: .................................... 26
  9.5.6. Incorporation of shaft design into network planning exercises .... 27

9.6. TRADITIONAL METHOD OF VENTILATION PLANNING .................. 29

Bibliography .............................................................................................. 30

9.1. SYSTEMS ANALYSIS OF THE PLANNING PROCEDURE

In Section 1.3.3, we emphasized the importance of integrating ventilation planning with production objectives and overall mine design during the early stages of planning a new mine or other subsurface facility. The team approach should continue for ongoing forward planning throughout the active life of the mine. Compromises or alternatives should be sought which satisfy conflicting demands. For example, the ventilation planner may require a major airway that is too large from the viewpoint of the rock mechanics engineer. An alternative might be to drive two smaller airways but large enough for any equipment that is to move through them. Regular
liaison between engineering departments and exchanges during inter-disciplinary planning meetings can avoid the need for expensive re-designs and should promote an optimized layout.

Figure 9.1 illustrates an organized system of ventilation planning for an underground mine. The procedure assumes the availability of computer assistance including ventilation simulation software, and eliminates most of the manual techniques and intuitive estimates of older planning methodologies (Section 9.6). The initial step is to establish a data base in a Basic Network File. For an existing facility this requires information gained from ventilation surveys. The latter are described in detail in Chapter 6. For a completely new mine, initial layouts should be discussed at the cross-disciplinary planning meetings. It is usual, in this case, that several alternative layouts are required to be investigated by the ventilation design team. The data base for a new mine is established from initial estimates of required airway geometry, mining method, roughness of airway surfaces and network layout. Such data will be revised and refined as the design progresses.

In this chapter, we shall discuss the major facets of the planning procedure illustrated on Figure 9.1.
9.2. ESTABLISHMENT OF THE BASIC NETWORK

9.2.1. New mines or other subsurface facilities

It will be recalled from Chapter 7 that a ventilation network consists of a line schematic, each branch representing one or more airways or leakage paths. In general, branches must be of known resistance or designated to pass a specified airflow. For a planned but yet unconstructed facility there are, of course, no airways or ventilation infrastructure that can be surveyed in order to establish actual values of resistance. Recourse must then be made to the methods of estimating airway resistance described in Section 5.4, incorporating airway geometry, types of lining (friction factors) and shock losses. Experience and data from other mines that utilize comparable layouts and methods of airway drivage, and operating in similar geological conditions can be most helpful.

A variety of alternative plans will often require investigation during the initial design of a new mine. Ventilation network analyses should be carried out in order to determine the efficiency and effectiveness of each layout. The results of those analyses may then be considered, together with the other aspects of mine design, at the general planning meetings.

9.2.2. Existing mines

The routes of surveys in the main ventilation infrastructure of an existing mine provide a skeleton network, often referred to as a ventilation schematic. Additional branches can then be added or incorporated as equivalent resistances to represent airways or leakage paths that were not included in the primary surveys. As illustrated in the computed network example of Section 7.4.6, branch resistances may be input to a network simulation program in several ways. This allows considerable flexibility in assembling a representative basic network, commencing from surveyed routes. Essentially, four methods are used to indicate resistance values in the first compilation of a basic network file or to update a basic network that has previously been established:

1. For surveyed airways, the measured values of frictional pressure drop, \( p \) (Pa), and corresponding airflow, \( Q \) (m\(^3\)/s), may be input to the basic network file. The computer can then calculate the resistance from the square law.

\[
R = \frac{p}{Q^2} = \frac{Ns^2}{m^8}
\]

2. There are situations in which it is difficult or impossible to determine resistance values for existing airways. If, for example, the airflow in an unrestricted branch is low at the time of measurement then so, also, will be the frictional pressure drop. Low frictional pressure drops will also occur in airways of large cross-sectional area even at substantial airflows. While any pressure difference shown by the gauge should be logged, those lower than 5 Pa may yield resistances of doubtful accuracy. Modern diaphragm gauges will indicate to within ±0.5 Pa. At 5 Pa, the corresponding error is 0.1 or 10 per cent. Airflow measurements taken by approved methods (Chapter 6) will normally be accurate to within 5%. Compounding these possible errors gives a maximum corresponding error in airway resistance of

\[
\frac{dR}{R} = \pm \frac{dp}{p} \pm 2 \frac{dQ}{Q}
\]

(differentiation of the square law \( R = p/Q^2 \))

\[
\text{giving } \quad \frac{dR}{R} = \pm 0.1 \pm (2 \times 0.05)
\]

\[
= \pm 0.2 \text{ or } 20 \text{ per cent for the accuracy of the resistance of an airway that has a frictional pressure drop of 5 Pa.}
\]
If, because of low values, the airflow and/or pressure drop cannot be measured to the required accuracies, the airway resistance should be calculated from the Atkinson equation

\[ R = k L \frac{\text{per}}{A^3} \frac{\rho}{1.2} \text{Ns}^2/\text{m}^8 \]  

(see equation (5.9))

where  
\[ k = \text{Atkinson friction factor at standard density (kg/m}^3) \]  
\[ L = \text{length of airway (m)} \]  
\[ \text{per} = \text{perimeter (m)} \]  
\[ A = \text{cross sectional area (m}^2) \]  
and  
\[ \rho = \text{air density (kg/m}^3) \]

Representative values of friction factor, \( k \), should, preferentially, be determined from similar airways that have been surveyed in that same mine. The density correction, \( \rho/1.2 \) is then often ignored. Otherwise, values of \( k \) may be estimated using one of the methods discussed in Section 5.3. In all cases, care should be taken to allow for shock losses and unrepresentative obstructions (Appendix A5, Chapter 5).

In many situations within a given mine, groups of airways are driven to the same dimensions and with a similar friction factor. Typical values of resistance per metre length can be established from measurements taken in, say, intakes, returns and conveyor routes. This facilitates the addition of new or unsurveyed branches to the network. However, here again care should be taken not to overlook obstructions or any other cause of shock losses.

3. It is often difficult to measure the resistances of doors, stoppings or seals directly, either because of low airflows or inaccessibility (Kissell, 1978). In practice, the resistances of doors or stoppings in mines vary from several thousand to, literally, 1 or 2 Ns²/m⁸. The reason for this wide range is the highly non-linear relationship between area available for flow and resistance

\[ R \propto \frac{1}{d^5} \]  

(see equation (5.15))

where \( d = \text{hydraulic mean diameter of opening (4A/per)} \).

Hence, the appearance of fractures or apertures in, or around, a previously tight stopping will result in dramatic reductions in its resistance.

The following ranges are suggested for planning purposes and for cross-sectional areas up to 25 m²:

- **Single doors**: 10 to 50 Ns²/m⁸ (typical value, 25 Ns²/m⁸)
- **Stoppings**: 50 to 2000 Ns²/m⁸ (typical value 500 Ns²/m⁸)
- **Seals**: 1000 to 5000 Ns²/m⁸

4. The resistances of leakage paths including worked out areas and caved zones are the most difficult to assess for a network schematic. Leakage airflows can often be measured or estimated from survey results. These may be entered as "fixed quantity" leakage airflows, leaving the computer to evaluate the corresponding resistances. This 'inversion' method remains valid while the number of fixed quantity branches is sufficiently small for unique values of resistance to be calculated. A network that is over-restricted by too many fixed quantity branches will produce a warning message from the VNET simulation package (Section 7.4.6). Care should be taken to remove "fixed quantity" airways and to replace them with branches set at the corresponding computed resistances prior to progressing with planning exercises.
9.2.3. Correlation study

Figure 9.1 shows the data pertaining to the initial network schematic stored in the Basic Network File within computer memory. In order to ensure that the basic network is a true representation of the mine as it stood at the time of the survey, and before that basic network is used to launch planning exercises, it is important that it is subjected to verification through a correlation study. This involves running a ventilation network simulation package (Section 7.4) on the data contained in the basic network file and comparing the computed airflows with those that were measured during the surveys. If all airways had been surveyed instantaneously with perfect observers and perfect instruments, and if no errors had been made in compiling the basic network, then the computed and measured airflows would show perfect agreement. This utopian situation is never attained in practice. Despite the internal consistency checks on airflow and pressure drop measurements made during a survey (Section 6.4.3) there will, inevitably, remain some residual errors from observational and instrumental sources.

A larger potential source of difference between computed and measured airflows is the fact that a VNET airflow distribution is mathematically balanced to a close tolerance throughout the network and represents a steady-state 'snapshot' of the ventilation system. On the other hand, the surveys may have taken several weeks or even months to complete. In a large active mine, updating survey data should be a routine and continuous activity of the mine ventilation department. During the time taken for a survey the airflow distribution may have changed due to variations in natural ventilating pressures or the resistances offered by the ever-changing work areas.

Lastly, errors made in transcribing survey data into the basic network or in key-punching the information into the basic network file will, again, result in disagreement between computed and measured airflows.

The correlation itself is normally carried out by a subsidiary program that lists the computed and observed airflows together with their actual and percentage differences. Leakage airflows of less than 3 m³/s may be ignored in this comparison as large percentage errors in such low airflows will usually have little influence on the overall accuracy of the network.

The overall network correlation is quantified as

\[
\frac{\Sigma \text{Absolute value of (differences between computed and actual airflows for surveyed branches)}}{\Sigma \text{Absolute values of (measured airflows for surveyed branches)}} \times 100
\]

(9.3)

A correlation is accepted as being satisfactory provided that:

(i) no significant airflow branch shows a difference of more than 10 per cent between computed and measured airflows; and

(ii) the overall correlation is also within 10 per cent.¹

However, because of the highly variable conditions that exist along mine traverses, the survey team will often be able to weight certain measurements as being more, or less, reliable than others. Such pragmatic considerations may be taken into account during the correlation study.

If the initial results of a correlation study indicate unacceptable deviations between computed and measured airflows then the basic network must be improved before it can be used as a basis for future planning exercises. The transcription of survey data into the basic network file should be checked. In the case of computed airflows being consistently higher or lower than the corresponding measurements then it is possible that an error has been made in the pressure

¹ Experienced teams can often achieve overall correlation within 5 percent.
ascribed to a main fan or may indicate that the resistances of primary airways such as shafts should be verified. Additional or check measurements of surveyed loops may be required.

It is fairly common that disagreements will occur in localized areas of the network. Again, this may indicate that the extent of the surveys had been insufficient to give an adequate representation of the mine network and that further measurements are necessary. More often, however, the problem can be resolved by adjustment of the estimated resistances or fixed airflows allocated to unsurveyed flowpaths. Provided that the main infrastructure of the surveyed network is sound then amendments to those estimated values will direct the simulated network towards an improved correlation. However, if measurements made in the surveyed loops are significantly out of balance then no amount of adjustment to subsidiary airways will produce a well-correlated basic network.

9.3. AIRFLOW REQUIREMENTS

The estimation of airflows required within the work areas of a mine ventilation network is the most empirical aspect of modern ventilation planning. The majority of such assessments remains based on local experience of gas emissions, dust, or heat load and is still often quoted in the somewhat irrational terms of m³/s per ton of mineral output, particularly for non-coal mines. Corrections can be applied for variations in the age of the mine, the extent of old workings, distances from shaft bottoms, depth and rates of production. However, as in all empirical techniques, the method remains valid only whilst the proposed mining methods, machinery, and geological conditions remain similar to those from which the empirical data were evolved. Attempts to extrapolate beyond those circumstances may lead to serious errors in determining required airflows. Fortunately, simulation techniques are available to assist in assessing airflow requirements for both gassy and hot mines.

The characteristics of strata gas emissions, heat flow and dust production are discussed in Chapters 12, 15 and 20, respectively. In this section, we shall confine ourselves to an examination of the methods used to determine the airflows required to deal with given emission rates of airborne pollutants.

The overall necessity is that in all places where personnel are required to work or travel, airflows must be provided in such quantities that will safeguard safety and health, comply with statutory requirements, and that will also furnish reasonable comfort (Section 1.3.1).

The quantity of air required for the purposes of respiration of personnel is governed primarily by the concentration of exhaled carbon dioxide that can be allowed in the mine atmosphere rather than the consumption of oxygen. For vigorous manual work, this demands about 0.01 m³/s of air for each person and is negligible compared with the quantities of air needed to dilute the other gases, dust, heat and humidity that are emitted into the subsurface atmosphere.

It is a legal requirement that air volume flows must meet the governing state or national mining laws. Such legislation will normally define minimum airflows that should be provided at specified times and places in addition to threshold limit values for airborne pollutants or the psychrometric condition of the air. The ventilation planner must be familiar with the relevant legislation. In countries where mining law is absent or does not cover the particular circumstances of the project then it is prudent to follow well established regulations of a major mining country.
9.3.1. Strata gas

The various methods used to predict the emission rates of methane are described in Chapter 12. Whichever technique is employed, the final calculation of airflow requirement is

\[
Q = \frac{100E_g}{C_g} \text{ m}^3/\text{s}
\]

(9.4)

where 
- \( Q \) = required airflow (m\(^3\)/s)
- \( E_g \) = gas emission rate (m\(^3\)/s)
- \( C_g \) = general body concentration to which gas is to be diluted (percentage by volume).

The value of \( C_g \) is often taken to be one half of the concentration at which the law requires action to be taken.

Example

It has been predicted that during a 7 hour working shift, 2500 m\(^3\) of methane will be emitted into a working face in a coal mine. If electrical power must be switched off at a methane concentration of 1 per cent, determine a recommended airflow for the face.

Solution

The average rate of gas emission during the working shift is

\[
E_g = \frac{2500}{7 \times 60 \times 60} = 0.0992 \text{ m}^3/\text{s}
\]

Let us take the allowable concentration for design purposes to be one half the legal limit. Then \( C_g = 0.5 \) per cent.

Equation (9.4) gives the required airflow as

\[
Q = \frac{100 \times 0.0992}{0.5} = 19.84 \text{ say 20 m}^3/\text{s}.
\]

9.3.2. Diesel exhaust fumes

There are wide variations in the manner in which different countries calculate the ventilation requirements of mines in which diesels are used. The basic stipulation is that there should be sufficient ventilation to dilute exhaust gases and particulates to below each of their respective threshold limit values. One technique, based on engine tests, is to calculate the airflow required to dilute the mass emission of each pollutant to one half the corresponding TLV (threshold limit value). The maximum of those calculated airflows is then deemed to be the required air quantity. Some countries require analyses of the raw exhaust gases in addition to general body air sampling downstream from diesel equipment. Distinctions may be drawn between short-term exposure and time-weighted averages over an 8 hour shift (Section 11.2.1.).

In addition to exhaust gases, national enforcement agencies may require that other factors be taken into account in the determination of airflow requirements for diesel equipment. These include rated engine power (kW or bhp), number of personnel in the mine or area of the mine, rate of mineral production, engine tests, number of vehicles in the ventilation split and forced dilution of the exhaust gases before emission into the general airstream.
There is, as yet, no method of determining airflow requirements for diesels that will guarantee compliance with legislation in all countries. Here again, regulations pertinent to the location of the project must be perused for specific installations.

The generic criterion that is used most widely for initial estimates of required airflow is based on rated output power of the diesel equipment. For design purposes, many ventilation planners employ 6 to 8 m³/s of airflow over the machine for each 100 kW of rated diesel power, all equipment being cumulative in any one air split. However, it should be borne in mind that the actual magnitude and toxicity of exhaust gases depend upon the type of engine, conditions of operation and quality of maintenance in addition to rated mechanical power. It is for this reason that mining law refers to gas concentrations rather than power of the diesel equipment. Some manufacturers and government agencies give recommended airflows for specific diesel engines. In hot mines, it may be the heat produced by diesel equipment that sets a limit on its use.

9.3.3. Dust

Pneumoconiosis has been one of the greatest problems of occupational health in the mining industries of the world. Concentrated and long-term research efforts have led to greatly improved understanding of the physiological effects of dusts, methods of sampling and analysis, mandatory standards and dust control measures (Chapters 19 and 20).

There are many techniques of reducing dust concentrations in mines, ranging from water infusion of the solid mineral through to dust suppression by water sprays and air filtration systems. Nevertheless, with current mining methods, it is inevitable that dust particles will be dispersed into the air at all places wherever rock fragmentation or comminution occurs - at the rock-winning workplace and throughout the mineral transportation route. Dilution of airborne particles by ventilation remains the primary means of controlling dust concentrations in underground mines.

Dispersed dust particles in the respirable range (less than 5 µm diameter) will settle out at a negligible rate. For the determination of dilution by airflow, respirable dust may be treated as a gas. However, in this case it is realistic to estimate dust makes at working faces in terms of grams (or milligrams) per tonne of mineral mined. This will, of course, depend upon the method of working and means of rock fragmentation.

The required airflow is given as

\[
Q = \frac{E_d}{C_d} \times \frac{P}{3600} \text{ m}^3/\text{s}
\]  

(9.5)

where

- \(E_d\) = the emission rate of respirable dust (mg/tonne)
- \(P\) = rate of mineral production (tonnes/h) and
- \(C_d\) = allowable increase in the concentration of respirable dust (mg/m³).

Example.
The intake air entering a working area of a mine carries a mass concentration of 0.5 mg/m³ respirable dust. Face operations produce 1000 tonnes of mineral over an 8 hour shift and add respirable dust particles to the airflow at a rate of 1300 mg per tonne of mineral mined. If the concentration of respirable dust in the return air is not to exceed 2 mg/m³, determine the required airflow.

Solution.

Average rate of mineral production  
\[
= \frac{1000}{8 \times 3600} = 0.0347 \text{ t/s}
\]

Average rate of emission of respirable dust  
\[
= 0.0347 \times 1300 \frac{\text{t mg}}{\text{s} \text{ t}} = 45.14 \text{ mg/s}
\]
Dust removal capacity of air, $C_d = (\text{allowable dust concentration} - \text{intake dust concentration})$

\[= \frac{2 - 0.5}{1.5} = 1.5 \text{ mg/m}^3.\]

Required airflow

\[
= \frac{45.14}{1.5} \frac{\text{mg m}^3}{\text{mg s}} = 30.1 \text{ m}^3/\text{s}
\]

For larger particles of dust, the treatment is rather different. In this case, it is primarily the velocity rather than the quantity of the airflow that is important. In any airway the distance over which dust particles are carried by the airstream depends upon the air velocity and the settling rate of the individual dust particles. The latter depends, in turn, upon the density, size and shape of the particle as well as the psychrometric condition of the air. The heavier and more spherical particles will settle more rapidly while the particles that are smaller or of greater aspect ratio will tend to remain airborne for a longer period of time.

An effect of non-uniform dispersal characteristics is that the mineralogical composition of the airborne and settled dust is likely to vary with distance downstream from the source. The sorting effect of airflow on dust particles explains the abnormally high quartz content of dust that may be found in the return airways of some coal mines (Section 20.3.8.).

The larger particles will not be diluted proportionally by increased airflow and, hence, higher velocities. As the airspeed increases, these particles will remain airborne for longer distances before settling out. Furthermore, as the air velocity continues to rise, previously settled particles may be entrained by turbulent eddies into the airstream. The dust concentration will increase.

The effects of these mechanisms are illustrated on Fig. 9.2. A minimum total dust concentration is obtained at a velocity of a little over 2 m/s. Fortunately, the total dust concentration passes through a fairly broad based curve and air velocities in the range 1 to 4 are acceptable. Above 4 m/s the problem is not so much a health hazard as it is the physical discomfort of large particles striking the skin. In addition to the question of ventilation economics, this limits the volume flow that can be passed through any workplace for cooling or the dilution of other pollutants.

**9.3.4. Heat**

Most of the pollutants that affect the quality of air in an underground environment enter the ventilating airstreams as gaseous or particulate matter and, hence, may be diluted by an adequate supply of air. The problem of heat is quite different, involving changes in the molecular behaviour of the air itself and its thermodynamic and psychrometric properties. Furthermore, it is normally the case that air entering mine workings from the base of downcast shafts, slopes or intake adits is relatively free of gaseous or particulate pollutants. On the other hand, the condition of the intake air with regard to temperature and humidity depends upon the surface climate, and the depth of the workings. Whilst the removal of heat by ventilation remains the dominant method of maintaining acceptable temperatures in the majority of the world’s mines, quantifying the effects of varying airflow is by no means as straightforward as for gases or dust.

The complication is that air velocity and, by implication, airflow is only one of several variables that affect the ability of a ventilating airstream to produce a physiologically acceptable mine climate. Calculations of the climatic effect of variations in airflow are necessarily more involved than for gases or particulates.

The heat energy content of air, defined in terms of kilojoules of heat associated with each kilogram of dry air, is known as Sigma Heat, $S$. This concept is discussed fully in Section 14.5.3. Sigma heat depends only upon the wet bulb temperature of the air for any given barometric pressure.
The stages of determining the airflow required to remove heat from a mine or section of a mine are as follows:

(a) Evaluate the sigma heat of the air at inlet, $S_1$, using equations (14.44) to (14.47).

(b) Evaluate the highest value of sigma heat, $S_2$, that can be accepted in the air leaving the mine or section of the mine. This threshold limit value may be specified in terms of one of the indices of heat stress (Chapter 17) or simply as a maximum acceptable (cut-off) value of wet bulb temperature.

(c) Estimate the total heat flux, $q_{12}$ (kW), into the air from all sources between inlet and outlet (Chapter 15). This may involve simulation studies for additions of strata heat.

Figure 9.2 Variation of dust concentration with respect to air velocity.
(d) The required airflow, \( Q \), is then given as

\[
Q = \frac{q_{12}}{\rho(S_2 - S_1)} \quad \text{m}^3/\text{s}
\]  

(9.6)

where \( \rho \) = mean density of the air (kg/m\(^3\))

The factor \( \rho q_1(S_2 - S_1) \) is sometimes known as the Heat Removal Capacity (HRC) of a given airflow, \( Q \).

It should be noted that as \( S_1 \) approaches \( S_2 \), or for high values of added heat, \( q_{12} \), the required airflow may become excessive. In this case, either \( q_{12} \) must be reduced or air cooling plant installed (Chapter 18).

Example.
Air enters a section of a mine at a wet bulb temperature, \( t_w \) of 20 °C and a density, \( \rho \), of 1.276 kg/m\(^3\). The mean barometric pressure is 110 kPa. It has been determined that 2 MW of heat are added to the air in the section. If the wet bulb temperature of the air leaving the section is not to exceed 28 °C and no air coolers are to be used, determine the required airflow.

Solution.
From equations (14.44 to 14.47, Chapter 14 on Psychrometry) sigma heat is given as

\[
S = \frac{0.622 e_{sw}}{(P - e_{sw})} \left(2502.5 - 2.386 t_w\right) + 1.005 t_w \quad \text{kJ/kg}
\]

(9.7)

where saturation vapour pressure at wet bulb temperature is calculated as

\[
e_{sw} = 0.6106 \exp\left\{ \frac{17.27 t_w}{237.3 + t_w} \right\} \quad \text{kPa}
\]

At intake \( t_w = 20 \)°C, giving

\[
e_{sw1} = 0.6106 \exp\left\{ \frac{17.27 \times 20}{237.3 + 20} \right\} = 2.3375 \quad \text{kPa}
\]

and

\[
S_1 = \frac{0.622 \times 2.3375}{(110 - 2.3375)} \left(2502.5 - 2.386 \times 20\right) + (1.005 \times 20) = 53.25 \quad \text{kJ/kg}
\]

At return Using the same equations with \( t_w \) set at the cut-off value of 28°C gives

\[
S_2 = 82.03 \quad \text{kJ/kg.}
\]

The required airflow at the intake density is given by equation (9.6)

\[
Q_1 = \frac{q_{12}}{\rho(S_2 - S_1)} = \frac{2000}{1.276(82.03 - 53.25)} \quad \text{kJ, m}^3/\text{kg, kJ}
\]

\[
= 54.5 \quad \text{m}^3/\text{s}
\]
In this section, we have assumed that the heat added to the air can be determined. Although this may be straightforward for machine heat and other controlled sources (Section 15.3), it is more difficult to quantify heat emitted from the strata. For deep mines, the design of combined ventilation and air conditioning systems has been facilitated by the development of climatic simulation programs. These may be incorporated into the planning procedure as shown at the bottom of Figure 9.1. A more detailed discussion of the interaction between ventilation network and climatic simulation programs is given in Section 16.3.5.

9.3.5. Workshops and other ancillary areas

In addition to areas of rock fragmentation, there are many additional locations in mines or other subsurface facilities that require the environment to be controlled. These include workshops, stationary equipment such as pumps or electrical gear, battery charging and fuel stations, or storage areas. For long term storage of some materials in repository rooms, it may not be necessary or even desirable to maintain a respirable environment. However, temperatures and humidities will often require control.

In mines, it is usual for workshops and similar areas to be supplied with fresh intake air and to regulate the exit flow into a return airway either directly or through a duct.

The airflow requirement should, initially, be determined on the basis of pollution from gases, dust and heat as described in Sections 9.3.1 to 9.3.4. However, in the case of large excavations, this may give rise to excessively low velocities with zones of internal and uncontrolled recirculation. In this situation, it is preferable to employ the technique of specifying a number of air changes per hour.

Example
An underground workshop is 40m long, 15m wide and 15m high. Estimate the airflow required (a) to dilute exhaust fumes from diesel engines of 200 kW mechanical output and (b) on the basis of 10 air changes per hour.

Solution
(a) Employing an estimated airflow requirement of 8 m$^3$/s per 100 kW of rated diesel power (Section 9.3.2) gives

$$\frac{200 \times 8}{100} = 16 \text{ m}^3/\text{s}$$

(b) Volume of room $= 40 \times 15 \times 15 = 9000 \text{ m}^3$

At 10 air changes per hour,

Airflow $= \frac{9000 \times 10}{60 \times 60} = 25 \text{ m}^3/\text{s}$

Hence, the larger value of 25 m$^3$/s should be employed.
9.3.6. Air velocity limits

The primary consideration in the dilution of most pollutants is the volume flow of air. However, as indicated on Figure 9.1, the velocity of the air (flowrate/cross-sectional area) should also be determined. Excessive velocities not only exacerbate problems of dust but may also cause additional discomfort to personnel and result in unacceptable ventilation operating costs.

Legislation may mandate both maximum and minimum limits on air velocities in prescribed airways. A common lower threshold limit value for airways where personnel work or travel is 0.3 m/s. At this velocity, movement of the air is barely perceptible. A more typical value for mineral-winning faces is 1 to 3 m/s. Discomfort will be experienced by face personnel at velocities in excess of 4 m/s (Section 9.3.3) because of impact by large dust particles and, particularly, in cool conditions. Table 9.1 gives a guide to upper threshold limit values recommended for air velocity.

<table>
<thead>
<tr>
<th>Area</th>
<th>Velocity (m/s)</th>
</tr>
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<tbody>
<tr>
<td>Working faces</td>
<td>4</td>
</tr>
<tr>
<td>Conveyor drifts</td>
<td>5</td>
</tr>
<tr>
<td>Main haulage routes</td>
<td>6</td>
</tr>
<tr>
<td>Smooth lined main airways</td>
<td>8</td>
</tr>
<tr>
<td>Hoisting shafts</td>
<td>10</td>
</tr>
<tr>
<td>Ventilation shafts</td>
<td>20</td>
</tr>
</tbody>
</table>

Table 9.1 Recommended maximum air velocities

In wet upcast shafts where condensation or water emissions result in airborne droplets, the air velocity should not lie between 7 to 12 m/s. Water blanketing may occur in this range of velocities. The resulting variations in shaft resistance cause an oscillating load on main fans and can produce large intermittent cascades of water falling to the shaft bottom.

9.4. PLANNING EXERCISES AND TIME PHASES

The dynamic nature of a mine or any other evolving network of subsurface airways requires that the infrastructure of the ventilation system be designed such that it can accommodate major changes during the life of the undertaking. For a completely new mine, the early ventilation arrangements will provide for the sinking of shafts, drifts or adits, together with the initial development of the primary underground access routes. When the procedures illustrated on Figure 9.1 are first applied to an existing mine, the establishment and correlation of a basic network file will often reveal weaknesses and inefficiencies in the prevailing ventilation system. The initial planning exercises should then be directed towards correcting those deficiencies while, at the same time, considering the future development of the mine.

Following the establishment of a new or revised ventilation design, further planning investigations should be carried out for selected stages of future development. Although this is commonly termed "time phasing", it might more accurately be regarded as representing phases of physical development rather than definitive periods of time. Schedules have a habit of slipping or are subject to considerable revision during the course of mining.

9.4.1. Network planning exercises

Referring once again to Figure 9.1, the first step in a ventilation network planning exercise is to establish the airflows required in all places of work, travel or plant location (Section 9.3). Air velocity limits should also be set for the main ventilation flowpaths (Section 9.3.6). Additional
constraints on the direction and magnitudes of pressure differentials across doors, bulkheads or stoppings, or between adjoining areas of the facility, may also be imposed. This is of particular importance in nuclear waste repositories.

The ventilation engineers should review the existing network (correlated basic network or previous time phase) and agree upon a series of alternative designs that will satisfy revised airflow objectives and meet constraints on velocity and pressure differentials. The network schematic should then be modified to represent each of those alternative designs in turn. The network modifications may include

- sealing worked-out areas and opening up new districts
- adding main airways and lengthening existing airways
- adding and/or removing air crossings, doors and stoppings
- adding, removing or amending fixed quantity branches
- adding or sealing shafts and other surface connections
- adding, removing or relocating main or booster fans, or changing to different fan characteristic curves in order to represent adjustments to fan speed or vane settings
- amending natural ventilation pressures.

It is worth reminding ourselves, at this stage, that current network simulation programs do not perform any creative design work. They simply predict the airflow and pressure distributions for layouts that are specified by the ventilation planner. While computed output will usually suggest further or alternative amendments, it is left to the engineer to interpret the results, ensure that design objectives and constraints are met, compare the efficiency and cost effectiveness of each alternative design and to weigh the practicality and legality of each proposed scheme.

Each alternative layout chosen by the design engineers should be subjected to a series of VNET simulations and network amendments until either all the prescribed criteria are met or it becomes clear that the system envisaged is impractical. Tables, histograms and spreadsheets should be prepared in order to compare fan duties (pressures and air quantities), operating costs, capital costs and the practical advantages/disadvantages of the alternative layouts.

9.4.2. Time phases

During the operating life of an underground mine, repository or other subsurface facility, there will occur periods when substantial changes to the ventilation system must be undertaken. This will occur, for example, when:

- a major new area of the mine is to be opened up or an old one sealed
- workings become sufficiently remote from surface connections that the existing fans or network infrastructure are incapable of providing the required face airflows at an acceptable operating cost
- two main areas of the mine (or two adjoining mines) are to be interconnected.

“Time phase” studies should be carried out on alternative schematics to represent stages before and after each of these changes. Additional scenarios may be chosen at time intervals between major changes or to represent transitional stages. The latter often impose particularly heavy duties on the ventilation system.

Time phase exercises should be conducted to cover the life of the mine or as far into the future as can reasonably be predicted, assuming a continued market for the mined product.

It is, perhaps, intuitive to conduct time phase exercises in an order that emulates the actual planned chronological sequence. This may not always be the most sensible order for the network
exercises. For example, if it appears inevitable that a new shaft will be required at some time in
the future, it may be prudent to first investigate the time phase when that shaft has become
necessary and then to examine earlier time phases. This assists the planners in deciding whether
it would be advantageous to sink the new shaft at a prior time rather than to wait until it becomes
absolutely necessary.

The time phase exercises will produce a series of quantified ventilation networks for each time
period investigated. These should be reviewed from the viewpoint of continuity between time
phases and to ensure that major additions to the ventilation infrastructure have an acceptable and
cost-effective life. Indeed, the need for such continuity should remain in the minds of the planners
throughout the complete investigation, although not to the extent of stifling viable alternative
designs at later stages of the time phase exercises.

The result of such a review should be a series of selected layouts that represent the continuous
development of the ventilation system throughout the projected life of the mine. Again, tables and
histograms should be drawn up to show the variations of fan duties and costs, this time with
respect to chronological order.

9.4.3. Selection of main fans

Care should be taken that any main fan purchased should be capable of producing the range of
pressure-volume duties necessary throughout its projected life. The variation in required fan
duties resulting from time phase exercises should be considered, particularly where a fan is to be
relocated at some time during its life.

For a new purchase, fan manufacturer's catalogues should be perused and, if necessary,
discussions held with those manufacturers, in order to ensure that the fan selected is capable of
providing the required range of duties. Fortunately, the availability of variable pitch axial fans and
inlet vane control on centrifugal fans allows a single fan to provide a wide range of duties with a
fixed speed motor. A more detailed discussion of fan specifications is given in Section 10.6.2.

9.4.4. Optimization of airflow systems

The network exercises carried out for any one time phase should incorporate a degree of
optimization concerning the layout of airways, and the locations and duties of new shafts and
fans. For a single major flowpath such as a proposed new shaft, a detailed optimization study can
be carried out (Section 9.5.5). However, the larger questions that can arise involve a balance
between capital and operating costs and might include a choice between additional airways and
fans of greater power, or the minimization of total power costs incurred by alternative
combinations of main and booster fans. Special purpose optimization programs have been written
to assist in the resolution of such questions. (Calizaya et al, 1988)

9.4.5. Short term planning and updating the basic network

This chapter is concerned primarily with the major planning of a subsurface ventilation system.
The time phases are chosen to encompass periods of significant change in the ventilation layout.
However, much of the daily work of mine ventilation engineers involves planning on a much
shorter time scale.

As working districts advance or retreat, the lengths of the airways that serve them change. Also,
strata stresses can result in reductions of the cross-sectional areas of those airways. Additional
roof support may be required resulting in increased friction factors. Stoppings, regulators, doors
and air-crossings may deteriorate and allow greater leakage. All of these matters cause changes
in the resistances of individual airways and, hence, of the complete network. Additionally, wear of
fan impellers can result in variations in the corresponding pressure-volume characteristic curves. For these reasons, a basic network file that correlated well with survey data at the start of any given time phase may become less representative of the changing real system.

In addition to regular maintenance of all ventilation plant and controls, further surveys should be carried out on a routine basis and, in particular, whenever measured airflows begin to deviate significantly from those predicted for that time phase. Visual inspections and frequent liaison with shift supervisors are also invaluable for the early identification of causes of deviation from planned airflows. Relationships with production personnel will remain more congenial if the latter are kept well advised of the practical consequences of blocking main airflow routes with supplies, equipment or waste material.

As new survey data become available, the basic network file should be updated accordingly. If the basic network file is maintained as a good representation of the current mine then it will prove invaluable in the case of any emergency that involves the ventilation system.

In the event that such updates indicate permanent and significant impact on the longer term plans then the network investigations for the future time phases should be re-run and the plans amended accordingly. In the majority of cases this may become necessary because of changes in mine production plans rather than from any unpredicted difficulties with ventilation.

9.5. VENTILATION ECONOMICS AND AIRWAY SIZING

The subsurface ventilation engineer must be capable of dealing with two types of costs:

(a) Capital costs that require substantial funds at a moment in time or distributed over a short time period. A main fan installation or sinking a new shaft comes into this category.

(b) Working or operating costs, these representing the expenditure of funds on an ongoing basis in order to keep a system operating. Although consumable items, maintenance and even small items of equipment may be regarded as working costs, ventilation system designers often confine the term ‘operating costs’ to the price of providing electrical power to the fans.

The problem that typically arises concerns the combination of capital and operating costs that will minimize the real total cost to the company or mining organization; for example, whether to purchase an expensive but efficient fan to give low operating costs or, alternatively, a less expensive fan that will necessitate higher operating costs. Another question may be whether the money saved initially by sinking a small diameter ventilation shaft is worth the higher ongoing costs of passing a required airflow through that shaft. These are examples of the types of questions that we shall address in this section.

In the following examples and illustrations, we shall employ the dollar ($) sign to indicate a unit of money. However, in all cases, this may be replaced by any other national unit of currency.

9.5.1. Interest payments

Money can be regarded as a commodity that may be circulated to purchase goods or services. If we borrow money to buy a certain item then we are, in effect, renting the use of that money. We must expect, therefore, that in addition to returning the borrowed sum, we must also pay a rental fee. The latter is termed the interest payment. Hence, if we borrow $100 for a year at an annual interest rate of 9 per cent, then we must repay $109 one year from now. Even if we owned the money and did not have to borrow, spending the $100 now would mean that we are prevented from earning interest on that sum by investing or lending it to an individual, company, bank or other financial institution. Using money always involves an interest penalty.
If we borrow an amount of money, $P$ (principal) at a fractional interest rate $i$ (e.g. at 9 per cent, $i = 0.09$) then we will owe the sum $S$ where

\[
\begin{align*}
\text{Now} & \quad S = P \\
\text{after 1 year} & \quad S = P (1+i) \\
\text{after 2 years} & \quad S = P (1+i) (1+i) = P(1+i)^2 \\
\text{after 3 years} & \quad S = P(1+i)^3 \\
\text{after n years} & \quad S = P(1+i)^n \\
\end{align*}
\]

Figure 9.3 gives a visual indication of the effects of compounding interest each year.

Example 1
$250\ 000$ is borrowed for 8 years at an annual interest rate of 10 per cent. Determine the total sum that must be repaid at the end of the period, assuming that no intervening payments have been made.

Solution
\[
S = 250\ 000 (1 + 0.1)^8 = \ 535\ 897
\]
Example 2
How long does it take for an investment to double in dollar value at an annual interest rate of 12 per cent?

Solution
In this example, the final sum is twice the principal, $S = 2P$. Hence, equation (9.8) becomes

$$2P = P (1 + 0.12)^n$$

giving

$$n = \frac{\ln(2)}{\ln(1.12)} = 6.12 \text{ years}$$

9.5.2. Time value of money, present value

It is clear from a consideration of interest that the value of a set sum of money will vary with time. At 9 per cent interest, $100 now is worth $109 in one year and $118.81 two years from now. The effect of national inflation will, of course, partially erode growth in the value of capital. To determine real purchasing power, rates of interest should be corrected for inflation. In this section we are assuming that this has been done and quoted interest rates have taken projected inflation into account.

9.5.2.1. Present value of a lump sum.

The time variation of the value of money makes it inequitable to compare two sums that are borrowed or spent at different times. We need a common basis on which both sums can be fairly evaluated. One method of doing this is to determine what principal or capital, $P$, we need to invest now in order that it will grow to a desired sum, $S$, in a specified number of years. This is given by a simple transposition of equation (9.8)

$$P = \frac{S}{(1+i)^n} \quad (9.9)$$

When determined in this way, the value of $P$ is known as the present value of the future sum $S$. If all future investments or expenditures are reduced to present values then they can correctly be compared.

Example
Three fans are to be installed; one immediately at a price of $260,000, one in 5 years at an estimated cost of $310,000 and the third 8 years from now at $480,000. Determine the total expenditure as a present value if the annual interest rate is 10 per cent.

Solution
From equation (9.9), $P = \frac{S}{(1+i)^n}$

<table>
<thead>
<tr>
<th>Fan</th>
<th>Time $n$</th>
<th>Purchase price $S$</th>
<th>Present value $P$</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>now</td>
<td>$260,000$</td>
<td>$260,000$</td>
</tr>
<tr>
<td>2</td>
<td>5 years</td>
<td>$310,000$</td>
<td>$192,486$</td>
</tr>
<tr>
<td>3</td>
<td>8 years</td>
<td>$480,000$</td>
<td>$223,924$</td>
</tr>
</tbody>
</table>

Total $676,410$

In this example, the first fan is actually the most expensive.
9.5.2.2. Present value of regular payments.

In the case of operating costs, payments must be made each year. Such future payments may also be expressed as present values in order to compare and compound them with other expenditures.

We assume operating costs, \( S_o \), to be constant and paid at the end of each year. Then at the end of the first year \((n = 1)\), equation (9.9) gives

\[
P_{o,1} = \frac{S_o}{(1+i)}
\]

where \( P_{o,1} \) = present value of the first year's annual operating cost \( S_o \) at an interest rate of \( i \).

Similarly, the present value of the same operating cost, \( S_o \), in the second year \((n = 2)\) is

\[
P_{o,2} = \frac{S_o}{(1+i)^2}
\]

It follows that the total present value, \( P_o \), of operating costs, \( S_o \) paid at each year-end for \( n \) years becomes

\[
P_o = S_o \left[ \frac{1}{(1+i)} + \frac{1}{(1+i)^2} + \frac{1}{(1+i)^3} + \ldots + \frac{1}{(1+i)^n} \right]
\]

(9.10)

The bracketed term is a geometric progression which can be summed to give

\[
P_o = \frac{S_o}{i} \left[ 1 - \frac{1}{(1+i)^n} \right]
\]

(9.11)

Figure 9.4 gives a graphical representation of this equation for \( S_o = $1 \)

![Figure 9.4 Present value of annual operating costs.](image)
Example 1
Electrical power costs at a mine are estimated to be $850 000 in each of the next 12 years. Determine the present value of this expenditure at 11 per cent interest.

Solution
With $S_0 = $850 000, $i = 0.11$ and $n = 12$ years, equation (9.11) gives

$$P_o = \frac{850 000}{0.11} \left[ 1 - \frac{1}{(1 + 0.11)^{12}} \right] = 5 \, 518 \, 503$$

Example 2
Tenders received from manufacturers indicate that a fan priced at $170 000 will cost $220 000 per year to run, while a fan with a purchase price of $265 000 will require $190 000 per year in operating costs. If the costing period is 5 years at an annual interest of 7 per cent, determine which fan is most economical.

Solution
Using equation (9.11) with $n = 5$ years, $i = 0.07$ and $S_0$ set at $220 000$ and $190 000$ respectively allows the present value of the operating costs to be determined.

<table>
<thead>
<tr>
<th>Fan</th>
<th>Current purchase price ($)</th>
<th>Annual operating cost($/year)</th>
<th>Purchase price ($)</th>
<th>Operating cost over 5 yr ($)</th>
<th>Total ($)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>170 000</td>
<td>220 000</td>
<td>170 000</td>
<td>902 043</td>
<td>1 072 043</td>
</tr>
<tr>
<td>2</td>
<td>265 000</td>
<td>190 000</td>
<td>265 000</td>
<td>779 038</td>
<td>1 044 038</td>
</tr>
</tbody>
</table>

In this example, fan 2 is shown to be the more economic choice.

During the analysis of time phase exercises (Section 9.4.2) it may be necessary to establish the present value of operating costs for periods that do not commence at the present time. By applying equation (9.11) to periods of $m$ years and $n$ years respectively, and taking the difference, the present value of operating costs for the future period $n$ to $m$ years becomes

$$P_{o, nm} = \frac{S_o}{i} \left[ \frac{1}{(1 + i)^n} - \frac{1}{(1 + i)^m} \right] \quad (9.12)$$
Example
This example shows an excerpt from a spreadsheet used to calculate the present values of capital and operating expenditures. Present values are calculated from equation (9.9) for capital (lump sum) expenditures and equation (9.12) for operating costs. The annual rate of interest is 10 per cent.

<table>
<thead>
<tr>
<th>Period</th>
<th>Capital expenditure($)</th>
<th>Operating cost ($/year)</th>
<th>Present value($)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. now</td>
<td>500 000</td>
<td></td>
<td>500 000</td>
</tr>
<tr>
<td>2. now to end of 4 years.</td>
<td></td>
<td>620 000</td>
<td>1 965 317</td>
</tr>
<tr>
<td>(n = 0, m = 4)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3. at end of 4 yrs</td>
<td>235 000</td>
<td></td>
<td>160 508</td>
</tr>
<tr>
<td>4. end of 4 yrs to end of 10 yrs (n = 4, m = 10)</td>
<td></td>
<td>680 000</td>
<td>2 022 797</td>
</tr>
</tbody>
</table>

Total present value = $4 648 622

Spreadsheets of this type can be set up on a personal computer to give a rapid means of costing the results of alternative network exercises.

9.5.3. Equivalent annual cost

When capital is borrowed, it is often more convenient to repay it, including interest, in equal installments each year, rather than as a lump sum paid at the end of the complete time period. This is a common method for domestic purchases such as a house (mortgage payments) or an automobile. In industry, this equivalent annual cost facilitates budgetary planning and enables major items of capital expenditure, such as a new shaft, to be spread evenly over a number of years. The method of equivalent annual cost allows capital expenditures to be compared with operating costs on a year by year basis and, for producing mines, also enables those capital expenditures to be expressed in terms of cost per tonne of mineral.

The equivalent annual cost is given by a re-interpretation of equation (9.11). \(P\) is the capital that must be invested now (present value) in order to make a regular payment, \(S\), each year. The statement may also be made in reverse, i.e. \(S\) is the regular payment, or annual equivalent cost, to be met each year in order to pay off the capital and interest on a borrowed amount, \(P\).

Transposing equation (9.11) gives the equivalent annual cost, EAC, as

\[
EAC = \frac{P i}{[1 - 1/(1+i)^n]} \quad (9.13)
\]
Example
A mine shaft is to be sunk at a cost of $5.6 million. The life of the shaft is estimated to be 15 years during which time the average planned rate of mineral production is 1.6 million tonnes per year. If the annual interest rate is 8.75 per cent, determine (a) the equivalent annual cost of the shaft in $ per year and (b) corresponding production cost in $ per tonne mined.

Solution
(a) Using equation (9.13) with $P = 5.6$ million, $i = 0.0875$ and $n = 15$ years, gives the equivalent annual cost as

$$ EAC = \frac{5,600,000 \times 0.0875}{[1 - 1/(1.0875)^{15}]} = $684,509 $$

(b) The corresponding production cost is

$$ \frac{684,509}{1,600,000} = $0.428 \text{ per tonne} $$

9.5.4. Ventilation operating costs

A fan unit, comprising an electric motor, transmission and impeller converts electrical energy into air power. The latter is reflected as kinetic energy of the air and a rise in total pressure across the fan. Air power delivered by a fan was quantified in Chapter 5 as

$$ p_f \times Q \times W \quad \text{(from equation (5.6))} $$

where $p_f$ = rise in total pressure across the fan (Pa) and $Q = \text{airflow (m}^3/\text{s})$

(See, also, Chapter 10 for the effects of compressibility).

However, the electrical power taken by the fan motor will be greater than this as losses occur inevitably in the motor, transmission and impeller. If the overall fractional efficiency of the unit is $\eta$, then the electrical input power to the motor will be

$$ \frac{p_f \times Q}{\eta} \times W $$

Electrical power charges are normally quoted in cost per kilowatt-hour. Hence, the cost of operating a fan for 24 hours per day over the 365 days in a year is

$$ S_o = \frac{p_f \times Q}{1000 \eta} \times e \times 24 \times 365 \quad $/year \quad (9.14) $$

where $e$ is the cost of power ($ per kwh$). In practice, the fan pressure, $p_f$ is often quoted in kPa, obviating the need for the 1000.

Equation (9.14) also applies for the annual cost of ventilating an individual airway. In this case, $p_f$ simply becomes the frictional pressure drop across the airway, $p$, at the corresponding airflow, $Q$, and $\eta$ the overall efficiency of the fan primarily responsible for ventilating that airway. In cases of multiple fans, a weighted mean average of fan efficiency may be employed.
Example
An underground airway is driven at a capital cost of $1.15 million dollars. During its life of 8 years, it is planned to pass an airflow, \( Q \), of 120 m\(^3\)/s at a frictional pressure drop, \( p \), of 720 Pa. The main fans operate at an overall efficiency, \( \eta \), of 72 per cent. If the annual rate of interest is 9.5 per cent and the average cost of electrical power is $0.06 per kWh, determine the annual total cost of owning and ventilating the airway.

Solution
The equivalent annual cost of owning the airway is given by equation (9.13) with \( P = 1.15 \) million, \( i = 0.095 \) and \( n = 8 \) years

\[
EAC = \frac{1150000 \times 0.095}{1 - 1/(1.095)^8} = 211652 \text{ per year}
\]

The annual operating cost is given by equation (9.14) with \( p_{ft} = 720 \) Pa, \( Q = 120 \) m\(^3\)/s, \( \eta = 0.72 \) and \( e = 0.06 \) per kWh

\[
S_o = \frac{720 \times 120}{1000 \times 0.72} \times 0.06 \times 24 \times 365 = 63072 \text{ per year}
\]

The total yearly cost is given by adding the cost of owning the airway and the annual operating cost of ventilation:

\[
\text{total annual cost } C = EAC + S_o = 211652 + 63072 = 274724 \text{ per year}
\]

9.5.5. Optimum size of airway

There are several factors that influence the optimum size of a subsurface airway, including
- the airflow to be passed through it
- the cost of excavation
- limitations on air velocity (Section 9.3.5)
- the span that can be supported adequately and
- the size of equipment required to travel through the airway.

This section considers the first two of these matters. However, the size of a planned new shaft or airway must satisfy the other constraints.

As the size of an airway is increased then its resistance and, hence, ventilation operating costs will decrease for any given airflow (Section 5.4.1). However, the capital cost of excavating the airway increases with size. The combination of capital and operating costs will be the total cost of owning and ventilating the airway. The most economic or optimum size of the airway occurs when that total cost is a minimum. The costs may be expressed either as present values or in terms of annual (equivalent) costs.

In order to quantify the optimum size, it is necessary to establish cost functions for both capital and operating costs that relate those expenditures to airway size.

9.5.5.1. Capital cost function
The business of arriving at a cost for excavating a mine shaft or major airway often involves protracted negotiations between a mining company and a contractor. Even when the task is to be undertaken in-house, the costing exercise may still be extensive.
There are, essentially, two components - fixed costs and variable costs. The fixed costs may include setting up and removal of equipment such as temporary headgear, hoisting facilities or a shaft drilling rig. Other items such as pipe ranges, electrical conduits, air ducts and track lines are usually independent of airway size but are a function of airway length.

The variable costs are those that can be expressed as functions of the length and cross-sectional area or diameter of the shaft or airway. These include the actual cost of excavation and supports.

The capital cost, $P_c$, may then be expressed as the cost function

\[ P_c = C_f + f(A, L) \]  

(9.15)

where \( C_f \) = fixed costs

and \( f(A, L) \) = function of cross-sectional area, \( A \) and length, \( L \).

In a simple case, the capital cost function may take the form

\[ P_c = C_f + aV + bL \]  

(9.16)

where \( V = \) volume excavated, \( A \times L \) (m\(^3\))

\( a \) and \( b \) are constants.

For mechanized excavations, there may be discontinuities in the capital cost function due to step increases in the size and sophistication of the equipment for differing sizes of airway.

9.5.5.2. Operating cost function

Equation (9.14) gave the annual ventilation cost of an airway to be

\[ S_o = \rho Q \frac{e}{1000 \eta} \times 24 \times 365 \]  

$/year \quad (9.17)$

However, from equation (2.50),

\[ \rho = R_t \rho \frac{Q^2}{} \]

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

and \( \rho \) = air density (kg/m\(^3\))

giving

\[ S_o = R_t \rho Q^3 \frac{e}{\eta} \times \frac{24 \times 365}{1000} \]

This demonstrates that the operating cost varies with airway resistance, \( R_t \), air density, \( \rho \), and the cube of the airflow, \( Q \).

Substituting for \( R_t \) from equation (2.51) gives

\[ S_o = \frac{f L \text{per}}{2A^3} \rho Q^3 \frac{e}{\eta} \times \frac{24 \times 365}{1000} \]  

$/year \quad (9.18)$

where \( f \) = coefficient of friction (dimensionless)

\( L \) = length of airway (m)

\( \text{per} \) = perimeter (m)

and \( A \) = cross-sectional area (m\(^2\))
In order to achieve the corresponding result with the Atkinson friction factor, \( k_{1.2} \), substituting for \( p \) from equation (5.8) allows the operating cost function of equation (9.17) to be written as

\[
S_o = k_{1.2} \frac{L \rho}{A^{1.2}} \frac{Q^3}{e} \times \frac{24 \times 365}{\eta} \times \frac{1000}{1} \text{ $/year} \tag{9.19}
\]

9.5.5.3. Case Study
During the design of a proposed circular shaft, the following data were generated:

**Shaft sinking:**
- Equipment set up and decommissioning costs = $650 000
- Excavation cost = $290/m³
- Fittings and lining = $460/m

**Physical data:**
- Depth of shaft, \( L \) = 700 m
- Effective coefficient of friction, \( f \) = 0.01
- \((\text{friction factor } k_{1.2} = 0.6f = 0.006 \text{ kg/m}^3)\)
- Mean air density, \( \rho \) = 1.12 kg/m³
- Airflow, \( Q \) = 285 m³/s
- Fan efficiency, \( \eta \) = 0.65
- Life of shaft, \( n \) = 15 years

**Additional financial data:**
- Annual rate of interest, \( i \) = 10 per cent
- Average cost of electrical power \( e \) = $0.075 per kWh

Determine the optimum diameter of the shaft.

**Solution**

**Task 1: Establish the capital cost function:**

From the data given, the capital cost function can be expressed in the form of equation (9.16)

\[
P_c = 650 000 + 290 V + 460 L
\]

where \( V = AL = \frac{\pi D^2}{4} \times 700 \text{ (m}^3)\)
and \( D = \text{shaft diameter (m)}\)

giving

\[
P_c = 650 000 + \left[ 290 \times 700 \times \frac{\pi D^2}{4} \right] + (460 \times 700)
\]

The capital cost function then simplifies to

\[
P_c = 972 000 + 159 436 D^2 \text{ $} \tag{9.20}
\]

This is the present value of the capital cost of sinking the shaft. The complete analysis may be carried out either in terms of present values or in annual (equivalent) costs. The latter method is employed for this case study. Hence, the capital cost of shaft sinking must be spread over the 15 year life using equation (9.13) with \( i = 0.1 \) and \( n = 15 \).
Equivalent annual cost of shaft sinking:

\[
EAC = \frac{(972000 + 159436D^2)}{[1 - 1/(1.1)^{15}]} \times 0.1 \quad \text{from equation (9.13)}
\]

\[
= 127793 + 20962D^2 \quad \text{$/year} \quad (9.21)
\]

**Task 2: Establish the operating cost function:**

This is accomplished by substituting the given data into equations (9.18) or (9.19)

Annual operating cost

\[
S_o = \frac{0.01 \times 700 \times \pi D}{2(\pi D^2/4)^3} \times 1.12 \times 285^3 \times 0.075 \times \frac{24 \times 365}{1000} \quad \text{$/year}
\]

\[
= 594.775 \times 10^6/D^5 \quad \text{$/year} \quad (9.22)
\]

**Task 3: Establish the total cost function:**

Addition of the capital and operating cost functions from equations (9.21) and (9.22) respectively gives the total annual cost function as

\[
C = 127793 + 20962D^2 + \frac{594.775 \times 10^6}{D^5} \quad \text{$/year} \quad (9.23)
\]

**Task 4: Determine the optimum diameter:**

This can be accomplished in either of two ways.

(a) By graphical means.

The three cost functions have been plotted on Figure 9.5. The minimum point on the total annual cost curve occurs at an optimum diameter of approximately 4.9m.

An advantage of constructing the cost curves is that it gives a visual indication of the behaviour of the cost functions. In particular, the operating and total costs rise rapidly when the airway diameter is reduced significantly below the optimum point. However, the total cost curve remains fairly shallow above the optimum point. This is a typical result and illustrates that the shaft size may be escalated to a standard diameter above the optimum to facilitate shaft sinking. Indeed, experience has shown the wisdom of sizing a shaft above the optimum. This allows subsequent flexibility for modifications to the airflow or mine production.

(b) By analytical means.

If the total cost function, \( C \), has been expressed as a convenient function of cross-sectional area, \( A \), or diameter, \( D \), then it can be differentiated to find the optimum location when the slope \( dC/dA \) or \( dC/dD \) becomes zero.

In this case study, differentiation of equation (9.23) gives

\[
\frac{dC}{dD} = 2 \times 20962D - \frac{5 \times 594.775 \times 10^6}{D^6}
\]

At \( dC/dD = 0 \)

\[
41924D^7 = 2973.875 \times 10^6 \quad \text{from which} \quad D = 4.93 \, \text{m.}
\]
This is an accurate value of the optimum diameter previously read from Figure 9.5 as the approximation 4.9m. In practice, the recommended diameter would be rounded up to at least 5m.

The corresponding total annual equivalent cost can be estimated from Figure 9.5 or calculated by substituting $D = 9.3$ into equation (9.23):

$$
C = 127793 + 20962 \times 4.93^2 + \frac{594.775 \times 10^6}{4.93^5}
$$

$$
= $841\,502 \text{ per year}
$$

9.5.6. Incorporation of shaft design into network planning exercises.

Figure 9.1 shows the optimization of major airways and fans to be a part of the ventilation planning process. However, it may be impractical to carry out such optimization for every investigatory cycle of a ventilation network exercise. The sizing of a shaft is normally conducted only after network analysis has established a satisfactory distribution of airflows underground and can be combined with the shaft design procedures described in Section 5.4.6.
The demands made upon a mine shaft may include access for personnel and equipment, the hoisting of planned tonnages of rock and the passing of specified airflows. We can now define a set of guidelines to assist in the management of a combined network analysis and shaft design exercise.

1. Assess the duties required for rock hoisting (tonnes per hour), number of personnel to be transported, time allowed at shift changes and the size, weight and frequency of hoisting materials and equipment.
2. Determine alternative combinations of conveyance sizes and hoisting speeds.
3. Conduct ventilation network investigations, initially on the basis of an estimated shaft resistance until a satisfactory distribution of airflow is achieved. This will establish the required airflow in the shaft.
4. Assess the dimensions of proposed shaft fittings including pipes, cables, guides and buntons.
5. Conduct an optimization exercise to find the size of shaft that will pass the required airflow at the minimum combination of operating costs and the capital expense of shaft construction (Section 9.5.5.).
6. Check the free area air velocity. If this exceeds 10 m/s in a hoisting shaft or 20 m/s in a shaft used only for ventilation then the cost of enlarging the shaft should be reviewed. It will be recalled that the total cost vs. diameter curve is usually fairly flat above its minimum point (Section 9.5.5.3).
7. Determine the coefficient of fill, \( C_F \), for the largest of the proposed conveyances (Section 5.4.6.3). If this exceeds 30 per cent for two or more conveyances, or 50 per cent for a single conveyance shaft, then the dimensions of the skip or cage should be reviewed or the size of the shaft, again, be re-examined.
8. Calculate the maximum relative velocity between the airflow and the largest conveyance, \( \frac{(u_a + u_c)}{(1 - C_F)} \) (See section 5.4.6.3 for nomenclature.) If this exceeds 30 m/s then additional precautions should be taken to ensure aerodynamic stability of the moving conveyances. In any event, the relative velocity should not exceed 50 m/s.
9. Assess the air velocities at all loading/unloading stations and, if necessary, redesign the excavations to include air bypasses or enlarged shaft stations.
10. Determine the total resistance of the shaft using the methods described in Section 5.4.6. Examine all feasible means of reducing the resistance including the streamlining of buntons and the aerodynamic design of intersections. For shafts of major importance, construct and test physical or CFD \(^2\) models of representative lengths of the shaft and main intersections.
11. Rerun ventilation network analyses with the established value(s) of shaft resistance in order to determine the final fan pressures required.

It is clear that there is a considerable amount of work involved in the design, sizing and costing of a proposed mine shaft or major airway. Fortunately, microcomputer program packages are available that reduce the effort to little more than assembling the essential data. These are particularly valuable for carrying out sensitivity studies in order to assess the effects of changes in design or financial constraints.

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\(^2\) Computational Fluid Dynamics
9.6. TRADITIONAL METHOD OF VENTILATION PLANNING

The methods of subsurface ventilation planning described in the previous sections of this chapter rely upon the availability of computers and appropriate software. A question that may arise is how such planning can be carried out without high speed computational aid. For this, we may revert back to the traditional methodology that was well developed prior to the computer revolution. Although the older techniques cannot begin to match the speed, versatility and detail of computer assisted planning, they do retain a role in estimating generic values for airflows, fan pressures and air quality at the early conceptual design stage of a proposed new mine or extension to an existing facility.

The traditional approach proceeds along the following sequence:

1. Determine air volume flows required in working areas. This can employ the methods discussed in Section 9.3. However, at a preliminary stage of planning, empirical values of airflow based on rate of tonnage might be used. In this case, care must be taken to ensure that the basis of the empirical guidelines is compatible with the intended mining method and geology.

2. Assess the airflow requirements for development areas, mechanical or electrical plant and workshops, and estimate the volume flows that pass through abandoned workings, stoppages and other leakage paths. The estimation of leakage flows relies strongly upon the experience and intuition of the ventilation engineer. Unfortunately, in many mines the volumetric efficiency (Section 4.2.3) is fairly low. Inaccuracies in the estimates of airflow through individual leakage paths, while perhaps of little consequence when considered individually, will accumulate into major errors in the main ventilation routes. The square law, \( p = RQ^2 \), then produces twice the corresponding percentage error in the frictional pressure drop. The treatment of leakage flows is probably the single greatest cause of imprecision in the traditional method of ventilation planning.

3. Indicate the estimated airflows on a mine plan and compound them to show air flowrates, \( Q \), through every major airway.

4. Using the given airflows and proposed sizes of airways, determine the corresponding air velocities. If these exceed limiting values (Section 9.3.6), the need for larger or additional airways is indicated.

5. Assess the resistance, \( R \), of each branch along the main ventilation routes, either from estimated friction factors and airway geometries or on the basis of local empirical data.

6. Using the square law, \( p = RQ^2 \), determine the frictional pressure drop, \( p \), for each main branch and indicate these on the mine plan.

7. Starting at the entrance of a downcasting surface connection, trace a path along intake airways to the most distant workings, through those workings, and back to the surface via return airways. Sum the frictional pressure drops around the complete traverse. This exercise is repeated for a number of such traverses to incorporate various working areas. The loop showing the greatest summation of frictional pressure drops then gives an approximation of the main fan pressure required to ventilate the mine. Subsidiary circuits may be controlled by regulators or upgraded by booster fans\(^3\). Pressure gradient diagrams can be employed to give a visual indication of the cumulative pressure drops.

The traditional approach is similar, in principle, to that used for the design of duct systems in buildings. It is simple in concept and requires little computational aid. Unfortunately, it suffers from some severe drawbacks:

1. It relies strongly upon the experience of the engineer and his/her empirical knowledge of the airflow distribution patterns. A mine is very different to a duct system in a building, not

\(^3\) where booster fans are permitted by the governing legislation.
only in scale but also because of the dynamic nature of mining operations, and the
tremendous variability in the geometry of airflow paths with respect to time as well as
location.

2. The highly interactive and nonlinear relationships that exist between ventilation
parameters are largely ignored. Leakage airflows through caved strata, old workings or
across stoppings, doors and air crossings are dependent upon the geometry of the flow
paths, the pressure differential and the degree of turbulence. Ascribing fixed values or
even fixed proportions of available airflow to leakage can do no more than achieve rough
approximations. In many mines, the majority of total airflow passes through leakage
paths. Errors in estimated leakages will accumulate and be reflected in the corresponding
main airflow routes and, because of the nonlinearity of the laws of airflow, can result in
large errors in the cumulative pressure drops.

3. There is a basic lack of reality in a procedure that estimates an airflow pattern and then
backtracks to find a fan pressure that will produce airflows capable of being manipulated
into the required distribution. In practice, when a fan is switched on, a complex
configuration of interdependent pressure drops and airflows is set up throughout the
network, the ventilating pressures producing airflows and the airflows, in turn, producing
frictional pressure drops. In effect, the airflows shown by the traditional method are
simply the initial estimates based on desired airflows in the workplaces and assumed
leakages. No attempt is made to simulate the actual airflow distribution that will occur
when the fans are switched on.

4. There is very little opportunity to study alternative options in order to optimize the
effectiveness and operating economics of the ventilation system.

5. Assuming the airflow distribution allows little flexibility in investigating the effects of fan
duty/position, or the adjustment or re-siting of regulators, doors and booster fans.

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