Chapter

18

Ventilation and Air Conditioning

18 Ventilation and Air Conditioning

18.1 Introduction

The historic role of ventilation was to provide a flow of fresh air sufficient to replace the oxygen consumed by the miners working underground. Contemporary mine ventilation primarily deals with noxious gases (mainly generated by trackless equipment underground). Ventilation effectiveness in this role depends on a simple fact: "once the noxious gases are mixed with air, they will remain uniformly diffused and will never separate." Therefore, if the problem gases (NO_X, SO_2, CH_4, CO, etc.) are diluted at their source with enough fresh air to render them harmless, they will remain safe until eventually exhausted from the mine. In the typical underground trackless mine, the amount of ventilation air required to ensure adequate dilution is far more than the amount required to replace oxygen consumed underground by personnel and diesel engines. The required amount of air is also sufficient to improve visibility and remove rock dust generated underground to the extent that silicosis is no longer a serious threat.

Today, LHD engines in underground hard rock mines are equipped with catalytic exhaust scrubbers to complete combustion of problem gases that is accomplished at an efficiency of approximately 90%. The LHD engines also produce minute solid particles (diesel particulate matter – DPM) due to incomplete combustion and impurities in the fuel. This matter consists of impregnated carbon and a variety of organic compounds, such as paraffin, aldehydes, and poly-nuclear aromatic hydrocarbons. Some of these compounds are recognized carcinogens. Unfortunately, the catalytic scrubber is not efficient at removal of these particulates and moreover they may not remain uniformly distributed in the exhaust air of the mine (they are subject to stratification). Dealing with this problem has recently become a prime focus of attention by regulators and operators.

The highest operating cost to provide contemporary mine ventilation is the electrical energy for the fans, which typically represents more than one-third of the entire electrical power cost for a typical underground mine.

The minimum quantity of fresh air is stipulated in the mine regulations that apply at the mine's location. The legal minimum is normally sufficient; however, an increase may be necessary when the mine regulations are insufficient (some developing countries) or to cool a hot mine.

Uranium mining ventilation is governed by different considerations and separate mine regulations. Uranium mines and other mines that encounter natural radiation are an advanced science that is not pursued in this handbook.

For common ventilation calculations, procedures assume the air is an incompressible fluid that answers to D'Arcy's equation. The formulas and calculations, based on work by Atkinson and McElroy, employ empirical "friction factors" that do not take into full account variations in pressure, temperature, evaporation/condensation, etc. In most cases, the simplified procedures yield satisfactory results; however, when mine air must be circulated over a significant vertical distance, or when air is required for cooling, a more sophisticated analysis is usually necessary.

Even when the simplified formulas are used, the calculations required for analyzing the network of airways in an existing or proposed mine are cumbersome. The difficulty is exasperated because the ventilation circuit for an operating mine changes day by day. In response, the McPhersons pioneered computerized modeling of mine networks. Today, most network ventilation problems are solved by computer using in-house programs or off-the-shelf commercial software.

In temperate climates, ventilation air may have to be heated during the winter months to provide comfort to the miners and avoid freezing the workings. Conversely, mines in Arctic regions may need cold air all year to maintain the permafrost regime. These “cold” mines are designed to operate below the freezing point. Hot mines in temperate or tropical climates typically require the air to be cooled. Deep underground mines always encounter warmer rock temperatures and the air is naturally warmed by adiabatic or auto-compression as it travels downward. Cooling by means of the ventilation air alone can become inadequate. More efficient cooling is obtained by chilling and adding ice to the process water delivered underground. Less efficient local (spot) cooling is provided by the release of compressed air underground and other means.

18.2 Rules of Thumb

General

- An underground trackless mine may require 10 tons of fresh air to be circulated for each ton of ore extracted. The hottest and deepest mines may use up to 20 tons of air for each ton of ore mined. *Source:* Northern Miner Press
- A factor of 100 cfm per ore-ton mined per day can be used to determine preliminary ventilation quantity requirements for most underground mining methods. Hot mines using ventilation air for cooling and mines with heavy diesel equipment usage require more air. Uranium mines require significantly higher ventilation quantities, up to 500 cfm per ton per day. Block cave and large-scale room and pillar mining operations require significantly lower ventilation quantities, in the range of 20 to 40 cfm per ton per day for preliminary calculations. *Source:* Scott McIntosh
- Ventilation is typically responsible for 40% of an underground mine’s electrical power consumption. *Source:* CANMET
- If the exhaust airway is remote from the fresh air entry, approximately 85% of the fresh air will reach the intended destinations. If the exhaust airway is near to the fresh air entry, this can be reduced to 75%, or less. The losses are mainly due to leaks in ducts, bulkheads, and ventilation doors. *Source*: Jack de la Vergne
- Mine Resistance - For purposes of preliminary calculations, the resistance across the mine workings between main airway terminals underground (shafts, raises, air drifts, etc.) may be taken equal to one-inch water gauge. *Source:* Richard Masuda
- Natural pressure may be estimated at 0.03 inches of water gage per 10 degrees Fahrenheit difference per 100 feet difference in elevation (at standard air density). *Source:* Robert Peele

Airways

- The maximum practical velocity for ventilation air in a circular concrete production shaft equipped with fixed (rigid) guides is 2,500 fpm (12.7m/s). *Source:* Richard Masuda
- The economic velocity for ventilation air in a circular concrete production shaft equipped with fixed (rigid) guides is 2,400 fpm (12m/s). If the shaft incorporates a man-way compartment (ladder way) the economic velocity is reduced to about 1,400 fpm (7m/s). *Source:* A.W.T. Barenbrug
- The maximum velocity that should be contemplated for ventilation air in a circular concrete production shaft equipped with rope guides is 2,000 fpm and the recommended maximum relative velocity between skips and airflow is 6,000 fpm. *Source:* Malcom McPherson
- The "not-to-exceed" velocity for ventilation air in a bald circular concrete ventilation shaft is 4,000 fpm (20m/s). *Source:* Malcom McPherson
- The typical velocity for ventilation air in a bald circular concrete ventilation shaft or a bored raise is in the order of 3,200 fpm (16m/s) to be economical and the friction factor, k, is normally between 20 and 25. *Source:* Jack de la Vergne
- The typical velocity for ventilation air in a large raw (unlined) ventilation raise or shaft is in the order of 2,200 fpm (11m/s) to be economical and the friction factor, k, is typically between 60 and 75. *Source:* Jack de la Vergne
- The typical range of ventilation air velocities found in a conveyor decline or drift is between 500 and 1,000 fpm. It is higher if the flow is in the direction of conveyor travel and is lower against it. *Source*: Floyd Bossard
- The maximum velocity at draw points and dumps is 1,200 fpm (6m/s) to avoid dust entrainment. *Source:* John Shilabeer
- A protuberance into a smooth airway will typically provide four to five times the resistance to airflow as will an indent of the same dimensions. *Source:* van den Bosch and Drummond
- The friction factor, k, is theoretically constant for the same roughness of wall in an airway, regardless of its size. In fact, the factor is slightly decreased when the cross-section is large. *Source:* George Stewart

Ducts

- For bag duct, limiting static pressure to approximately 8 inches water gage will restrict leakage to a reasonable level. *Source:* Bart Gilbert
- The head loss of ventilation air flowing around a corner in a duct is reduced to 10% of the velocity head with good design. For bends up to 30 degrees, a standard circular arc elbow is satisfactory. For bends over 30 degrees, the radius of curvature of the elbow should be three times the diameter of the duct unless turning vanes inside the duct are employed. *Source:* H.S. Fowler

Ducts (continued)

- The flow of ventilation air in a duct that is contracted will remain stable because the air-flow velocity is accelerating. The flow of ventilation air in a duct that is enlarged in size will be unstable unless the expansion is abrupt (high head loss) or it is coned at an angle of not more than 10 degrees (low head loss). *Source:* H. S. Fowler

Fans

- Increasing fan speed by 10% may increase the quantity of air by 10%, but the power requirement will increase by 33%. *Source:* Chris Hall
- For quantities exceeding 700,000 cfm (330 m^3/s), it is usually economical to twin the ventilation fans. *Source:* William Meakin
- The proper design of an evasée (fan outlet) requires that the angle of divergence not exceed 7 degrees. Source: William Kennedy

Air Surveys

- A pitot tube should not exceed 1/30th the diameter of the duct. *Source:* William Kennedy
- For a barometric survey, the correction factor for altitude may be assumed to be 1.11 kPa/100m (13.6 inches water gage per thousand feet). *Source:* J.H. Quilliam

Clearing Smoke

- The fumes from blasting operations cannot be removed from a stope or heading at a ventilation velocity less than 25 fpm (0.13m/s). A 30% higher air velocity is normally required to clear a stope. At least a 100% higher velocity is required to efficiently clear a long heading. *Source:* William Meakin
- The outlet of a ventilation duct in a development heading should be advanced to within 20 duct diameters of the face to ensure it is properly swept with fresh air. *Source:* J.P. Vergunst
- For sinking shallow shafts, the minimum return air velocity to clear smoke in a reasonable period of time is 50 fpm (0.25m/s). *Source:* Richard Masuda
- For sinking deep shafts, the minimum return air velocity to clear smoke in a reasonable period of time is 100 fpm (0.50m/s). *Source:* Jack de la Vergne
- For sinking very deep shafts, it is usually not practical to wait for smoke to clear. Normally, the first bucket of men returning to the bottom is lowered (rapidly) through the smoke. *Source:* Morris Medd

Mine Air Heating

- To avoid icing during winter months, a downcast hoisting shaft should have the air heated to at least 5^{0}C. (41^0 F.). A fresh air raise needs only 1.5^{0}C. (35^0 F.). *Source:* Julian Kresowaty

- When calculating the efficiency of heat transfer in a mine air heater, the following efficiencies may be assumed.

  90% for a direct fired heater using propane, natural gas or electricity

  80% for indirect heat transfer using fuel oil

  Various Sources

- When the mine air is heated directly, it is important to maintain a minimum air stream velocity of approximately 2,400 fpm across the burners for efficient heat transfer. If the burners are equipped with combustion fans, lower air speeds (1,000 fpm) can be used. *Source:* Andy Pitz

- When the mine air is heated electrically, it is important to maintain a minimum air stream velocity of 400 fpm across the heaters. Otherwise, the elements will overheat and can burn out. *Source:* Ed Summers

Heat Load

- The lowest accident rates have been related to men working at temperatures below 70 degrees F and the highest to temperatures of 80 degrees and over. *Source:* MSHA

- Auto compression raises the dry bulb temperature of air by about 1 degree Celsius for every 100m the air travels down a dry shaft. (Less in a wet shaft.) The wet bulb temperature rises by approximately half this amount. *Various Sources*

- At depths greater than 2,000m, the heat load (due to auto compression) in the incoming air presents a severe problem. At these depths, refrigeration is required to remove the heat load in the fresh air as well as to remove the geothermal heat pick-up. *Source:* Noel Joughin

- At a rock temperature of 50 degrees Celsius, the heat load into a room and pillar stope is about 2.5 kW per square meter of face. *Source:* Noel Joughin

- In a hot mine, the heat generated by the wall rocks of permanent airways decays exponentially with time – after several months it is nearly zero. There remains some heat generated in permanent horizontal airways due to friction between the air and the walls. *Source*: Jack de la Vergne

- A diesel engine produces 200 cubic feet of exhaust gases per Lb. of fuel burned and consumption is approximately 0.45 Lb. of fuel per horsepower-hour. *Source:* Caterpillar® and others

Heat Load (continued)

- Normally, the diesel engine on an LHD unit does not run at full load capacity (horsepower rating); it is more in the region of 50%, on average. In practice, all the power produced by the diesel engines of a mobile equipment fleet is converted into heat and each horsepower utilized produces heat equivalent to 42.4 BTU per minute. *Source:* A.W.T. Barenbrug
- The heat load from an underground truck or LHD is approximately 2.6 times as much for a diesel engine drive as it is for electric. *Source:* John Marks
- The efficiency of a diesel engine can be as high as 40% at rated RPM and full load, while that of an electric motor to replace it is as high as 96% at full load capacity. In both cases, the efficiency is reduced when operating at less than full load. *Various Sources*
- Normally, the electric motor on an underground ventilation fan is sized to run at near full load capacity and it is running 100% of the time. In practice, all the power produced by the electric motor of a booster fan or development heading fan is converted into heat and each horsepower (33,000 foot-Lb./minute) produces heat equivalent to 42.4 BTU per minute. (1 BTU = 778 foot-Lbs.) *Source:* Jack de la Vergne
- Normally, the electric motor on a surface ventilation fan is sized to run at near full load capacity and it is running 100% of the time. In practice, about 60% of the power produced by the electric motors of all the surface ventilation fans (intake and exhaust) is used to overcome friction in the intake airways and mine workings (final exhaust airways are not considered). Each horsepower lost to friction (i.e. static head) is converted into heat underground. *Source:* Jack de la Vergne
- Heat generated by electrically powered machinery underground is equal to the total power minus the motive power absorbed in useful work. The only energy consumed by electric motors that does not result in heat is that expended in work against gravity, such as hoisting, conveying up grade, or pumping to a higher elevation. *Source:* Laird and Harris

Air conditioning and Refrigeration

- In the Republic of South Africa, cooling is required when the natural rock temperature reaches the temperature of the human body (98.6 degrees F). *Source:* A.W.T. Barenbrug
- A rough approximation of the cooling capacity required for a hot mine in North America is that the tons of refrigeration (TR) required per ton mined per day is 0.025 times the difference between the natural rock temperature (VRT) and 95 degrees F. For example, a 2,000 ton per day mine with a VRT of 140 degrees F. at the mean mining depth will require approximately 0.025 x 45 x 2,000 = 2,250 TR. *Source:* Jack de la Vergne
- The cold well (surge tank) for chilled surface water should have a capacity equal to the consumption of one shift underground. *Source:* J. van der Walt
- At the Homestake mine, the cost of mechanical refrigeration was approximately equal to the cost of ventilation. *Source:* John Marks

18.3 Tricks of the Trade

- To quickly estimate the annual power cost of a main ventilation fan in dollars, simply multiply the fan horsepower by 65 times the total cost per kWh in cents. For example, at 5 cents per kWh, the annual power cost for a 2,000 HP fan will be 2,000 x 65 x 5 = $650,000. If only the energy cost is known, multiplying by 100 (instead of 65) will take the demand cost and surcharges on the power bill into account. *Source:* William Meakin
- To quickly estimate the annual power cost for a mine's primary ventilation system, simply multiply the unit power cost in cents per kilowatt hour by the total pressure drop in inches water gage and again by the total airflow in cfm. Then divide by 80 to obtain the annual power cost in dollars. For example, at 5 cents per kilowatt hour, the annual power cost for a mine with 800,000 cfm at 20 inches water gage will be 5 x 20 x 800,000/80 = $1,000,000. *Source:* C. E. Gregory
- An in-line fan will collapse flexible ducting. This can be avoided by providing a gap between the fan and the duct behind it. *Source*: Bob Steele
- A silencer on a ventilation fan at surface for a shaft sinking or ramp drive is often not sufficient. Usually the answer is to employ a double silencer. *Source:* Denis Blais
- When a mine is being developed, a long heading from a shaft may be best served by a pull system (suction duct), while the shaft is already served by a push system. The conflict can be overcome by extending the suction duct up the bottom portion of the shaft. *Source:* Gabby Juteau
- If a pull system (suction duct) is employed for a long drive, it has the advantage of removing smoke (blasting fumes) much more rapidly. The end of the duct must be kept 200 feet (60m) from the face to prevent damage by fly rock. Unfortunately, this produces a "dead end" at the face. The problem is corrected by employing a small, portable auxiliary fan that can force fresh air towards the face – often through an 8-inch (200 mm) diameter flexible ducting. *Source:* Robert Mayo
- In a long drift or diesel alimak raise heading that is too small in section to accommodate a typical ventilation duct, a much smaller diameter duct may be employed if a "blower" (low pressure air compressor) is employed instead of a ventilation fan. *Source:* McNally and Sons
- The published friction factors for hard rock mines found tabulated in textbooks were typically derived many years ago when mine headings were much smaller than they are now. In addition, the figures were intended to include an allowance for shock losses at entrance or exit. These values are not accurate enough to provide reliable answers for today's mines, especially for long headings. Using 2/3 of the values listed will often give answers that are reliable for most modern mining applications. *Source:* Jack de la Vergne

- The estimation of the required water gage of a mine fan by different engineers for the same application can vary widely. This is one reason why axial flow fans are usually selected for underground hard rock mines. If the motors and switchgear are ordered oversize, the problem can be addressed. *Source:* Chris Hall
- Vane-axial fans are not always the best choice for primary circuits. Centrifugal fans are cost competitive, quieter, less likely to have problems (stalling, vibration, balance), and require less maintenance. *Source:* John Marks
- For main fans, it is preferable to install two fans in parallel rather than a single unit. The reason is that one fan will supply 66% of the normal air capacity while the other (sealed off) is down for repairs. It is better still to select 3 fans, two of which will supply 90% of the normal air capacity. This is usually enough to maintain full production. *Source:* A.W.T. Barenbrug
- The main exhaust fans at a base metal mine should normally be equipped with stainless steel or aluminum blades to inhibit corrosion. *Various Sources*
- In a rail heading, the vent duct is usually best hung at the apex of the arch. In a flat-backed trackless heading, it will be best protected if it is hung high on the wall on the ditch side. *Source:* Jack de la Vergne
- A common misconception is that the addition of water vapor to dry air increases its density (“weight”). In fact, moist air is very slightly lighter than dry air at the same temperature and pressure, although its specific heat is increased. *Source:* Andy Pitz
- When calculating the pressure loss for a long heading served by a ventilation duct, only the resistance of the duct need be considered. The resistance of the heading is negligible by comparison. *Source:* William Meakin
- In large stopes, it is often not practical to provide through ventilation at the velocity required to clear smoke. A high-velocity auxiliary fan (with no duct attached) may be used to direct an air stream sufficient to displace smoke from a working area. *Source:* Bill Wright
- A change to seven days per week operations underground eliminates the opportunity to carry out large blasts on the weekend and may result in a large increase in air requirements to maintain acceptable working conditions. *Source:* Jozef Stachulak
- Most methods to calculate natural ventilation pressure are based on the assumption that it is due to differential air density. This assumption is erroneous. (Natural ventilation is caused by the conversion of heat into mechanical energy.) Nevertheless, considering air density provides results accurate enough for practical application. *Source:* Howard Hartman
- To prevent turning vanes in a duct from “singing,” it is often advisable to lace them with wires welded across the leading edges of the vanes and to the duct wall. The wire thickness should be at least equal to the thickness of the vanes. *Source:* H. S. Fowler
- The air resistance of a mineshaft containing a manway may be significantly reduced (as much as 57%) by means of placing a smooth walled brattice to separate this compartment from the main airflow. *Source:* Glükauf (S. Bär)

- A haulage ramp should not carry ventilation air in the same direction and velocity as a loaded haulage truck. Otherwise, a cloud of exhaust smoke and dust will envelop the truck as it travels. *Source:* Bob Brown
- An exhaust shaft or raise from a deep mine should not carry ventilation air at velocities in the vicinity of 10m/s (2,000 fpm). Air entering the bottom of an upcast airway is saturated and as it rises, the temperature drops due to auto-decompression and vapor condenses. The droplets of water will fall or rise depending on the air velocity, except at about 10m/s, where they will remain suspended until a water blanket builds up that throttles the airflow. This will eventually (as little as two hours) stall the vent fans, at which point the suspended water blanket cascades down the airway, and the fan usually starts up again. *Source:* J. de V. Lambrechts
- In the heat balance calculation for a hot mine, the amount of heat generated by blasting operations, body metabolism, and electrical cables is relatively insignificant. It can be considered offset by the cooling effect of releasing compressed air – without sacrificing accuracy of the heat balance, as a whole. *Source:* Jack de la Vergne
- In the heat balance calculation for a hot mine, the amount of heat generated by the hydration of Portland cement (85-100 cal/gram) and fly ash binder in backfill is relatively insignificant. It can be considered offset by the cooling effect of the water and solids in the backfill sent underground at surface temperature – without sacrificing accuracy of the heat balance, as a whole. *Source:* Jack de la Vergne
- When ore or waste rock descends in a pass, the loss of potential energy is divided between attrition (comminution of ore/rock) and heat of friction. In the heat balance calculation for a hot mine, the amount of heat generated is relatively insignificant and may be ignored without sacrificing accuracy of the heat balance, as a whole. *Source:* Jack de la Vergne
- In the heat balance calculation for a hot mine, the amount of heat generated by broken ore and rock is typically the principal source of heat input. It can be significantly increased in a base metal mine by oxidation of the broken ore. Therefore, the ore handling system should be designed to remove broken ore and rock quickly. In other words, shrinkage stoping, "deferred pull," and over-sized underground bins should be avoided. *Source:* Jack de la Vergne

18.4 Conversion Factors

Table 18-1 shows conversion factors using metric and Imperial units.

Table 18-1 Conversion Factors

Conversion using Metric and Imperial Units	
1 m^3/s =	2,120 cfm
1 *l*/s =	2.12 cfm
1 kPa =	4.02 inches water gage
1 bar =	100 kPa
1 bar =	14.50 psi
1 bar =	401.8 inches water gage
1 psi =	27.7 inches water gage

18.5 Constants and Typical Values

Table 18-2 shows the applicable constants and values.

Table 18-2 Constants and Values

Absolute zero	-459^0 F (-273^0 C)
Standard Temperature & Pressure	STP is 60^0 F @ 14.70 psi
Standard Temperature & Pressure	STP is 15.5^{0}C @ 101 kPa
Specific heat of air (constant pressure)	C_P =0.24 Btu/Lb./^{0}F
Specific heat of air (constant volume)	C_V =0.17 Btu/Lb./^{0}F
Specific heat of vapour (constant pressure)	0.45 Btu/Lb./^{0}F
Latent heat of vaporization	1060 Btu/Lb. of water
Latent heat of freezing	144 Btu/Lb. of water
Density of water vapour at STP	0.0475 Lb. /cubic foot
Density of dry air at STP	0.0764 Lb. /cubic foot
Density of saturated air at STP	0.0759 Lb. /cubic foot
Gas constant for dry air, $\gamma = C_P/C_V$	1.404

Notes

1 Btu = Heat required to raise the temperature of 1 Lb. of H_2O by 1°F
1 Btu ≈ Heat required to raise the temperature of 1 cu. foot of air by 60°F
1 Btu ≈ Heat required to raise 1 cfm of air through 1°F for one hour
1 Btu ≈ 1 kJ, 1 Btu = 1.054 kJ (kilojoules)
1 Btu = 778 foot-Lbs.
1 ton of refrigeration = 200 BTU/min = 3.517 kW

18.6 Design of the Primary Ventilation Circuit

McElroy first proposed an ideal ventilation circuit for an underground metal mine in 1935. He placed the fans on surface at two return airshafts on the extremities of the ore body. Fresh air was drawn down the operating (production) shaft, which was located near the center of the ore zone. Control of airflow was provided by doors placed on either side of the production shaft at each operating level.

Twenty-six years later, Hartman proposed a similar layout.

> *"The ideal arrangement of main openings is to locate the intake airway(s) at or near the center of operations and to ring the active mining areas with exhaust airways. In practice, this is never completely realized for obvious reasons.*
>
> *"Utilization of a hoisting shaft as a main airway may seriously damage both the operational and ventilation functions of the shaft. In cold climates (e.g. Canada), it is conventional to nearly neutralize the hoisting shaft. This practice requires provision of additional openings to serve as airways ... the separation of the hoisting and ventilation functions in shafts appears to be preferable."*

This latter circuit (with separate entries for fresh and exhaust air) is the one typically recommended by ventilation consultants and the one most desired by operators. Unfortunately, for very deep mines it is cost prohibitive. In this case the production shaft must be designed as a primary ventilation airway (normally the fresh air entry).

In the ideal case, with a simple application there is a main ventilation shaft or raise from surface at the extremities of the ore body, one for fresh air (FAR) and one for return air (RAR). With surface fans at both the FAR and RAR, the neutral point is centrally located within the mine workings and this arrangement is said to provide the best circuit to simplify the control over the distribution of air within the mine network. Forced ventilation with a fan only at the FAR is not usually employed except in temporary or special circumstances.

Recently, it has become popular to have the major fan installation at the RAR only. In theory, this arrangement avoids the requirement for air locks and (when there is a power failure) stagnant air from dead ends will not be drawn into active mine workings. It is also easier to eliminate velocity pressure loss with a properly designed evasée (outlet). The disadvantage is that the fans are more susceptible to erosion from exhaust air than clean air. In temperate climates, a small fan is usually placed at the FAR when mine air heating is required for the winter months. If the production shaft is downcast (normal preference), a separate entry into the shaft for ventilation air is provided. A small fan is installed to avoid pulling cold air down through the collar from the headframe during winter. The slight positive pressure from the small fan controls the leakage of heated air back through the shaft collar into the headframe.

When a ramp entry is required to be downcast with forced air, it is practical to drive a vertical entry raise (or separate horizontal entry) extending to surface from a point near the portal of the ramp. Leakage is prevented with the installation of double ventilation doors (air lock) between the raise and the mouth of the portal. When a ramp entry is upcast, warm saturated exhaust air from underground meeting cool ambient air on surface will precipitate a thick mist or fog at the portal, which may become a significant problem. In some cases, infra-red “fog cutters” have been employed with success, but a better means is likely to exhaust the return air through a short raise (or adit) similar to the one previously described for fresh air.

The design and construction of a separate ventilation entry to a ramp or adit is usually straightforward; however, the same is not true of the entry (“plenum” or “ogee”) into a production shaft. Normal practice is to sacrifice some losses by designing an entry that is safe, economical, and practical to build. The entry is best designed with a cross section equal or greater than the cross sectional area of the shaft (to avoid instability of the air stream due to expansion). The maximum air velocity in the plenum is restricted to 2,400 fpm (12m/s) to reduce the effect of shock and turbulence. When a right angle is required to meet a horizontal section that leads to the shaft, the outside corner is built square but later “smoothed” with falsework. At the shaft, the lip of the entry is permanently chamfered and small corners remaining in this “sub-collar” are filled with shotcrete.

Usually, the permanent mine ventilation will not be required until a long time after the production shaft collar is first constructed. To save time and conserve capital, it is common practice to construct only that portion of the ventilation plenum adjacent to the shaft along with the shaft collar. A temporary bulkhead is provided for this stub, to be removed at a later date when construction of the permanent ventilation system is required.

18.7 Natural Ventilation

In hilly or mountainous terrain, if there is a large difference between the temperature of the rock underground and the atmosphere, significant amounts of ventilation air will flow from an entry at one elevation to an exit at another. The airflow may become stagnant and then reverse direction from day to night or summer to winter. To provide reliable airflow, mechanical (“forced”) ventilation is required. It should be designed capable of accommodating (and not fighting) the natural ventilation pressure.

Many hard rock miners consider that natural ventilation is of no consequence to force ventilated underground mines that have entries at a similar elevation on surface. In fact, all underground mines are subjected to the effects of natural ventilation. Moreover, each individual loop in the underground circuit is affected.

In cool shallow mines that are less than 1,500 feet (450m) deep, the effect of natural ventilation is not reliable. The airflow due to natural ventilation can tend to flow in either direction, or not at all. Fortunately, the pressures generated by natural ventilation in this case are not significant and may be ignored in routine calculations for mechanical ventilation.

In hot shallow mines and in deep mines, the rock temperature is higher than atmospheric temperature, hence there is a transfer of energy to the ventilation air. The effect is to induce natural ventilation that invariably acts in favor of (improves) the mechanical ventilation system. It may be sufficient by itself to permit safe exit from the mine in the event of a major power failure.

18.8 Design of Ventilation Shafts and Raises

The area (A) of a ventilation entry to a mine required for a given flow of air (Q) may be quickly determined with rules of thumb that provide a typical design velocity (V) and the following elementary formula:

$Q = VA$ (metric or Imperial units)

Table 18-3 provides approximate diameters required for circular shafts and circular raises at different quantities of airflow for typical optimum design velocities. The diameters determined from these tables are useful for preliminary work and provide a means to check for planning blunders. The air velocities for open entries shown in the chart are sometimes exceeded in shallow mines (shorter lengths of entry). The velocities shown for equipped shafts should not be exceeded by more than 10% in any event.

Table 18-3 Diameters for Circular Shafts and Raises

Diameters for Circular Ventilation Entries to Underground Mines					
	Open Raw (Unlined)	**Open Raisebored (Unlined)**	**Open Concrete Lined**	**Equipped Concrete Lined Steel Sets**	**Equipped Concrete Lined Rope Guides**
Typical Design Velocity	2,700 fpm	3,300 fpm	3,200 fpm	2,200 fpm	1,800 fpm
Typical K	60-75	20-25	20-25	100 -150	25-30
Circular Shaft or Raise Diameter Required (feet)					
Airflow (cfm)	**Open Raw (Unlined)**	**Open Raisebored (Unlined)**	**Open Concrete Lined**	**Equipped Concrete Lined Steel Sets**	**Equipped Concrete Lined Rope Guides**
250,000	10.9	9.8	10.0	12.0	13.3
500,000	15.4	13.9	14.1	17.0	18.8
750,000	18.8	17.0	17.3	20.8	23.0
1,000,000	21.7	19.6	19.9	24.1	26.6
1,250,000	24.3	22.0	22.3	26.9	29.7
1,500,000	26.6	24.1	24.4	29.5	32.6

Example

Find the diameter, D, of a raise-bored hole equivalent to a proposed 10 by 10 (3m by 3m) alimak raise intended for ventilation.

Facts:
1. The raisebored hole is equivalent to a proposed 10 by 10 (3m by 3m) alimak raise
2. The alimak raise is intended for ventilation

Solution: 1. For an arbitrary length of raise (say 1,000 feet) the resistance, R of the alimak raise is R $=kPL/5.2\ A^3 = 70 \times 40 \times 1{,}000/5.2 \times 100^3 = 0.538$.

2. If D is the diameter of the raisebored hole, its resistance, $kPL/5.2\ A^3$ may be expressed as $R = 64kL/5.2\pi^2D^5 = 0.538$.

3. If k is 25 for the bored raise, then $D^5 = 64 \times 25 \times 1{,}000/27.6 = 57{,}970$ and D = 9.0 feet.

18.9 Friction Factor for an Equipped Mineshaft

The "friction factor" in this case includes skin friction and the frictional equivalent of turbulence (shock) resulting from disturbance to the air stream at the buntons and stations. The friction factor of a shaft that incorporates sets, utility lines, and conveyances is typically at least five times that of a bald shaft. This is one reason that production and service shafts are normally not used as primary ventilation airways; however, economics dictate that very deep mining operations fully utilize the production shaft for ventilation. These deep shafts normally employ an auxiliary conveyance instead of a manway to reduce resistance.

Using basic principles, the calculation of the friction factor for a production shaft is complicated. The commonly accepted procedure is to employ the method described by Bromilow, which is too lengthy to be included in this handbook, but may be found in published reference literature[1]. An easier method to determine the friction factor of a proposed production shaft is to use ratio and proportion from the measured resistance of a similar existing shaft. To complete this procedure, assumptions are made as to the portions of the total resistance of the shaft. In a typical calculation where the shaft diameters and set spacing are similar, the known resistance of the existing shaft may be divided as follows.

- 25% to the wetted perimeter of the shaft wall.
- 50% to the area of the sets facing the air stream.
- 10% to the conveyances
- 15% to shock loss at entry and exit.

The sets (buntons and dividers) are the culprits responsible for most of the resistance, which is one reason that shaft designers want the set spacing to be as far apart as practical. Increasing the set spacing from 5m to 6m (16½ feet to 20 feet) reduces the friction factor of the sets by approximately 8% and the shaft by about 4%. This small advantage is lost if the size of the buntons must be increased to accommodate the wider spacing.

A better opportunity to lower resistance is to use buntons that have a small drag coefficient. For this purpose, some shafts were equipped with fabricated diamond-shaped buntons. These are no longer employed, mainly because of the high fabrication cost. Today, deep shafts are invariably equipped with sets made of cold rolled rectangular sections, which are structurally stronger for their weight and relatively efficient with respect to drag. Figure 18-1 illustrates the drag coefficients.

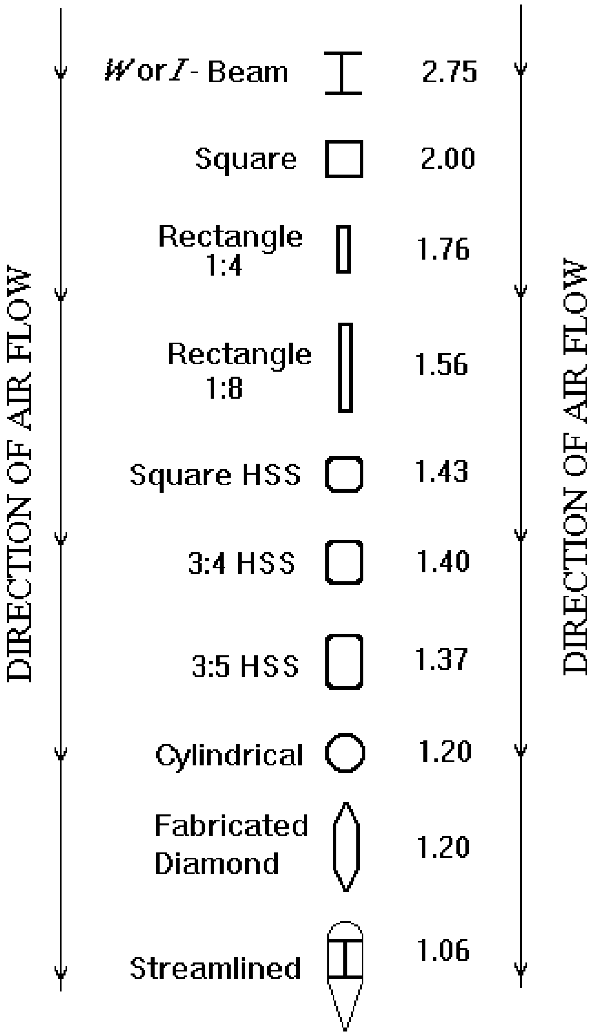

Figure 18-1 Drag Coefficients

By conventional wisdom, vertical installations in the shaft (guide ropes, electric cables, and pipes) ought to increase the shaft resistance because they increase the rubbing surface and slightly decrease the cross sectional area of the airway. However, they may be ignored when calculating the resistance of a proposed production shaft. This is because wind tunnel tests carried out in Germany[2] demonstrated the opposite effect: When measured in a wind tunnel, thin steel rods inserted longitudinally in a shaft model produced <u>less</u> resistance (as much as 13%). This phenomenon, discovered by S. Bär, is long forgotten, but should be revisited. If it is significant, discarded hoist rope suspended in an open ventilation shaft or raise could reduce resistance and hence power costs, for example. The flanged connections in a pump column remain a detriment to shaft resistance. Wherever practical, high strength Victaulic® style "low-profile" couplings should be used instead.

[1] Bromilow, John, *The Estimation and the Reduction of Aerodynamic Resistance of Mineshafts*, IME Transactions, London, Vol. 119, Part 8, 1960

[2] Bär, S., *Der Wetterwiderstand von Förderschâcten und die Moglichkeiten zu seiner Verringerung. Nach Modellversuchen.* Glückauf, 1949, 85, 327

Ventilation Velocity Case History

Table 18-4 shows the tabulation of ventilation velocities reported for actual hoisting shafts equipped with steel sets.

Table 18-4 Case Histories
(Velocities exceeding 9m/s)
Circular Concrete Shafts Equipped with Steel Sets
(Hoisting Shafts)

Location	Mine	Shaft	Diameter		Q		Air Velocity	
			m	feet	m^3/s	cfm	m/s	fpm
RSA	Vaal Reefs	No.2	7.9	26	458	970,000	9.3	1,827
RSA	Pres Brand	No.3	7.3	24	500	1,060,000	11.9	2,343
RSA	Pres Brand	No.5	7.3	24	500	1,060,000	11.9	2,343
RSA	West Deeps	No.3	7.9	26	448	950,000	9.1	1,789
Canada	Ojibway	Access	3.7	12	118	250,000	11.2	2,210
Canada	Williams	Production	7.9	26	472	1,000,000	9.6	1,883
Canada	Potacan	Production	4.9	16	231	490,000	12.4	2,437
Canada	Potacan	Service	4.9	16	231	490,000	12.4	2,437
Canada	Kidd Creek	No.1	7.3	24	467	989,000	11.1	2,186
USA	Texas Gulf	Green River	4.9	16	260	550,000	13.9	2,735
USA	Occidental	Service	10.4	34	944	2,000,000	11.2	2,203
USA	Barrick	Meikle	5.5	18	295	625,000	12.5	2,456

18.10 Ventilation Duct Design

Ventilation ducts are required for advancing most development headings, including shafts, drifts, and ramps. (Raise drives are normally ventilated with compressed air.) The two common types of ventilation duct are metal tubing (“hard line”) and fabric tubing (“bag”). Bag duct is only suitable for forced ventilation unless it is reinforced with spiral wiring that greatly increases its resistance. Ducts are normally circular in cross section; however, in special circumstances, oval or rectangular ducts are employed and these may be constructed of fiberglass, metal, or even concrete. These ducts may be sized for preliminary approximations on the basis of velocity. Table 18-5 provides calculated diameters required for ventilation ducts at different quantities of airflow for typical maximum design velocities. The nearest larger standard sized ventilation duct (i.e. 18, 24, 36, 48, 60, 72 inches) may be selected from the diameters shown in the tables for practical application.

Table 18-5 Ventilation Duct Diameters

	Type of Ventilation Duct			
	Hard Line (plastic/f-glass)	**Hard Line (metal)**	**Smooth Bag (plastic fabric)**	**Spiral Bag (plastic fabric)**
Typical Resistance (K factor)	13	15	20	60
Maximum Design Velocity (fpm)	4,000	3,750	3,350	2,250
Airflow (cfm)	**Minimum Duct Diameter (inches)**			
5,000	15	16	17	20
10,000	21	22	23	28
15,000	26	27	29	34
20,000	30	31	33	40
40,000	43	44	47	56
50,000	48	49	52	62
75,000	59	61	64	78
100,000	68	70	74	90

18.11 Selection of Electric Powered Ventilation Fans

Fans are first designed by determining the total pressure needed (H_t) to deliver the required quantity of air (Q). H_t is the sum of the static pressure and the velocity pressure.

$H_t = H_s + H_v$

or

TP = SP +VP

The velocity pressure may be ignored when designing main fans because they normally incorporate a well-designed inlet/outlet. Velocity pressure is always ignored in the case of an in-line booster fan on a hard line duct; however, the velocity pressure is normally considered in the case of a single fan installation for primary development or shaft sinking because the duct outlet is abrupt.

Example

Find the approximate fan design pressure and fan horsepower required for sinking a circular concrete ventilation shaft.

Facts:
1. The circular concrete ventilation shaft is 18 feet in diameter
2. The shaft will be sunk to a depth of 800 feet
3. The desired velocity of the return air is 50 fpm
4. The friction of return air in the shaft is negligible
5. Bag duct ($k = 20 \times 10^{-10}$) is employed

Solution: 1. The quantity of air required is $Q = VA = 50\pi \times 9 \times 9 = 12{,}723$ cfm

2. To this 10% is added to account for leaks: $12{,}723 \times 1.1 = 14{,}000$ cfm
3. A duct size of 28 inches is first selected by interpolation from the table above
4. The nearest larger standard duct size of 30 inches is selected in this case
5. $SP = kPLQ^2/5.2A^3 = 20 \times 2.5\pi \times 800 \times (0.14)^2/5.2 \times (1.25\pi \times 1.25)^3 = 4.0$ inches
6. $VP = (V/4000)^2 = (Q/4{,}000A)^2 = 14{,}000/4{,}000 \times \pi \times 1.25 \times 1.25 = 0.7$ inches
7. $TP = SP + VP = 4.0 + 0.7 = 4.7$ inches
8. The fan $HP = TP \times Q/6350 = 4.7 \times 14{,}000/6350 = 10.4$ HP

In the absence of manufacturer's fan curves, it may be assumed that a fan for this service equipped with a silencer will have an efficiency of approximately 70%. A 15 HP vane-axial fan will likely be satisfactory.

18.12 Air Fans

Fans that are run by compressed air are often employed for small exploration or development headings where it is not practical to provide electrical power. These fans may be purchased in various sizes up to 24 inches in diameter; however, the most common size is 12 inches. This fan consumes approximately 75 cfm of compressed air and will provide approximately the airflow shown in Table 18-6.

Table 18-6 Air Fan Airflow

H_t (inches water gage)	2	3	4	5	6	7
Airflow, Q (cfm)	3,900	3,400	2,900	2,400	1,900	1,000

18.13 Testing Ventilation Fan Performance

The best place to measure a fan's flow is just upstream where the area of the cross-section is well defined and the airflow is less turbulent. The fan manufacturer usually provides suitable openings for inserting a Pitot tube. Twenty readings are commonly taken, five in each quadrant. The depth of the readings provides equal areas covered within the inlet, so that a simple arithmetical average provides a reliable value. The air velocity may then be determined from the average pressure by the following formula.

$$V = 1{,}100\,(VP/\gamma)^{1/2}$$

In which, V = velocity (feet/minute)

VP = average pressure (inches water gage)

γ = air density (Lbs./cubic foot)

Example

Find the air stream velocity (V), and flow (Q), at the following fan installation.

Facts:
1. The average of velocity pressure (VP) measurements is 1.00 inches water gage
2. The ambient air density is 0.0757 Lbs./cubic foot
3. The diameter of the inlet duct is 4 feet

Solution:
1. $V = 1{,}100 \times (1/0.0757)^{½} = 4{,}000$ fpm
2. $Q = VA = V \times \pi R^2 = 4\pi V = 50{,}000$ cfm

18.14 Threshold Limit Value

The threshold limit value (TLV) is the maximum safe concentration of a noxious gas or dust in the atmosphere underground. A SF is included in each TLV value. The SF varies from 10:1 for lethal gases, such as carbon monoxide (CO) to as low as 1½:1 for irritants, such as ammonia (NH_3). Only about one half of the total airborne dust is resprirable. Gaseous contaminants are measured in ppm (volume) and respirable dusts are measured in mg/m^3 (milligrams per cubic meter). Examples are provided in the Table 18-7.

Table 18-7 Contaminants and Respirable Dusts

Contaminant	**CO_2**	**CO**	**SO_2** **NO_x**	**NH_3**	**Limestone Dust**	**50% Silica Dust**
TLV	5,000 ppm	50 ppm	5 ppm	25 ppm	$5 mg/m^3$	$0.2 mg/m^3$

In Arctic mining operations, wet drilling underground is a problem because of permafrost. One base metal mine has been able to drill dry (with dust collectors) because the host rock is limestone; however, most hard rock mines have high silica content in the ore. Dust collectors are inadequate in this case, mainly because the TLV is in the order of 10 times more stringent. The same problem arises when road headers are employed in hard rock mines.

*Example**

Determine the approximate ventilation air capacity (Q) required for a diesel engine underground.

Facts:
1. The underground diesel engine consumes fuel at a rate of 0.45 Lbs./HP hour.
2. The sulfur content of the fuel is ½%.
3. The efficiency of dilution is 90%.
4. The ventilation air weighs 0.074 Lbs./cubic foot.
5. The entire sulfur content converted to sulfur dioxide (SO_2), which is 2.25 times as dense as air.
6. The TLV for SO_2 is 5 ppm.

Solution: 1. In one hour the engine will burn 0.45 x 0.005 =0.00225 Lb. of sulfur (MW = 32) to produce 2 x 0.00225 = 0.0045 Lb. of sulfur dioxide (MW = 64) for each horsepower.

2. The weight of air required for dilution to 5 ppm at 90% efficiency is

(0.0045 x 1,000,000)/(5 x 0.90 x 2.25) = 444 Lbs.

3. The ventilation required = 444/(0.074 x 60) = 100 cfm/HP (cubic fpm per horsepower)

* The solution presented for this example is based on simplification of the actual combustion process and hence may not be entirely accurate.

18.15 Diesel Particulate Matter

As pointed out in the introduction, dealing with the problem of respirable DPM is a problem that has recently become a prime focus of attention by regulators and operators. The emissions from diesel engines produce minute solid particles (DPM) due to incomplete combustion and impurities in the fuel. This matter consists of impregnated carbon and a variety of organic compounds, such as paraffin (wax), aldehydes, and polynuclear aromatic hydrocarbons. Some of these compounds are recognized carcinogens. Unfortunately, the standard catalytic scrubber (oxidation catalytic converter) is not efficient at removal of these particulates and moreover the particulates do not remain uniformly diffused in the exhaust air of the mine (they are subject to stratification).

The TLVs discussed in the previous section dealt in part with rock dusts that were measured in milligrams per cubic meter. DPMs are measured in micrograms per cubic meter. (One milligram = 1,000 micrograms.)

The most suitable instruments and correct sampling procedures for DPMs remain items of controversy, but in general terms, it may be said that the traditional emissions encountered in trackless hard rock mines averages approximately 700 micrograms/m^3. Proposed legislation in the USA (MSHA) sets a concentration limit of 400-micrograms/ m^3 after an 18-month initiation period. After five years, the limit would be lowered to 160 micrograms per m^3.

These limits represent a serious problem for mine operators. Some mining associations and individual operators are actively protesting the proposed regulations as well as the severity of the new limits – with sound arguments. Among other objections, it is pointed out that there is no scientific documentation that exposure to the current levels of diesel emissions is sufficiently dangerous to cause miners to “suffer material impairment of health or physical capacity.” At the same time, a number of larger mining companies in the USA and Canada are proactively engaged in and/or privately funding research aimed at lowering the present level of emissions. The reader desiring additional information may refer to articles found at the MSHA web site (www.msha.gov) and a number of technical papers contained in the Proceedings of the 6th International Mine Ventilation Congress.

Following is a list of remedies that have been contemplated in anticipation of proposed new regulations.

- Electronic ignition (usually provided on new LHD equipment purchases) to improve combustion efficiency.
- Exhaust filters (sintered metal or ceramic based exhaust after-treatment devices).
- Fuel borne catalyst (as opposed to an exhaust-based catalyst) to improve combustion efficiency.
- Very low sulfur diesel fuel (reduces sulfite particulates). Regular diesel fuel sold in North America today by the major oil companies is extremely low in both sulfur and paraffin content.
- “Biodiesel” fuel derived from vegetable oils (approximately 3 times as expensive as ordinary diesel fuel).
- Engine replacement at 4,000 hours of service (expensive).
- Increase mine ventilation capacity (often not practical for an existing mine).
- Ban smoking in hard rock mines (after-smoke is detected as DPM).
- Eliminate rock drill oil for hand-held pneumatic drills and replace with semi-solid grease (oil mist may be detected as DPM). This remedy is already implemented at a number of hard rock mines for this and other reasons.
- Dusk masks, respirators, etc. for miners.

18.16 Heat Generated by the Auto-Compression of Air

When air descends in a mineshaft, it is heated by auto-compression. The potential energy possessed by the air at the top of the shaft is converted into heat energy by the time the air reaches the shaft bottom.

The increase in heat content due to auto-compression of 1 kg of air passing down a duct or dry shaft/raise may be calculated using the following formula (C_{pa} is the heat capacity of air in kJ / kg ·°C).

$$\Delta Q = \frac{\text{gravitational acceleration} \times \text{mass} \times \text{distance}}{1{,}000}$$

$$\Delta Q = 9.81 \times 1 \times \frac{100}{1{,}000} = 0.981\,\text{kJ/kg}$$

The increase in dry bulb temperature = $\Delta Q / C_{pa}$ = 0.981/1.02 = 0.96°C. This value corresponds closely to the rule of thumb that states that the dry bulb temperature will rise by one degree C. for each 100m that air descends in a ventilation airway. The wet bulb temperature (determined from a psychometric chart) will rise by approximately half this amount (assuming no transfer of moisture from the shaft wall to the air stream).

Burrows, Hemp, Lancaster & Quilliam, *The Ventilation of South African Gold Mines*, Cape & Transvaal Printers Ltd., Cape Town, 1974.

18.17 The Required Capacity of a Mine Air Heater

In cold climates, it is usually required to heat the ventilation air above the freeze point, otherwise ground water seeping into the ventilation entry will freeze. (In some cases, the ice build-up has been sufficient to eventually choke the airway.)

Certain mines have been successful in avoiding this requirement. For example, providing internal water rings in concrete lined shafts and raises will keep them watertight.

Most mining operations in temperate climates are required to heat the ventilation air during the winter with heaters using natural gas, propane, diesel fuel, or electricity. For this purpose, off-the-shelf mine air heaters may be purchased for development projects or small mining operations. Larger installations usually require custom-built heaters. In either event, an initial requirement exists to determine the capacity of the heater required (typically measured in Btu/hour). For this purpose, the following simple formula may be applied to obtain the value in Btu/hour units (1 kW =3,412 Btu/hour).

$$Q = 1.08 \times \Delta_T \times \text{cfm}$$

[The formula is based on standard air density (0.075 Lbs./cubic foot). Where this is significantly different from the actual density of the air at the mine's location, the formula should be extended to account for this (multiply result by local density/0.075)]

Example

Find the heater capacity required to raise the air temperature of ventilation air from -40°F to +40°F at standard air density.

Fact: There is 100,000 cfm of ventilation air.

Solution: $Q = 1.08 \times 80 \times 100{,}000 = 8{,}640{,}000$ Btu

18.18 Heat Load

Miners working in a hot environment may sweat two to three gallons of water in a shift. To help avoid heat stress, this water should be replaced with cool drinking water. Taking salt tablets is no longer believed beneficial. Most miners retain enough salt if they put a little extra in their diet. At least one mine in the USA and one in Canada has provided *Gatorade*® for their miners to help restore electrolytic balance.

The natural rock temperature near surface of an underground mine is equal to the mean annual temperature on surface. The rock temperature rises about 1 degree F. for each 100 feet of depth; however, in hard rock mines, the gradient will vary as much as 50% higher or lower depending on the conductivity of the earth's crust at the mine location (exceptions exist). Ventilation alone may not be sufficient to remove the heat generated from freshly broken rock and new faces when the natural rock temperature reaches approximately 95 degrees F. To determine the amount of additional cooling necessary for a particular mine, a heat balance calculation is often made. Table 18-8 shows a typical example.

Table 18-8 Heat Balance

Origin	Description	Heat Loading (BTU/min)	Heat Removal (BTU/min)
Rock	Broken rock and wall rock	370,000	
Ventilation	Skin friction and shock losses	60,000	
Ventilation	Auto-compression	20,000	
Diesel engines	Mobile Equipment	40,000	
Electric	Motor and cable losses	20,000	
Electric	Lighting	1,000	
Explosives	Heat of detonation	4,000	
Ground water	Seepage	3,000	
Backfill	Hydration of cement	2,000	
Personnel	Metabolism	1,000	
De-watering	Hot water pumped to surface		30,000
Service Water	(not chilled)		40,000
Compressed air	Expansion		25,000
Ventilation air	(not chilled)		320,000
Refrigeration required	(to service water and ventilation)		106,000
	TOTALS	**521,000**	**521,000**

The tabulation indicates a refrigeration requirement of 106,000 Btu/minute, which is equal to 106,000/200 = 530 TR.

For a proposed mine, most of the quantities in the above table can be calculated with accuracy. One exception is determining the heat from the rock, which may include heat from oxidation of broken ore. Unfortunately, this is the major contributor to the heat load. Because the calculated quantity of refrigeration is obtained by difference, the inaccuracy of the heat load will be magnified in the figure obtained for the required quantity of refrigeration. Perhaps a better method is to identify one or two comparable mines and determine the refrigeration capacity required by ratio and proportion. It should be noted that the design basis for most North American mines has been 75 degrees (wet bulb) in the stope. In Europe, it is 80 degrees F. and the design basis is even higher in South Africa.

18.19 Cooling

Various types of cooling devices are employed for underground workings ranging from a simple atmospheric cooler ("swampy") to a mechanical refrigeration plant that produces ice on surface for delivery underground (usually in the service water). As a general rule, the deeper the mine workings, the more sophisticated the cooling plant.

Because mechanical refrigeration is very expensive, it is considered the method of last resort and every effort is expended in mine planning to avoid it or reduce its requirements. Following are some of the methods employed.

- Increase the mine ventilation capacity
- Route intake air through old workings near surface
- Reduce the amount of broken ore left underground in stopes and bins
- Convert diesel powered trackless equipment to electric
- Conduct heat tolerance testing for work applicants
- Provide a five-day acclimatization schedule for new hires
- Provide slightly saline (0.1%) drinking water for the miners
- Convince miners to drink more water than required to slake thirst
- Provide ice vests for the miners
- Provide air conditioned lunch/refuge rooms for rest breaks
- Replace ditches with sealed pipes
- Seal off old workings
- Provide air conditioned cabs for equipment operators
- Provide remote stations for equipment operators
- Increase the compressed air capacity of the mine (and provide "air movers").
- Work short shifts underground
- Hire miners from tropical climates

Each of these procedures is beneficial, but only the first two are of major significance. The second of these is limited in application to those mines with suitable old workings.

Increase the Mine Ventilation Capacity

Increasing the volume of ventilation air for a deep hot mine is only significant if it increases the velocity of air in the workings. There is a marked improvement in the comfort and work efficiency of the miners with increase in velocity, as shown in Table 18-9.

Table 18-9 Mine Ventilation Capacity

Velocity of Ventilation Air	Maximum Desirable Wet-Bulb Temperature[1]
50 fpm	75^0 F
100 fpm	81^0 F
200 fpm	84^0 F
300 fpm	85^0 F

[1] (Relative values based on approximately equal comfort and work efficiency)

An economical limit exists to the size of air entries and development headings to accommodate high air volumes. The economical limit seems to correspond to a maximum practical ventilation rate of approximately 250 cfm per ton of ore and waste rock broken per day. For an operating mine going deeper into hot ground, it can be an extraordinary expense to provide new ventilation entries. One solution is to re-circulate a portion of the underground air, passing it through filters and scrubbers. This procedure was first employed with success at Butte, Montana and later at the Homestake mine in South Dakota.

Route Intake Air through Old Workings Near Surface

Drawing the fresh air through old workings, pit rubble, or caved workings has proven the most successful way of avoiding mechanical refrigeration in temperate climates. For example, this method is applied at deep mines in the Sudbury, Timmins, and Red Lake mining areas of Canada with success. Not only does it provide cooling year around, but also the requirement (and expense) of heating the fresh air during the cold winter months is avoided. (The procedure is also employed in Wyoming, but mainly to avoid heating air in winter.)

18.20 Mechanical Refrigeration

Refrigeration plants are distinct from simple cooling devices incorporating water sprays only. In many instances, only mechanical refrigeration can provide the necessary cooling power for a hot mine.

The heart of a refrigeration unit is a compressor that pressurizes and thus heats a suitable gas, such as ammonia. The hot compressed gas is then cooled with a water spray in a condenser until it becomes liquid. Subsequently, it is allowed to expand through a valve, which cools it and restores it to a gaseous state. The cold gas is used to cool ventilation air on surface and chill (or provide ice for) service water by means of a heat exchanger. The cold service water, sometimes containing ice particles ("frazil ice"), is sent underground for drill water, dust spray, and use in underground bulk coolers (water spray) for localized air-cooling. Like air, water descending in a mine will lose potential energy and become warmer. In deep mines, the potential energy of the water is often recovered with a Pelton wheel generator.

In the past, primary mechanical refrigeration units were often installed underground. Today, refrigeration units are invariably installed on surface, for a number of reasons. One is that freons (CFC refrigerants) are no longer employed as the refrigerant gas in accordance with the Montreal Protocol. Alternative refrigerants, such as ammonia are a potential hazard underground. (It is reported that at one mine in South Africa this problem is to be overcome by locating an underground refrigeration plant that uses ammonia in the foot of an exhaust shaft.)

The capacity of a mechanical refrigeration plant is traditionally measured in tons. A ton of refrigeration will freeze one short ton of water at 32 degrees F. in 24 hours. This is equivalent to 200 BTU/minute or 3.157 kW.

Table 18-10 Case Histories of Deep Hot Mining Operations

Company	Mine	Unit	Location	Main Minerals	Scale Tonnes per day	Mining Average	Depth Deepest	Main Access	Ore Geom.	Dip or Plunge	Mining Method	Backfill Types	Primary Cooling	Refrigeration Capacity	Cooling per tonne/day of ore	Ventilation Capacity	cfm per tonne/day of ore	Status
Falco	Kidd Creek	No.3	Timmins	Zn Cu	3,400	1,900m	2075 m	Winze	MS Lenses	78-90^0	UC Blasthole	CRF	Pit Rubble	Nil		234 m^3/s	146	Operating
INCO	Creighton	No. 9	Sudbury	Ni Cu	3,800	varies	2255 m	Shaft	Intrusive	70^0	Modified VRM	Paste	Caved Zone	Nil		381 kg/s	177	Operating
Hecla	Lucky Friday		Idaho	Ag Pb	1,000	1,80 m	1900m	Shaft	Vein	85-90^0	LW-UCF	Paste	Chill Water					Operating
Homestake	Lead	Deep	South Dakota	Au	3,700	1,850m	2600m	Winze	Pipe/Lens	70-80^0	CAF	CSF	Bulk Air	10.9 Mw	2.95 kW/t/d	391 m^3/s	223	Standby
Bharat	Kolar	Champion	India	Au	900	2,400m	2740m	Winze	Vein		Shrinkage	CTF	Bulk Air	4.0 Mw	4.44kw/t/d			Closing
JCI	WAGM	South Main	RSA	Au	5,400	2,100m	2600m	Winze	Reef	15-20^0	Longwall	CTF	Chill Water					Operating
JCI/PD	WAGM	South Deep	RSA	Au	8,300	2,500m	2800m	Shaft	Reef	15-20^0	CAF	Stiff*	Ice Slurry	85.0 Mw	10.2 kW/t/d	924 kg/s	195	Starting
AAC	Elandstrand	Deep	RSA	Au	6,000	2,600m	3533m	Winze	Reef	21^0	Sequential Grid	CTF	Chill Water	42.0 Mw	7.0 kW/t/d	660 m^3/s	232	Operating
Goldfields	Harmony		RSA	Au	5,000			Winze	Reef		Longwall	CTF	Plate Ice			690 m^3/s	292	Operating
AAC	Western Deep	South	RSA	Au	6,190	3,300m	3700m	Winze	Reef		Sequential Grid	CTF	Vacuum Ice	75.0 Mw	12.1kw/t/d	700 m^3/s	240	Operating
AAC	Val Reefs	South	RSA	Au	6,000			Winze	Reef	15-25^0	Longwall	CTF	Chill Water	23.9 Mw	4.0 kW/t/d	720 m^3/s	254	Operating
Morro Velho	Mina Grande	Belo Horizonte	Brazil	Au	500	2,189m	2475m	Winze	Vein	12-15^0	CAF	Waste	Atmos-pheric	-	-	10.1 m^3/s	43	Closed
Lepanto	F.S.E.	Deep	Phil-lipines	Au	17,500	1,570m	1700m	Shaft	Plug	N/A	Blasthole	Paste	Ice slurry	72.0 Mw	4.1 kW/t/d	1321m^3/s	160	Planning