Correctly Estimating Primary Airflow Requirements for Underground Metalliferous Mines

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ABSTRACT

A common problem for many mines is insufficient total primary airflow or insufficient airflow in the correct places as the mine changes over its life. This has a number of impacts on the business operation and may result in the need for a major and unexpected capital upgrade of the system, cancellation of a planned production increase, reduction in production targets, increased operating costs, failure to provide contracted metals to customers or acceptance of substandard conditions in the workplace. In some cases, it can even result in premature closure of the mine.

In recent times, higher metal prices have given an expected economic boost to many mines, potentially prolonging their life but simultaneously requiring the mine to develop to deeper depths or requiring assets such as shafts and fans to have their service life extended.

Insufficient primary airflow is a particular problem in deeper mines as the ventilation circuits become longer, more convoluted, have higher shock losses and the actual airflow requirements increase due to a combination of leakage, dispersion of the workings and the problems of managing the additional heat loads due to higher virgin rock temperatures and autocompression.

This paper discusses the reasons why mines experience unexpected shortfalls in primary ventilation capacity with its associated impacts on metal targets, profits and frequently on the health, safety and morale of the workforce, as well as its productivity.

METHODS OF ESTIMATING MINE AIRFLOW REQUIREMENTS

The primary ventilation system is a major contributor to the capital and operating cost of most mines. It also has a major bearing on the health and safety of the workforce. Probably the most important single design parameter for the primary ventilation system is the overall airflow requirement and errors in correctly establishing this value have a wide variety of domino effects on other aspects of the mine design. There are a variety of methods of estimating primary airflow requirements in a mine (Wallace, 2001; Tuck, Finch and Holden, 2006; Watkinson and De Souza, 2001), including:

- benchmarking against operations with the same mining method and then pro-rating for different production rates (Calizaya, Sutra and Stephens, 2005);
- Ventsim™ modelling at key, specific milestones in the mine life (Ponce Aguirre, 2006);
- manual allocation of airflows to individual activities on individual working levels by month or quarter (or year) for the life of mine (Wallace *et al*, 2005); and
- estimations based on the total diesel engine fleet capacity (kW) and a statutory requirement, such as 0.05 m^3/s per kW of rated engine power (Lang and Ross, 1998).

It is very common to find feasibility studies that are based largely or entirely on the last of the above estimation methods. Unfortunately, there is almost no technical credibility for such an approach, and it inevitably results in underestimating the mine airflow requirements. If such a flawed strategy was true, then mines operating before the early 1960s (when there were no diesel engines underground) needed no airflow!

REASONS MINES SYSTEMATICALLY UNDERESTIMATE AIRFLOW REQUIREMENTS

Failure to provide for leakage in the auxiliary ventilation ducts

Most workplaces in underground mines are ventilated using auxiliary ventilation duct. It is the role of the primary ventilation system, in part, to provide sufficient fresh air to every auxiliary fan. However, leakage occurs in every duct between the fan and the working place. It is not practicable or economic to eliminate leakage in ducts. For most applications, a leakage of 30 per cent between the fan (duct inlet) and the face (duct outlet) would be considered a good practice outcome. Poor installations have leakage of 50 per cent or more.

For example, assume a heading is to have a 270 kW LHD operating in it. Using a statutory requirement of 0.05 m^3/s of air per kW of rated diesel engine power, 13.5 m^3/s must be provided to the face where the engine is working. At a duct leakage of 30 per cent, at least 13.5 + 30 per cent or 17.6 m^3/s must be passing through the fan. It is this higher value that must be provided by the primary ventilation system as the leakage loss in the duct is not reducible in a practical sense.

It is not uncommon to find operators installing a second fan and duct to overcome the lack of flow to a face where the underlying problem is, in fact, the poor installation or maintenance of the original duct sitting next to it.

Failure to provide for leakage in the workings

Leakage also occurs across every ventilation control – doors, stoppings, closed regulators, etc. Leakage occurs through orepasses and chutes and through active stopes. Leakage also occurs through old workings. These small airflow losses are frequently called *short-circuiting*, but they are leakage nevertheless. The magnitude of this leakage through both the active and old workings depends very much on the style of primary ventilation system, the mining method and the age and extent of workings. It can range from as little as ten per cent to as much as 300 per cent of the actual airflow that is usefully employed in the workplaces. To ignore or underestimate it results in a mine that has insufficient air available for active workplaces, which often then results in excessive contaminant build-up or recirculation. It is therefore essential that leakage is examined in detail for each operation, and a clear provision in the overall airflow estimates made for it.

Failure to provide for essential anti-recirculation, bypass flows

Most auxiliary fans in underground mines are hung from the back/roof or mounted on the floor. In both cases, it is essential for sufficient air to be bypassing the fan inlet so that the fan does not recirculate. This problem has a number of aspects to it.

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Single fan

A minimum allowance for this 'bypass' air should be 30 per cent of the air entering the fan or 0.5 m/s in the drive past the fan, whichever is the higher. This is also similar to saying that every portion of airway will have at least 0.5 m/s of airflow.

Using our example above, if the fan is hung in a 5 m × 5 m heading, then the flow past the fan must be the higher of 17.6 m^3/s + (5 × 5 × 0.5) (= 30 m^3/s) or 17.6 + 30 per cent (= 23 m^3/s). To be confident this fan will not recirculate, at least 23 m^3/s but probably closer to 30 m^3/s should be passing by the fan. Providing only 23 m^3/s means only 5.5 m^3/s is bypassing the fan, which is only about 0.2 m/s in this size heading. This is barely perceptible. Providing *auxiliary* ventilation of 13.5 m^3/s to the face therefore requires the *primary* ventilation system to provide 23 to 30 m^3/s at the fan inlet.

Varying fan duties and intake flows as the duct length increases

In practice, fans do not move a fixed airflow. They operate on a fan curve. A 180 kW auxiliary fan, for example, can deliver as little as 26 m^3/s or as much as 46 m^3/s, depending on the duct length and diameter. The anti-recirculation allowance should therefore be based on the maximum flow through the fan in the particular application. In our example above, if the fan and duct has been selected so that the fan can deliver the required airflow at 400 m, but the same fan and duct is used from the *start* of the development when the duct is very short, and the fan produces 46 m^3/s with only a short length of duct, then the airflow passing the fan should be such that the fan cannot recirculate even at this 'maximum' value. Alternatively, a smaller fan could initially be installed, or if a two-stage fan has been installed, then only one stage could be operated until the duct length has increased to the point where the second stage is also required.

Multiple fans on levels

An additional issue when using a single large fan with multiple branching ducts is the additional airflow required due to the higher bypass allowance compared to multiple smaller fans and single ducts. Consider the common situation in which four blind headings (each in parallel) must be ventilated from a common drive (Figures 1 and 2). In most cases only one LHD will operate in any heading at a time, but three headings need ventilation due to other non-diesel activities occurring simultaneously with the LHD (surveyors, electricians, geologists, pumpies, pipe fitters, etc). There are at least four ways commonly used to achieve this. They all have very different airflow requirements but also result in very different working conditions and very different operational issues:

- Option one is typical of Australian mines that are developing a new level where there is a return air raise on that level. The priority heading on the level is to reach the return air raise. Once the RAR is reached, the level has its own exhaust. Development of other headings can be set up to use a single large fan with multiple branching ducts to provide sufficient airflow for the LHD irrespective of which heading it is in, as shown in Figure 1. Assuming 13.5 m^3/s is required at the face for a 270 kW LHD each branch of the duct leaks 30 per cent of the face flow, then the total requirement back at the fan is 3 × 17.6 = 53 m^3/s. If the common drive is 5 × 5 m, then the bypass requirement at the fan based on wind speed and drive size is 0.5 m/s is 0.5 × 5 × 5 (= 12.5 m^3/s) and based on the 30 per cent bypass allowance is 21 m^3/s. The total flow into this level under option 1 would therefore be 53 + 21 = 84 m^3/s.
- Option two (Figure 2) assumes that the volume of air to the level can be based solely on the single diesel operating in the level, irrespective of the number of headings being ventilated. This is a common strategy employed in sublevel caves, partly because it avoids having ducts passing in front of draw points, where they are likely to be damaged during the frequent blasting of SLC rings. Under this ventilation design, the air from any heading can be 'reused' by all downwind headings as shown in Figure 3. The total requirement into the level is therefore only the requirement for the one heading with the diesel in it, ie 17.6 m^3/s plus the bypass allowance for a single fan or 12.5 m^3/s (in this case) for a total of 30 m^3/s. With the fans being daisy-chained and only one diesel on the level, this 'bypass' allowance of 12.5 m^3/s can be reused for each fan. Operators will argue that this is sufficient air as it is the legal requirement for the LHD and that is the only diesel operating. However, it is clear that any activity downwind of the LHD will be seriously contaminated. This cannot be called a good design.
- Option three (shown in Figure 3) is the same as option two, but provides sufficient air to the level as if the LHD could be in all headings simultaneously. The total airflow requirement for the level is 3 × 13.5 m^3/s = 54 m^3/s. Note that this is also sufficient to ensure no fan recirculates so no additional bypass allowance is required in this case. The advantage of this over option 2 is that more air is available for dilution of any dust or gases produced by the diesel, irrespective of which heading it is in.
- Option four (Figure 4) uses four individual fans with four ducts, but locates all the fans back at the start of the level so that their intakes are in parallel. Every face is being fed fresh air. The requirement for the level is the 90 m^3/s.

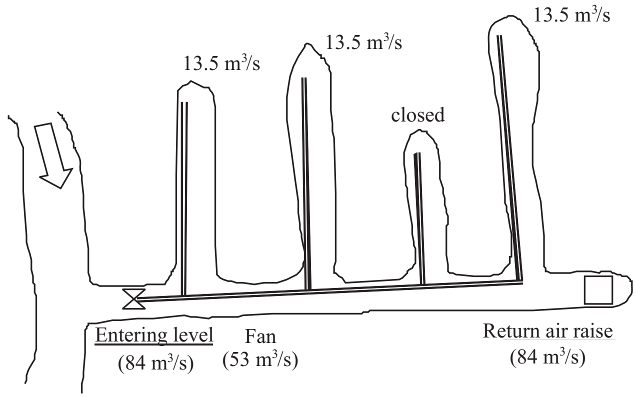

FIG 1 - Fan and duct layout and airflow requirements for single fan and branching ducts. This is typical of level development in many Australian mines, where the priority heading has been to reach the return air raise on the level.

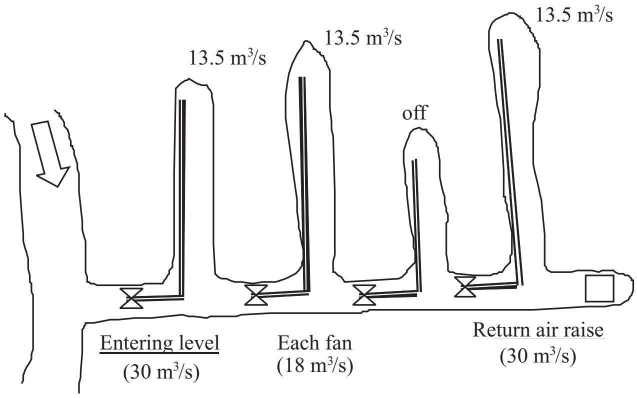

FIG 2 - Fan and duct layout and airflow requirements for daisy-chained multiple fans and ducts ('reuse'). This is a common strategy for many sublevel cave operations.

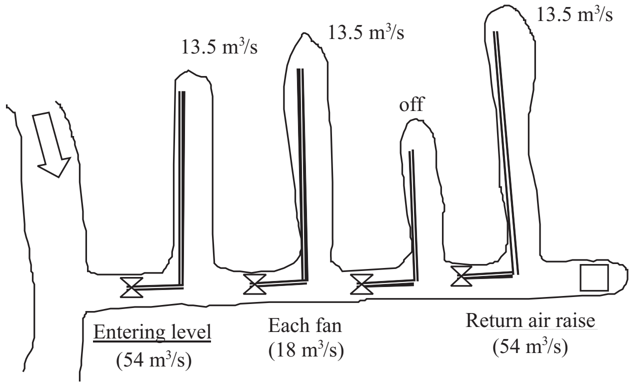

FIG 3 - Fan and duct layout and airflow requirements for daisy-chained multiple fans and ducts. This is also a common strategy for many sublevel cave operations.

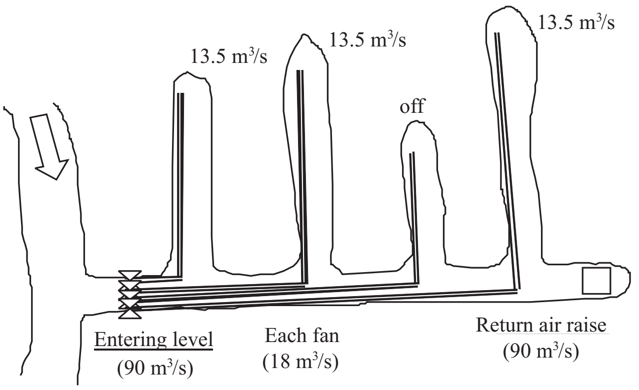

FIG 4 - Fan and duct layout and airflow requirements for parallel-intake multiple fans and ducts.

In each of these cases, many smaller mines would not install a return air raise on the level, so that instead of the level incasting, all air on the level would outcast back to the ramp. The fans would have to be hung in the ramp upwind of the level, making options one, two and three impractical.

The number of potential 'airflow estimates' for this one level with three ventilated headings is therefore very large, resulting in a potentially large variation in primary airflow estimates. There are clearly important health and safety consequences for each of these arrangements, especially for any option that has one fan downwind of any heading producing dust, gas, fumes or heat (eg bogging, shotcreting) as any downwind heading will be sending contaminated air to its face. This may be acceptable if the only downwind activity is (say) an inspection that occurs only infrequently (eg a pump inspection) but will not be acceptable for most mainstream activities unless crews are relocated while the upwind mucking operation is in progress.

Decisions regarding the ventilation set-up therefore cannot be made just on the basis of airflow requirements. Important additional issues that underground operators and mine design staff should be evaluating include:

- Options with multiple fans give more flexibility and control as it is easier to turn a fan on or off than to manually throttle a duct outlet open or closed. For example, option four uses the same amount of air as option one, but it is easier and more effective to turn off fans that are not required, than tie off duct ends that are not required. If duct ends are 'tied off', then the fan airflow drops back a little, but more air tends to come out of the ends that are still open. In addition, the duct that is tied off still leaks up to the tie point. Contrast this to the multiple fan installation; turning off one fan does not mean that the other fans move more air. Also fans and ducts that are off do not leak any air. Some similar flexibility is also offered by multi-stage fans, where only one stage can be used when the duct is short, and the second stage turned on when the duct resistance becomes higher as the duct becomes longer.
- Where multiple parallel ducts are used, there must be sufficient width across the back to accommodate the ducts. Option four is therefore probably not practical and if installed, will have high downtime due to the need for repairs and lose efficiency due to damage to the duct by passing vehicles. In some cases, two larger fans feeding two larger ducts with offtakes into each of the headings may be a better option.
- Where multiple fans are used, if one fan goes down, then the level can still operate three headings but if the fan goes down in option one, then no headings can operate on that level.
- Where a fan is upwind of another fan, if both headings are fired it may be difficult to get in to start the downwind fan. In this case, only the upwind heading is flushed of blasting gases. Only when this is completed is it safe to access the second fan to turn it on. Similarly, if the level exhausts back to the ramp, then re-entry times may become very long. Different strategies therefore impact on re-entry times and re-entry conditions.
- A flexible auxiliary ventilation system is very important in areas where there are many nearby headings. However, where multiple fans or ducts are employed with multiple offtakes, proper T-pieces or Y-pieces with proper duct closure systems (eg dampers) are essential, otherwise leakage or wastage will be excessive, usually resulting in substandard conditions in the more distant faces. In addition, the fans must be set up to draw from an uncontaminated fresh air source. Where these conditions are met, a two-parallel duct system can cover as many as eight to 12 headings (but not all being ventilated simultaneously).
- If blasting is going on in these development ends (eg they are draw points) then options one and four have to take ventilation duct past an active draw point with active LHDs in operation, etc. The ventilation arrangement is more easily damaged and less flexible.
- The use of a single common ventilation duct (option one) has to be larger to carry the higher flow (53 m^3/s) compared to the maximum duct flow in option two of 18 m^3/s, which means the development size must be bigger for option one.
- Offsetting this, of course, is the fact that options one and four draw air for all the headings from the one clean source, whereas options two and three effectively reuse part or all of the air.

The point is that this seemingly small difference in strategy (perhaps not even mentioned in the feasibility study) can have significant operating consequences and also affect the total primary airflow requirement and hence primary fan flow, pressure and motor size. Auxiliary ventilation strategy should therefore always be carefully and explicitly discussed in any feasibility study, as it is a key building block in the overall mine airflow estimate.

Where practicable, a better option for sublevel caves is shown in Figure 5 or Figure 6. In both cases, each heading is receiving its own split of fresh air. These strategies have been used in at least one recent sublevel cave operation in Australia.

Failure to provide for diesel equipment mobility

A development crew may have only one 'large' diesel (typically the LHD). However, this same crew will need ventilation duct run into at least four or five headings (and sometimes many more) due to the cyclical nature of the development activity in

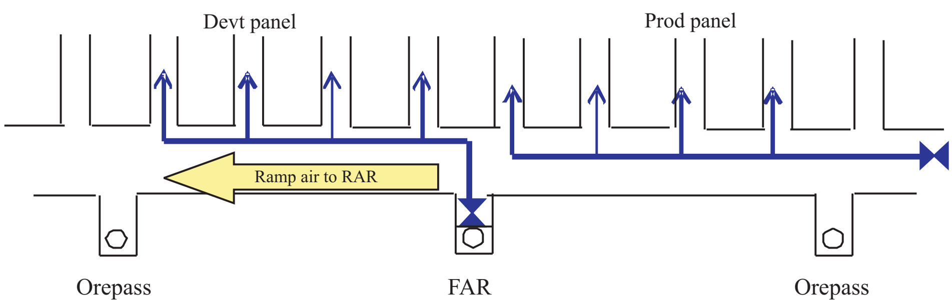

FIG 5 - Ventilation layout for a sublevel cave using decline air for production and air from a fresh air raise on the level for development. Production obtains fresh air from the ramp to the right. The downwind development activities receive fresh air from the level fresh air raise.

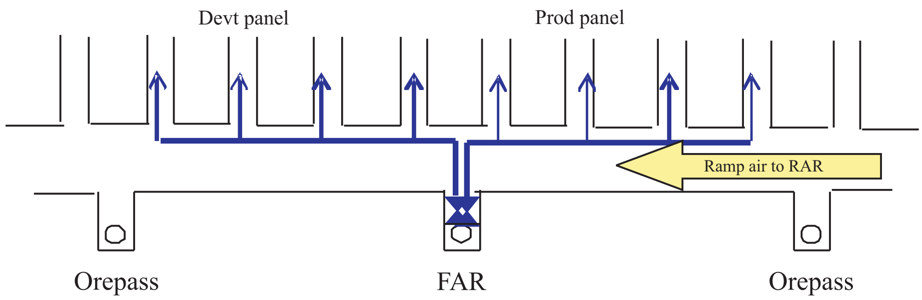

FIG 6 - Ventilation layout for a sublevel cave using air from the level fresh air raise for both production and development.

metalliferous mines and delays in various faces. These headings are usually not on the one level but may be spread out over the mine. Typical heading activities include: boring, blasting, mucking, ground support (immediate bolting, final bolting, cables, mesh, shotcrete, etc) and other services. In addition, leaving the ventilation duct 'on' helps drain the heading of heat, an important factor if the heading is long and the virgin rock temperatures are high. It is therefore naive to assume that when the LHD is not in a heading that no ventilation is required in that heading. Whilst it may be practical to 'reuse air' via daisy-chaining if all the headings are nearby, it is certainly not practical to do this if the headings are not nearby, in which case a higher demand on the primary ventilation system (both intake and exhaust) is required to service one mobile LHD working over several non-contiguous areas.

Failure to provide for ramps and other underground fixed plant and infrastructure and travelways

A relatively frequent problem in estimating airflow requirements is failure to provide for the bottom portions of ramps (or connecting pieces between ramps) to be ventilated. For example, once a ramp is completed to depth, pumps or other equipment are usually placed at the bottom of the ramp. Even if trucks do not travel to this depth, this leg (and any other leg) of the ramp, or any connections between ramps, must be provided with an adequate airflow. It is best to provide any ramp with a dedicated allocation of air to ensure that every leg is ventilated, and that no leg suffers from dead spots or low flows or flow reversals or recirculation.

Similarly, sufficient airflow must be provided for all other underground places where persons must work or travel, even if no diesel equipment operates in these areas. This includes workshops, magazines, cribrooms, battery charging stations, orepasses and crushers.

Therefore building up airflow estimates based solely on the airflow requirements for the larger diesel engines will result in these areas being overlooked and the overall airflow requirement underestimated.

Failure to understand the relationship between airway dimensions and minimum wind speeds

Twenty years ago, mine development was typically 4 × 3.7 m or 14.8 m^2. Achieving a minimum of 0.5 m/s in such a heading required 7.4 m^3/s. A more typical size for standard development today would be 5.5 × 5.5 or 30.3 m^2 requiring 15.2 m^3/s (over double) the airflow to maintain the same 'minimum standard' of 0.5 m/s. If every travelway in the mine needs at least 0.5 m/s, then doubling the cross-sectional area of the development doubles the airflow requirement, even if no diesels are in use in the mine at all.

Failure to recognise which is the critical airborne contaminant to be diluted

The airborne contaminants of most concern in most metalliferous mines have historically been dust (especially respirable silica), strata gases (if any), diesel exhaust gases and blasting fumes. Diesel engines and diesel fuel today are much cleaner than in the past and this has improved underground workplaces substantially; however, this has been offset by two things: the introduction of ever larger diesel engines (which intrinsically require more airflow) and the growing concerns regarding diesel particulate matter (DPM) and other contaminants, especially carbon monoxide and nitrogen dioxide.

Despite this, heat is often the most difficult contaminant to keep under control in many underground metalliferous mines in Australia. With the increasing depth of many operations and the continued reliance on diesel truck haulage to surface via ramps, this problem is becoming ever more significant.

There are several mines in Australia where the peak summer surface temperatures are 22° WB. The operations were originally shallow with the ramp as the main intake carrying typically 150 m^3/s. With four 50 t trucks operating in the ramp, a surface VRT of 28°C, a geothermal gradient of 20°C per vertical kilometre, and typical rock thermal properties, air temperatures in the ramp will be as shown in Table 1.

Since the ramp is the intake for the mine, the active workplaces must be fed with air taken off the ramp. An active

TABLE 1

Ramp temperatures at various depths.

Depth below surface (m)	0	250	500	750	1000
Ramp temperature (°WB)	22.0	25.3	26.4	27.9	29.3

development heading typically sees an increase in air temperature from the fan inlet to the face and back out to the return of between 2° and 5° WB, depending on the length of the heading and the activity at the time in the heading. Australian mines typically work to a maximum acceptable WB temperature of 32°C in the heading, so that a sensible design value to use for air entering the auxiliary fan on the ramp is 26.5° WB. In the above example, this means once this operation gets to about 500 vertical metres below surface, the ramp air temperatures will no longer be sufficient to keep the workplaces on the levels at an acceptable condition. In practice, the situation will be worse than this as this scenario assumes a constant 150 m^3/s down the ramp and that the *only* off-take is at the bottom of the ramp. If air is being progressively bled off the ramp into returns, or taken off the ramp and then returned to the ramp in a vitiated state, then ramp temperatures will increase much more quickly.

Clearly, the specifics will vary with the mine location and other factors; however, the point remains that even if the number of trucks remains the same (which implies a decrease in production with depth due to the increasing t-km), the ramp temperatures will rapidly approach limiting conditions. At this point, production from these deeper regions must be curtailed, or the operation has to accept unsatisfactory working conditions. The third option is to upgrade the primary ventilation system (eg with a dedicated intake shaft feeding the working levels, rather than the ramp) but this requires forward planning and a mine life that will justify the necessary capital expenditure.

As an aside to this issue, it is important for both managers and workers to remember that air-conditioned cabins do not prevent gas entering the cabin. There have been several incidents where LHD operators have been affected by gas after entering headings after stope blasts.

Failure to plan for reasonable capacity increases and other contingencies

Mines do need higher airflows (if other factors are kept constant) as the mine deepens. This is due to a number of reasons:

- Workings become more dispersed. Often 'old' levels still need some ventilation due to remnant mining or to inspect services (power, water, pumping) or for egress.
- Workings become hotter, due to both the increase in virgin rock temperature with depth, and the impacts of autocompression.
- The best value ore has been removed, resulting in a higher proportion of production from lower grade pillars or remnants. These almost always have higher airflow requirements that the more productive and high tonnage primary stopes.
- With declining grades as the mine ages, a typical response is to increase production to achieve similar metal targets. Higher production puts a greater demand on the ventilation system.
- More diesel equipment (burning more fuel) is required to haul the longer distances from lower depths.

Failure to provide for likely changes in diesel technology

Diesel engines are becoming larger and it is unlikely that this trend will stop. It has been common practice in the past (often tacitly or specifically approved by the authorities) to ignore 'small' diesel engines when estimating airflow requirements. However, consider these two facts: 25 years ago, underground utility vehicles (eg a Landcruiser) had engines in the order of 60 kW; today these same vehicles have engines twice this size. *In fact, the bulk of production LHDs 25 years ago had engines that were smaller than many of the service vehicles that are now in use!* Furthermore, the number of these 'small' diesel engines in use in Australian mines as a proportion of the total fleet has also increased substantially in the past 25 years as the workforce has become more mobile. Therefore, some allowance for diesel engine size increases and the increasing number and size of utility vehicles is also required.

Failure to provide for increased mine resistance

As mines become deeper, even assuming no additional airflow is required, the mine resistance increases because the primary airways become longer. In addition, every 'dog leg' or branch or obstruction in an airway results in an increase in the resistance of that airway due to *shock losses*. A typical 90° dogleg has a shock loss equivalent to about an additional 50 m of raise. Hence a 400 m vertical raise developed as a series of 40 m sections, each having a 90° offset, has an 'equivalent' length of 900 m, ie over twice the true length. If the offsets were each 10 m long, then the equivalent length increases to 1000 m. These shock losses can and often do become the dominating impact on deeper mines as the primary airways are often extended one section at a time as the mine develops far below its original planned depth. However, it is not uncommon to come across Ventsim™ models that provide for no shock losses at all.

In addition, it is not always practical to overcome this increased resistance by simply adding more fans in series, as this can lead to complex interactions between these fans and difficulties starting or operating the system.

Failure to understand the impact of both increased mine resistance and leakage on airflow requirements and fan performance

Most mining engineers understand that, in ventilation, the pressure loss is equal to resistance multiplied by the square of the airflow, ie $P = R\ Q^2$.

What they fail to fully appreciate is that older mines and deeper mines need more air just to maintain their production rate (due to longer hauls, higher VRTs and other effects noted above). Older mines also suffer from more leakage as the workings spread out. Both these factors mean that 'Q' must increase. The more extensive workings and higher shock losses also means that 'R' increases. The net effect is that the pressure loss through the mine increases dramatically. However, the response of the mine fan to the increase in 'R' is to reduce airflow, in accordance with its fan curve. So at a time when the system pressure requirement is increasing, the fan pressure capability is, in effect, decreasing.

Failure to understand the incremental nature first-cost of primary ventilation

Most mines want to keep their initial capital cost of construction and commissioning to the minimum practical value to maximise their NPV. This results in a focus on minimising contingency allowances for primary ventilation airflows or system resistances.

However, the incremental cost of some additional capacity in the primary ventilation system (shafts, fans, etc) at the time of original installation is almost always much less than the cost to expand the system at a later date.

Mining engineers see their role as developing safe, practical, robust and cost-effective designs for mines. However, frequently the ventilation system is designed to be at or close to its limit from the start, and the design itself often uses optimistic assumptions. By contrast, no mechanical engineer would consider designing a shaft winding system without taking into account normal operating constraints and contingencies (downtime, delays, etc). The building code used for surface structures already incorporates 'factors of safety'. It is therefore very important to examine the incremental cost of putting in a stronger ventilation asset base at the start of the project, and if this incremental cost does not seriously impact on NPV, then to put in the additional contingency. It is a rare mine that has ventilation over-capacity!

Failure to properly assess ventilation planning and implementation lead times

Often the planning time frame for the ventilation system in a mine is only a few months, or perhaps 12 to 15 months approaching the annual budget. This is often about the tenure of the ventilation officer or engineer. The net result is that it is common for mines to have a very short-term view on ventilation, frequently focusing on the secondary and auxiliary ventilation and neglecting primary ventilation planning. The implicit assumption is that future primary ventilation needs can be achieved by merely incremental changes, ie that the ventilation system is similar to a rubber band that can just be stretched a little more as required.

The flaw in this approach is that mines inevitably do need a major upgrade to their primary ventilation system at some point, especially when they are being extended beyond the provisions of the original feasibility study in depth, lateral extent or production rate. This need is not recognised due to the inadequate planning time frame. Once recognised, the lead time before commissioning is high as the plan cannot be implemented quickly enough either due to the need to obtain the various chain of approvals, or the difficulty of mobilising the additional contractor resources required to create new surface shafts or purchase and install new surface fans.

CONCLUSIONS

Failure to properly understand and proactively manage primary airflow requirements is impacting on a number of Australian mines and is likely to be an unexpected and unwanted 'sleeper' issue and unforeseen constraint in many more in the future. Many operations are struggling to capitalise on their potential for higher immediate profits and longer lives (and cash flows) resulting from the current high commodity prices due to ventilation constraints. Other operations are struggling to provide acceptable working conditions as they move deeper or expand laterally. A sound technical understanding of the increasing requirement for primary air as mines become deeper and hotter, as trucks become more powerful and as truck haulage (and diesel consumption) increase, is essential if underground mines are to provide the financial, safety and productivity outcomes that should be able to be realised by their owners.

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