Benchmarked data for current operations in Australia suggest that truck haulage tends to dominate for ore tipping depths up to 600 m from surface and at production rates below 1.5 Mt/a. However, the data also suggest a trend to decline access with truck haulage at depths up to 850 m and productions rates of up to 2.5 Mt/a.

For many mining projects, one of the key variables when analysing mine entry options is the increasingly high upfront capital costs associated with shaft development and related infrastructure. This, combined with high capital cost escalation for infrastructure development, continues to be a risk for many projects driven by increasing commodity and labour prices. High preproduction capital costs, combined in some cases with the limited ability to fully delineate the mineral resource for inclusion into the financial model, can have a significant impact on the payback period and project economics when using discounted cash flow analysis. In contrast, declines can provide quick access to the upper portions of the orebody for exploration, bulk sampling or production. Mobile equipment capital costs can be deferred in time by the use of a mining contractor and contractor-supplied equipment, which can have a positive impact on the overall net present value (NPV) of the project and mitigates risks of capital cost escalations.

A considerable amount of effort has been made in recent years to advance the practice of high-speed or rapid drill and blast development. This has led to significant gains in decline advance rates through the effective use of resources throughout the development mining cycle, and from the application of new equipment and new technologies. Ground support methodologies, such as the use of in-cycle fibrecrete in areas with poor ground conditions, and specialised rock-scaling and rock bolting practices have led to consistent advance rates in a variety of ground conditions. Advances in drilling are also having an impact, including long round drilling, high-powered drifters, automated drilling and drill boom positioning systems. While the profile sizes have generally increased to accommodate larger equipment and provide for additional ventilation capacity, the size of the equipment used in decline development has also increased, which has tended to offset the time required to excavate the additional material blasted during the development cycle. In Australia, advance rates for some projects have increased from an industry benchmark of approximately 5.3 m/d to over 8 m/d for a single heading decline.

Reduced time in accessing the orebody and commencing production has a significant impact on the NPV and capital payback time on most mining projects, which is favoured by most stakeholders. The expertise in rapid development, for both decline and mine development in general, is well established in Australia, North America and Europe. In contrast, advance rates for shaft sinking and related construction activities have not significantly improved over the last 15 years. Shaft projects still require specialised skills sourced from a relatively small labour pool. Most specialised sinking contractors are located in South Africa and Canada.

Mining methods continue to evolve as larger and more productive mobile equipment is used during the extraction process, allowing economy of scale aimed at reducing operating costs. These mining methods require efficient mobile equipment access between the various workplaces, which in most cases can be accomplished more effectively with decline access. It is becoming very common, even for smaller mines with narrow orebodies, to use a smaller fleet of 50 - 60 t haulage trucks and 6 - 8.5 m^3 loaders. In most cases it is not practical to move these large equipment units throughout the mine via a vertical shaft.

Currently a number of initiatives are under way which will continue to improve decline advance rates and decline development unit costs. These initiatives will continue to impact future comparison studies in decline versus shaft access options, mainly through more rapid access to the orebody for mining. Future developments in road headers and TBMs will continue to improve the economics of using these methods for mining projects.

DEVELOPMENTS IN UNDERGROUND MINE VENTILATION

Hard rock mining

A number of major influences, trends and developments in underground hard rock mine ventilation have been apparent over the past 15 years. Larger and more powerful diesel equipment has required larger development sizes in which to operate. Larger development in turn requires more airflow (m^3/s) to meet acceptable minimum wind speeds (typically 0.5 m/s).

The higher productivity from these machines frequently means more development faces need to be available to achieve an efficient process. The move towards total ground support has increased the number of stages in the average development cycle. Larger diesel engines burn more fuel and need more airflow to dilute noxious fumes and diesel particulate matter (DPM), as well as to dilute the heat from the engine. This has been partly offset by cleaner engines and cleaner (lower sulfur) diesel fuel, but the overall trend, also taking into account the growing number of light vehicles in underground mine fleets and the growing size of engines in these light vehicles, is towards higher total airflows.

The requirement for multiple headings to achieve high productivity in advance rates has created a trend towards larger ventilation ducts and larger (180 kW or larger) auxiliary fans, with branch lines coming off a trunk duct. This allows the branches servicing non-active faces to be closed off, redirecting available air to the branches feeding working faces. For ducted systems with a long life (such as the level ventilation for some sublevel caves), some operations have moved to rigid duct either spiral-wound galvanised iron or polyethylene.

The highly mobile nature of a modern underground mining fleet has resulted in air being reused by linking ventilated workplaces in series. This reduces the likelihood of nearly fresh air being sent to the exhaust. This trend away from one-pass (parallel) ventilation to series ventilation has reduced the ability for people to escape past a fire or products of combustion by using a parallel escape route. This in turn has meant it is frequently necessary to travel through smoke to get to fresh air or safety. This has resulted in hard rock mines universally requiring people underground to wear self-contained self-rescuers at all times. The duty of care towards emergency workers means it is no longer acceptable to risk emergency workers' lives (eg mine rescue teams) to rescue trapped workers. This has changed the emphasis from aided rescue to self-rescue.

The trends towards series ventilation and trucking of ore to surface (via a surface ramp) has created a commonly employed primary ventilation system in which the ramp is the sole mine intake. This often means the second means of egress (eg a ladderway connecting the levels) can be compromised by a fire in the first means of egress (the ramp). With no vertical intake shaft, there are no fresh air bases in the mine. Many operators no longer reticulate compressed air, or do not use (fireproof) steel pipes for compressed air. The combined impact of these factors has seen the adoption of fully independent, self-contained refuge chambers located throughout the mine, usually capable of supporting people underground for a minimum of 36 hours without assistance or resupply.

There is a trend towards completing long development headings as single headings using high-pressure, low-leakage flexible duct rather than as twin parallel headings (creating a flow-through ventilation circuit, using one heading as intake and one as return).

The developments in remote and tele-remote operation of LHDs, along with new developments in blasting technology, have allowed mining methods to be developed in which stopes do not have any exhaust (return). This has increased the dependence on auxiliary (ducted) ventilation systems, especially in longitudinal stopes in which the blind drawpoint where the production LHD is operating can be a few hundred metres long.

The increased size of development means that ventilation controls need to be large and expensive, and they are time-consuming to install and maintain. In many cases ventilation controls are not being installed at all where they would have been in the past, resulting in short-circuiting. A typical example is where a stope is open between levels, resulting in a short-circuit within the stope, creating a zone of low wind speed on the ramp connecting the same two levels.

Reductions in allowable personal exposure levels for respirable crystalline silica dust, and a growing awareness and increased regulatory focus on the health impacts of DPM, are impacting on airflow allowances, with the design value increasing from typically 0.04 m^3/s per kilowatt of rated diesel engine power in the late 1990s to 0.05 m^3/s per kilowatt (based on larger diesels only) to the proposed design value of 0.06 m^3/s per kilowatt (based on all diesels). The latter is broadly in line with most overseas western countries.

The change from predominantly eight-hour shifts to almost universal 12-hour shifts has changed the impact of many atmospheric contaminant limits (eg time-weighted averages or TWAs). In many cases, it has halved the allowable limit. For example, the TWA for carbon monoxide for an eight-hour shift is 30 ppm, but this decreases to 15 ppm for a 12-hour shift.

The change in shift length also impacts on working in hot conditions and thermal stress limits. The common earlier strategy of reducing the shift length (eg from eight to six hours) was increasingly unproductive compared with 12-hour shifts, and was also found to not be providing protection against heat illness. A more graduated system for managing hot conditions, usually focused around trigger action response plans (TARPs), has been adopted. Air-conditioned cabins have become the norm on most underground mobile equipment. This helps with heat stress and also dust exposures.

The substantial reduction in the cost of hand-held electronic gas sensors and the increase in the number of gases that can be measured have changed re-entry procedures after blasting, with most operations now explicitly testing for toxic blast gases prior to re-opening applicable workplaces.

Where mines are deep and it is not effective to use truck haulage to surface, long conveyors to surface have been used on some occasions where shaft hoisting would previously have been used. The additional fire risk of conveyors over hoisting shafts means the airways containing the conveyors are generally found, by risk assessment, to be better suited to use as neutral intakes (ie where the air is not reused elsewhere in the mine) rather than full intakes.

The very high usage of turbo-chargers on underground diesel engines has resulted in a range of additional precautions to prevent and/or ameliorate the impact of engine fires, including use of aqueous film-forming foam (AFFF) systems on all, or at least larger, diesel engines.

The trend towards deeper mines has resulted in a substantial increase in installed mine cooling (refrigeration) capacity. To date this has been almost entirely surface bulk air cooling, but some underground plants have been installed, generally with less than satisfactory results, frequently related to the difficulties of maintaining the plants or poor initial design, installation or choice of siting.

There has also been a trend to developing underground mines beneath existing or completed open cut operations. In some cases, problems have occurred with recirculation of air between the mine exhaust and intake or temperature inversions associated with the depth of the ventilation shaft collars within the open cut.

Whilst a number of operations have trialled various forms of 'ventilation on demand' (semi-automation of the ventilation system) and there is substantial interest in this to reduce mine airflow requirements, shaft and fan capital costs and mine operating costs (especially power), as yet there is no clear trend towards more general application of this technology. Apart from its initial cost, the main problems are maintaining the systems and the time (including production delays) and effort required for relocating sensors and automation components as fans and ventilation controls are relocated.

Developments in personal computer technology and software for solving mine ventilation problems means that more ventilation design options can be investigated. This includes not only airflow engineering, but also gas analysis, re-entry calculations after blasting and temperature and cooling investigations.

Coal mining

Coal mine ventilation is a primary control for many core hazards (frictional ignition, CH_4 or CO_2 seam gas accumulation, respirable dust and spontaneous combustion) for which there are, with some variation in specific values, internationally recognised standards. The scientific principles describing the behaviour of ventilation systems and their associated influence on coal mine hazards have been known for decades and remain valid today; for example, Atkinson's equation and Coward's triangle. However, the underground coal mining industry worldwide has a long history of accidents, many directly or indirectly related to the ventilation systems employed. Tragically, multiple-fatality accidents continue to occur in the coal mining industries of both developed and developing countries. The worldwide death toll due to methane ignitions in underground coal mines in 2012 will be of the order 3500-based on 2011 statistics.

The last fatal methane explosion in Australia was at Moura No 2 mine in 1994 (11 deaths due to a spontaneous combustion event leading to methane ignition). However, perhaps the most significant methane ignition in recent times was at Blakefield South mine in December 2010, where there was an ignition of gas in a large goaf and in close proximity to the face crew. The cause remains uncertain and not one worker was injured. It is understood that a similar event occurred at San Juan mine (USA) in September 2011, where again no workers were injured. In both these mines, ventilation and other systems were operated to promote an inert goaf atmosphere for the control of spontaneous combustion. In this respect, the now customary practice in most Australian coal mines of promoting inert goaf atmospheres, some with proactive nitrogen injection, has evidently resulted in safer conditions.

The volumetric capacity of ventilation systems employed in coal mines depends on the mining activities being undertaken and the ventilation allocation to each. For a typical longwall operation comprising a longwall, two gate roads and mains development the minimum volumetric capacity (without seam gas) would be about 50 + 3 × 40 = 170 m^3/s for panels and 200 - 250 m^3/s for the mine as a whole to allow for distributional inefficiencies. Without seam gas, it is the provision of acceptable velocity (minimum 0.3 m/s but 0.5 m/s preferred in development) and dilution of diesel exhaust emissions (typically 0.06 m^3/s per kilowatt) that are the design criteria.

With seam gas, the dilution capacity of development and longwall circuits are determined by limiting methane concentrations (1.0 per cent diesels, 1.25 per cent power, 2.0 per cent workers) and air velocities. The highest volumetric capacities for gassy single longwall operations in Australia currently range from 500 to 600 m^3/s, although the gassiest mines are emitting 6000 to 9000 L/s methane, well beyond the practicable dilution capacity of any ventilation circuit. It is also inevitable that higher longwall production rates will increase gas emission to the point that mines previously operating with ventilation alone will have to introduce gas drainage.

The prevailing norms in the Australian underground coal mining industry are increasing production rates, larger mining blocks to minimise development metres per tonne and re-enforcement of the need to operate with inert goaf atmospheres, with or without a significant risk of spontaneous combustion. In the early to mid-1990s longwall production rates of 2 Mt/a to 3 Mt/a were acceptable and face widths were typically 200 - 250 m. Today, Australian longwall production rates can exceed 7 Mt/a, with face widths up to 400 m. However, geological conditions often constrain development rates to below those achieved in US place changing panels, with the inevitable problem of maintaining positive development float. Consequently, most longwall mines employ two heading gate roads (1 km to 6.5 km in length) and as few mains headings as possible. The resulting design conflict is that of requiring higher ventilation rates in circuits with lower distributional capacity. As the volumetric capacity of the commonly employed conventional U ventilation circuit is determined by the face ventilation rate, it is not possible to change ventilation capacity pro rata with