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CFD simulation of methane roof layering in underground coal mines

Introducing a 0.1 mg/m³ limit for DPM in Western Australia: What are the impacts on the mine ventilation system?
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This book is a comprehensive reference source for ventilation professionals in dealing with challenges at the coalface.

It continues to be a valuable source of information for students in acquiring an advanced level of knowledge in the science of mine ventilation.
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In the previous editorial we considered the current state of affairs in the mining industry as well as the impact of energy and specifically, energy costs for underground mines. I concluded that it can only be described as an industry in immense distress.

This distress is fuelled by lowering commodity prices, slower than expected world growth and continuous cost increases. This is also one of the main themes of the planned MVS conference on the 1st and 2nd of September. The Conference theme deals with the relevance of our profession in a very cost constrained environment.

In this issue of the Journal, the use of controlled re-circulation is dealt with in great detail. This practice has not found nearly enough application as it is one of the easiest solutions to containing soaring energy costs when one considers all the associated risks carefully. Also in this journal, industry champions at the Mining Indaba consider the cost issue and highlight the impact of energy cost increases.

A recent article published in Australia looked at the impact that this low commodity cycle has had on direct employment. It highlights the support for industry employment, the impact on training at universities and obviously technical specialists, including mine ventilation engineers.

The effects of a start stop attitude by most mining houses result in a lack in capacity building when prices are low. But, when they attempt to play catch-up in good times, the high point in the commodity cycle is often missed. These are the same cycles that we in South Africa are exposed to, which have previously resulted in a very low number of students completing MEC studies, resulting in the subsequent skills shortage when the MHS Act needed implementation.

Furthermore, additional symptoms of the latest commodity cycle that lasted for almost 15 years were that many large mines were planned, but only came into production at the end of the cycle, resulting in a further over capacity when the specific commodities e.g. iron ore came on line.

Many of these operations have now been placed in care and maintenance resulting in significant mining job losses.

These issues impact on all employees within the mining industry including our profession.

In this issue of the Journal just a small number of the responsibilities of the ventilation engineer are addressed. These include energy efficiency in design and execution of our daily tasks; preventing mine explosions; and keeping our workforce healthy through good ventilation and hygiene engineering practices.

The role is obviously much wider than the highlighted three topics, but it is vital for mine management to understand the critical role that our profession plays in creating safe and healthy environments, where employees can work productively. This must be done in such a manner that we utilise resources optimally.

It can be concluded that the Mine Ventilation Engineer of the future will face further significant challenges as we acknowledge the principle of zero harm to workers.

We still need to implement many better and more advanced solutions to make sure we create a sustainable environment where health, safety and productivity is a natural outcome.
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Controlled recirculation of ventilation

Why is this not applied more often to save energy?

S.J. Bluhm and R.E. Funnell, BBE Consulting South Africa

ABSTRACT

Controlled recirculation has been examined often over the past 50 years as a means of reducing the quantity of air that must be delivered from surface. The potential benefits include reduced shaft sizing, reduced size of main fans with associated energy savings. Other spin-offs include reduced heat load from auto-compression of air in the shaft and better positional efficiency of underground recirculation air coolers, both resulting in reduced energy for mine cooling.

There have also been several arguments against the use of controlled recirculation, including risk of increased concentration of radiation and the potential inability to scrub certain gases and diesel particulate matter. But as energy resources are becoming scarcer and power more expensive, there are many more and stronger motivations that can be made in favour of applying controlled recirculation.

With the perspective on energy management, the paper first discusses power tariffs, recirculation simulation functions within VUMA and VUMA-live ‘real time’ simulation and control and it is noted that, together, VUMA-live and controlled recirculation could be considered as the platform for the ultimate ventilation-on-demand system.

The paper then examines three case studies, two real and one simulated. The Oryx mine had a recirculation scheme recirculating 300 kg/s into a downcast of 600 kg/s around a sub-shaft system that ran successfully for more than a decade and made great sense from an energy perspective. The Vale Taquari potash mine has been effectively and safely applying controlled recirculation for the past eleven years with a total of 200 kg/s being recirculated in two districts. The simulated case study relates to an ultra-deep mine with a large refrigeration component that presents a compelling picture for recirculation from an energy management perspective. Finally the paper discusses the development of a scrubber system that would enable the application of controlled recirculation and notes some areas of potential research.

BACKGROUND AND HISTORY

General interest in the use of controlled recirculation systems in mines has been around for many years. This is particularly true in deep or extensive mines where considerable energy costs are involved in providing all the ventilation air from surface. The use of controlled recirculation in mines is an established technology but it has not been that widely applied in terms of its potential to save energy.

However with the awareness on energy management and power costs worldwide and particularly in South Africa this will no doubt change. The power tariff for South African mines has, in recent years, increased dramatically e.g. 2008 - 46%; 2009 - 38%; 2010 - 25%; 2011 - 26%; 2012 - 17% and 2013 - 9%. Thus, in a period of six years, this power cost increased by more than 400%. While South African power is not particularly expensive compared to international standards, these changes have fundamentally modified the design and operating approaches for mine refrigeration systems and will make controlled recirculation more-and-more attractive.

There are great incentives to improve energy efficiency and gain power cost savings. Figure 1 shows the South African mining power tariff structure. Note the fairly large variations between diurnal peak and standard costs and between summer and winter. These diurnal and seasonal variations can be exploited in energy management systems associated with controlled recirculation. For example, in cold periods, the use of cold ambient air from surface does not incur a cost penalty, whereas recirculated air incurs the underground refrigeration penalty. There are benefits in managing the systems for less recirculation during cold periods, power is not particularly expensive compared to international standards, these changes have fundamentally modified the design and operating approaches for mine refrigeration systems and will make controlled recirculation more-and-more attractive.

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been considered in Canada; and for the efficient distribution of underground cooling in the Homestake gold mine in the United States. However, until the early 1980s, controlled recirculation was generally based on small-scale localised systems. There were significant developments in the 1980s with the implementation of a large scale recirculation system at Loraine gold mine, SA (Rose and Burton, 1992), and a major scheme at the Wearmouth Colliery in the UK (Robinson and Harrison, 1987). Since then a number of fairly large controlled recirculation schemes have been successfully implemented and reported in the literature.

However, although the use of controlled recirculation in mines can be considered as an established technology it has not been that widely applied in terms of its potential to save energy. It is the contention of this paper that there is great potential in this regard and this situation may well change going forward.

With controlled recirculation, the management of the ventilation operation becomes more versatile because there are a wider range of control parameters. These include the selection of fresh downcast and recirculated air flow rates and mixture of surface and underground cooling duties. Indeed, controlled recirculation could be considered as the ultimate ventilation-on-demand system. In general terms, the following considerations apply.

Recirculation generally saves overall energy. Increasing recirculation decreases the surface fan requirements but increases the underground air cooling component. ‘Heat load’ due to auto-compression is proportional to the downcast ventilation air which can be varied within certain ranges. ‘Free energy’ obtained by natural ventilation pressure is reduced as the quantity of recirculated air is increased. Rationalisation of underground air cooling is possible with large centralised air coolers.

**VUMA RECIRCULATION MODELLING**

Over the years there has been a significant amount of work done on modelling the effects of controlled recirculation on dust, gas levels, heat, radiation as well as the transient effects of shutting-down controlled recirculation systems. There are also published reports where underground measurements have verified the models (Robinson, 1972; Burton et al, 1984) and, as a result, the effects of controlled recirculation can be predicted with some confidence.

These models and algorithms have been incorporated into the VUMA network programmes and VUMA is a powerful tool for optimising and designing recirculation systems and maximising energy savings.

VUMA has many standard built-in functions as regards recirculation that include the tracking of radon (and its decay sequences), dust and other gaseous contaminant concentrations throughout the network. This tracking can easily be viewed using colour ranges and limits can be set as design criteria for the programme to highlight branches where concentrations exceed those limits. The surface and underground air coolers, main and recirculation fans, scrubbers (with different component effectiveness) are all catered for using appropriate branch types.

VUMA-network has been adapted to enable accurate real-time predictions of the complete network to be made, based on a calibrated initial network and a limited number of measurements (of flow, pressure, temperature and gas concentration) made at strategic locations. The updated ‘real time’ software has been called VUMA-live. The predicted values are compared in real time with measured values and the network results are thus calibrated on a continuous basis.

Indeed, together, VUMA-live and controlled recirculation could be considered as the platform for the ultimate ventilation-on-demand system.

**ORYX MINE EXAMPLE**

Deep and / or hot mines are of particular interest in this paper which has a focus on energy issues. While there have been a number of effective controlled recirculation applications in deep South African mines, a most outstanding example has been the system at Oryx Mine (Rose and Burton, 1992). This system was commissioned in the mid 1990s and ran effectively, and often better than specification for more than a decade. It was only decommissioned when the mining in the area was depleted.

This gold mine produced 160 kt/month and included a sub-vertical shaft system down to a depth of 2400 m, see Figure 2. The mine was ventilated with 600 kg/s fresh cold downcast air which was supplemented by 300 kg/s reconditioned, recirculated air underground to give 900 kg/s total intake air to the workings. The recirculation fraction was 33% with the recirculated air cooled to a temperature of 18°C in spray chambers located at a depth of 1855 m. It goes without saying that a comprehensive control and safety system was part of the system (but this is not discussed in any detail here). The layout of the spray chambers is shown in Figure 3.

![Figure 2. General schematic of Oryx shaft system](image)
whereas recirculated air will incur full underground refrigeration cost penalties. Within the constraints of the fan / shaft / airway system there is merit in increasing the air flow from surface and decreasing the recirculated flow in winter. For Oryx, it was found that the recirculation air flow rate could be reduced from 300 kg/s to only 150 kg/s in midwinter, with a corresponding increase in the downcast air flow rate from 600 kg/s to 750 kg/s. This resulted in a reduction in underground air cooling and recirculated fan power and pumping power for returning cooling water to surface. Despite an increase in the surface fan power requirement, the overall power requirement would reduce by about 25% in mid-winter.

**TAQUARI MINE EXAMPLE**

Vale’s Taquari potash mine in Brazil has many interesting features including virgin rock temperatures of 50°C at depths of 700 m. The mine has a small shaft system with limited ventilation carrying capacity and no potential of increasing this significantly. The only solution to increasing the primary down-cast flow would have included new shafts and an energy intensive ventilation system. Rather, this limitation was overcome by installing two underground recirculation systems that have allowed increased production, extension of the mine and achieving high overall system energy efficiencies (Bluhm, Funnell and de Oliviera, 2013). The first recirculation system was started in 2003 and the second in 2005 and have operated safely and effectively for 11 years and 9 years respectively.

Continuous Miners and Road Headers are used for production and development, there is very limited diesel equipment and drilling-and-blasting is minimal. The mine does not experience flammable gas, quartzite, silica, radon or derivatives. The mine also had a very energy intensive refrigeration approach and had a critical (and classical) shortage of primary ventilation capacity from surface due to limited shaft capacity. In summary, the mine was a good candidate for large-scale controlled recirculation.

The original refrigeration system used spot refrigeration plants in each of the sections. This approach was highly energy inefficient and had numerous problems including: high compressor power; fouling of heat exchanges; intake air ‘stolen’ to cool condensers and leakage of chilled air from ducts. In the 1990s, the concept of spot refrigeration plant had already been extended to its absolute limits at this mine. Indeed, it had become a very good example of the energy inefficient abuse of spot refrigeration plant (that is similarly misused on many mines to this day).

The system was changed to centralised bulk air coolers served from central surface refrigeration plant and controlled recirculation underground was one of the main features of this approach. The two recirculation systems allow an overall ventilation duty of 560 kg/s which is 87% more than the limited downcast duty of 300 kg/s. The first recirculation takes place near the base of the shafts (central system) and that the second recirculation (north system) is actually a ‘recirculation-within-recirculation’ system, see Figure 6.

The selection of design flow rates for recirculation depended on optimisation of energy loads such as main and recirculation fan power, underground air cooler water and pumping needs and...
surface air cooler capacity. Following VUMA simulation of many scenarios, it was concluded that the primary ventilation system would have a total cumulative capacity (downcast and recirculated) as shown in Table 1.

The refrigeration plant produces cold water which is distributed to the surface air cooler and to each of the underground bulk air coolers. The cold water to the underground air coolers is delivered at a relatively high pressure and, as part of the overall energy management, energy recovery turbines are installed at the recirculation centres, Figure 7. The turbines are of the reverse-pump back-pressure type, are directly coupled to the main return pump-motor sets and recover some 600 kW of power.

The bulk air coolers are all direct-contact multi-stage spray chambers in which cold water is sprayed upwards in a flat V-pattern into the warm air flow. Heat exchange occurs directly across the large surface area of spray. Where the cool air emerges from the chambers, high-performance mist eliminators are fitted to ensure that no water is carried over. The spray chambers are versatile heat exchangers and have multi-stages of sprays allowing a high effectiveness and an important component of the overall energy efficiency.

The central recirculation air cooler is located underground near the shaft station and its construction is based on a substantial concrete lining on the excavated rock surfaces. The north recirculation air cooler is located in the deeper, hotter north section 3 km from the central shafts. The construction of the north air cooler is based on a fabricated steel ‘box’ type structure within the larger excavation, see Figure 8.

The spray chambers act as dust scrubbers and the return air from workings is scrubbed of most of its dust load. This increases the salinity of the water. The salinities are carefully monitored and fresh make-up water is required to control salinity and replace water loss by blow-down and evaporation.

Table 1. Summary of VUMA simulations

<table>
<thead>
<tr>
<th>Air coolers</th>
<th>Total ventilation</th>
<th>Refrigeration process needs</th>
<th>Surface air cooler</th>
<th>Central air cooler</th>
<th>North air cooler</th>
<th>Underground water flow</th>
</tr>
</thead>
<tbody>
<tr>
<td>Surface</td>
<td>300 kg/s</td>
<td>12.1 MW</td>
<td>11.5 MW</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>+ Central</td>
<td>460 kg/s</td>
<td>21.2 MW</td>
<td>11.5 MW</td>
<td>8.0 MW</td>
<td>-</td>
<td>90 L/s</td>
</tr>
<tr>
<td>+ North</td>
<td>560 kg/s</td>
<td>28.5 MW</td>
<td>11.5 MW</td>
<td>8.0 MW</td>
<td>5.7 MW</td>
<td>180 L/s</td>
</tr>
</tbody>
</table>
With recirculation, there are significant energy savings arising from not only the reduced power consumption of the surface fans, and more efficient air circulation with less leakage, but also reduced heat load from auto-compression in downcast shafts and better positional efficiency of underground air coolers.

As noted there are opportunities to adjust recirculation to suit changing circumstances brought about seasonal variations, daily and shift-related variations and movement of mining equipment. In essence, recirculation allows a versatile form of ventilation-on-demand or cooling-on-demand.

Studies conducted a few years ago (Deep Mine research programme South Africa) specifically examined mining below 3000 m. These studies defined ‘global’ recirculation which is usually around the shaft system (as per Oryx example above) and ‘local’ recirculation in smaller districts further into the mine workings. See Figure 9.

These studies required a relatively sophisticated model to be developed to fully appreciate and quantify the interaction between the surface and underground cooling plant, air flows, cooling demands, heat rejection capacity, ventilation leakages, etc. Note the duty of the underground plant is constrained by the heat rejection capacity of the return air to surface which reduces as global recirculation increases. Any reduction in the size of the underground refrigeration plant had to be augmented with cooling from surface which has a completely different and usually greater, energy demand when all the pumping requirements are included.

For this model mine, the total energy input for ventilation and refrigeration (including related pumping and energy recovery turbines) was around 81.3 MW electrical of which 30.3 MW (40%) was for the fans.

Figure 10 indicates the effect of increasing the global recirculation rate up to 60%. Introducing 30% global recirculation reduced the
power consumption by 17.4 MW which is a saving of more than 20%. It is interesting to note that most of these savings (86%) came from fan power with much less of the savings from reduced refrigeration. This will be characteristic of most ultra-deep hot mines. It is interesting to note that the nature of the refrigeration and cooling changed with a 15% shift towards surface refrigeration, as the capacity of underground plant shrank in proportion to the quantity of air returning to surface. Increasing the global recirculation rate reduces the power requirement further but with diminishing returns. In practice however it is unlikely that 60% could be recirculated globally as dust and gas concentrations would have the potential of being high.

Another scenario that was modelled introduced six regions within the deep orebody where local recirculation was employed. The global recirculation was set at 30% and, in addition, the local recirculation was also increased up to 30%. The overall power requirement for this scenario reduced by about 6%. This was not particularly attractive and it was concluded that the more significant energy gains would, not unsurprisingly, relate to the global recirculation. However, it is noted that when intermittent operation is introduced, such as that promoted by ventilation-on-demand and cooling-on-demand, the benefits of local recirculation will be more attractive.

**DEVELOPMENT OF SCRUBBER SYSTEMS**

From the above it is clear that the application of controlled recirculation is hugely attractive from the energy perspective. This then begs the question as to why controlled recirculation is not more widely used. The answer has got to be in the real and perceived safety and health risks. Indeed the use of recirculation in South African gold mines was growing in impetus in the 1990s but was then impeded by the growing awareness of, for example, radon issues.

It is felt that a very significant factor stopping the widespread implementation of recirculation is the concern of contamination from blast fumes, radon, respirable dust particles and diesel particulate emissions. These contaminants typically include but are not necessarily limited to nitrogen monoxide (NO), nitrogen dioxide (NO2), other higher oxides of nitrogen, carbon monoxide (CO), sulphur dioxide (SO2), radon daughter products, diesel particulate matter (DPM), respirable and irrespirable dust particles.

In the late 1950s and early 1960s a number of investigations were carried out to determine a suitable method of eliminating nitrous gases from blasting fumes. These investigations led to the development of fume filters for multi-blast developments. The fume filters consisted of beds of vermiculite through which the blast fumes were passed. The vermiculite bed served as a medium onto which a dilute chemical solution of sodium carbonate, potassium permanganate and water was sprayed. As the fumes passed through, the NO was first oxidised to NO2 and then absorbed in the water solution, together with the other oxides of nitrogen. The overall measured efficiency for total NO2 filtration was up to a maximum of 95%. But these filters had no effect on CO levels and dilution with additional fresh air was essential to achieve acceptable conditions. In 1960, a total of 15 plants were in operation but this equipment required continuous maintenance and these filters were never widely used.

The BBE survey of the overseas mining industry has not revealed a single instance of blast fume filters currently being used. The survey included: Australia, Canada, France, Germany, United Kingdom, South Africa and the United States of America. It would appear that the problem of treating multi-blast fumes is not being seriously addressed anywhere in the world.

BBE have supported the development of air scrubber technology for reconditioning underground mine ventilation and have led a number of research initiatives. It is contended that the majority of contaminants can be effectively removed by a system that incorporates the following common features:

- Large direct contact surface area between contaminated air and a chemical / water solution.
- Addition of a strong oxidising agent to the chemical / water solution.
- Addition of a neutralising agent to the chemical / water solution.

The first feature poses no problem and the required large direct contact surface area can be relatively easily achieved with atomising nozzles in a large reactor-type spray chamber. Atomising spray nozzles would be used to create an extremely intense mist zone that establishes a very large surface area thus providing the interface for the required gas-water reactions. The multitude of small droplets also provides an impaction-removal mechanism for particulate capture. The spray volume and droplet sizes would be controlled to match scrubbing needs and, in general, droplets of less than 50 µm must be produced (Booth-Jones, Annegran and Bluhm, 1984).

The order-of-difficulty for the capture of airborne particles is:

- Radon-daughter-attached dust.
• Diesel particulate matter (DPM).
• Respirable mineral dust.
• Large dust particles outside respirable range.

The spray-filled reactor set up to achieve some capture of the attached-radon-daughter dust will be well suited to, and typically exceed, the other particle capture needs. Having set up the reactor zone for the radon-carrying-dust, the scrubbing of gases becomes an achievable by-product provided the correct oxidising and neutralising chemicals are used and this is where the second and third features are provided.

The contaminants SO2, NO2 and other higher oxides of nitrogen are soluble in water to form sulphuric acid, nitrous acid and nitric acids. The addition of neutralising agents to the resulting chemical-water solution to maintain a high pH and will allow the spray-filled reactor to continue removing these contaminants from the ventilation air. Both sodium hydroxide (caustic soda) and sodium carbonate (soda ash) are considered to be suitable neutralising agents.

The remaining gaseous contaminants include NO and CO. Both of these gases have a low solubility in water and will therefore be more difficult to remove in the spray-filled reactor. However, NO will naturally oxidise to NO2 over time and this reaction will be accelerated in the presence of a strong oxidizing agent. Potassium permanganate, hydrogen peroxide and ozone were considered as possible reagents. Potassium permanganate was used effectively in early scrubbers but pilot plant work has shown that the waste products of the hydrogen peroxide process are easier to handle. The same pilot plant tests even showed that CO can be removed to a certain extent when hydrogen peroxide was used.

Ozone has an oxidation potential higher than both potassium permanganate and hydrogen peroxide and is used extensively in industry and water treatment applications for this reason. Ozone does not need to be transported and stored underground but can be generated as required on site from the oxygen present in ventilation air. Similar capture efficiencies and waste products are expected from both the ozone and hydrogen peroxide oxidation processes. Selection of the most suitable oxidising agent has not been finalised, but ozone appears to be a suitable candidate.

Figure 11 shows the layout of a scrubber that is well suited to integration with a conventional horizontal bulk air cooling spray chamber and the resulting combination will both cool and scrub contaminants from the air.

This scrubber will be capable of removing blast fumes (to some degree), radon daughter particles (to some degree), respirable dust and diesel emissions from the air.

Typical concentrations and expected capture efficiencies are suggested as shown in Table 2.

**CONCLUSION**

It can be concluded as shown in Table 2 that there is a a good probability of success with regard to achieving the above contaminant capture efficiencies. The simulations using ventilation...
Contaminants | Typical value | Capture efficiency
---|---|---
Respirable dust [mg/m³] | 8 | 90%
Non-respirable dust [mg/m³] | 3 | 100%
Respirable carbon particles [mg/m³] | 0.3 | 80%
Nitric oxide (NO) [ppm] | 18 | 70%
Nitrogen dioxide(s) (NO₂) [ppm] | 2 | 85%
Carbon monoxide (CO) [ppm] | 35 | 25%
Sulphur dioxide (SO₂) [ppm] | 2 | 90%
Carbon dioxide (CO₂) [ppm] | 400 | 50%

The recirculation schemes appear acceptable even for the toughest contaminant and the hot, dirty, mechanised, metal mines. Almost all recirculation systems in hot mines will need to include a bulk air cooling system and spray cooling chambers are well suited to this task. Even without modification these air cooling spray chambers will function as low efficiency dust scrubbers and capture the larger non-respirable dust particles. This work contends that by adding an atomised spray section to conventional air cooling spray chambers, relatively high contaminant capture efficiencies can be achieved. The result is a reconditioning system that will unlock the full potential of underground recirculation.

REFERENCES
Pressure piling and the impacts of blast relief to protect primary fans in a highwall longwall operation

D.J. Brake, Mine Ventilation Australia

ABSTRACT
A significant outcome from the Pike River coal mine disaster in New Zealand was reinforcing the importance of being able to re-establish primary ventilation as soon as possible after an underground explosion. This helps reduce the build-up of further explosive gas mixtures in the workings which potentially may have much more devastating impacts than the initial explosion. Two key factors that Australian regulators have subsequently identified as not having been sufficiently understood or considered in the past are the phenomena of ‘pressure piling’ and the design of blast relief at primary fans. This paper, which includes a case study, examines just what is meant by the term ‘pressure piling’ and how it impacts on pressure spikes at surface fan installations in the event of an underground explosion. It describes modelling which found that in the case of an highwall-style retreating longwall operation where the portal for the surface conveyor drift also contains the surface primary exhaust fan for that longwall panel, provision of sufficient pressure relief (and hence volume flow relief) at the portal to protect the fans may result in such high velocity pressures within the conveyor drift, as the explosive overpressures escape, that severe damage to the conveyor, and possibly plugging of the drift by conveyor debris, could result. In such a case, even if the primary fan is protected by the design of the pressure relief, the restart of the fan may not restore the primary ventilation circuit due to the total or partial blockage of the main exhaust circuit upwind of the fans. Potential implications and solutions to this problem are discussed.

INTRODUCTION
The report from the Royal Commission on the Pike River Coal Mine Tragedy (Royal Commission on the Pike River Coal Mine Tragedy, 2012) identified a number of major problems at that operation, some of which relate to the way the mine was ventilated.

To further understand the Australian mining Inspectorate’s concerns, the following (selective) extracts should be noted from the Pike River Commission’s report:

“On Friday 19 November 2010, at 3:45 pm, the mine exploded. Twenty-nine men underground died immediately, or shortly afterwards, from the blast or from the toxic atmosphere ...

“Over the next nine days the mine exploded three more times before it was sealed ...

“The commission is satisfied that the immediate cause of the first explosion was the ignition of a substantial volume of methane gas ...

“The area most likely to contain a large volume of methane was a void (goaf) formed during mining of the first coal extraction panel in the mine. A roof fall in the goaf could have expelled sufficient methane into the mine roadways to fuel a major explosion. It is also possible that methane which had accumulated in the working areas of the mine fuelled the explosion, or at least contributed to it.

“The original mine plan specified two main fans located on the mountainside next to a ventilation shaft. Two planning changes were made. Pike decided to relocate the fans underground in stone at the bottom of a ventilation shaft. Placing a main fan underground in a gassy coal mine was a world first. The decision was neither adequately risk assessed nor did it receive adequate board consideration. A ventilation consultant and some Pike staff voiced opposition, but the decision was not reviewed. Putting the fan underground was a major error.

“The placement of the main fan underground and the damage caused to the back-up fan on the surface meant that the mine could not be reventilated quickly ...

“The expert panel concluded that the size and duration of the explosion indicated it was fuelled by a large volume of methane, perhaps up to 2000 m³. Methane accumulated in the hydro goaf following mining was estimated at up to 5000 m³. Another roof fall like that which occurred on 30 October 2010 would have caused a large pressure wave bearing a substantial volume of methane.

“The pressure wave would have flowed down the hydro panel roadways and entered the main mine roadways, with the potential to flow inbye, particularly if a temporary stopping failed and allowed the wave to enter the main intake roadway. Methane carried along the roadways by the pressure wave would be diluted by air into the explosive range. Under “Proposals for Reform”, the Enquiry made the following comments regarding ventilation (italics author’s comments):

Ventilation and gas monitoring

“Placing main ventilation fans underground in coal mines should be specifically prohibited. It is unlikely that a mining company would do so in the future, given the consequences at Pike River, but the matter should be put beyond all doubt. Main fans should be required to be protected against explosion and other hazards, in accordance with appropriate international standards.

“In addition to requiring a ventilation officer, standards for ventilation control devices, such as stoppings that control airflow, need to be specified.

“Minimum requirements for gas monitoring systems are needed so that the mine’s atmosphere can be continually and comprehensively analysed.”

In Australia, mining is regulated at the State not Federal level. To
avoid the likelihood and / or reduce the consequences of a similar event occurring in Australian coal mines, state government regulators subsequently asked mine operators to review their ventilation systems and primary fan installations to ensure that if an underground explosion were to occur, the primary ventilation system could be restarted in the shortest practical period of time.

In this regard, two key factors that the regulators asked operators to consider are the impacts of:

- “Pressure piling”, and
- Blast relief at primary fans to avoid any serious damage to the fans, i.e. damage that would prevent the fan being restarted quickly after a blast and the primary ventilation exhaust shaft returning to operational status.

These concerns were further expressed in a presentation by the Queensland government agency SIMTARS (Davies and Smith, 2013) in which the following general comments about blast protection in Queensland coal mines were made (italics are additional comments by this author):

“It appears the (surface fan) enclosures were designed based on an explosion occurring in the enclosure itself (i.e. not the potential for a much larger blast upwind of the fan within the mine)

“Explosion vents amount at best to 7 m² in total cross sectional area and are usually placed strategically around the fan housing

“Ventilation shafts are typically around 20 to 25 m² in cross sectional area

“No standards exist in Australia”

“Pressure piling” is discussed more fully later in this paper but effectively it refers to the following situation:

- An explosive gas mixture is simultaneously present in two (or more) interconnected volumes,
- An explosion is initiated in one volume (the first volume),
- For reasons discussed shortly, the peak explosive pressure reached in the other (second) volume is higher than the peak reached in the first volume and is higher than the value that would otherwise be predicted for that volume from thermo-dynamic analysis.

It can be seen from Figure 1 that the explosion panels for the surface backup fan installation at Pike River were small but probably typical for current practice in Australia.

Regarding the underground main primary fan (of which no post-blast photos are available, as the mine has not to date, January 2014, been re-entered), the report also states:

“Pike did not install explosion proofing for the main underground fan, did not site the fans in rock and the blast panels on the surface fan proved inadequate during the explosion.”

In summary, the concern of the regulators was that:

- An initial methane explosion could so damage the surface fans that primary ventilation cannot be quickly re-established. However, at least some persons underground could possibly still be alive.
- The failure to be able to re-establish the primary ventilation leads to a more general methane accumulation issue underground, which prevents any search and rescue operations and then triggers a far more devastating methane or coal dust explosion which not only destroys much of the rest of the mine, but also proves fatal to any persons still alive.

- “Pressure piling” is one factor that can contribute to explosion relief on the primary fans being undersized, or in the wrong location, or of the wrong type.

**OBJECTIVES**

In this case study, four steps were identified as being required to address these concerns:

- Understand the theory of dynamic (time-dependent) overpressure propagation with respect to the factors in an underground mine (e.g. geometry of the mine and openings, etc.).
- Understand the range of “credible” explosions that could occur in terms of near-instantaneous overpressures generated and locations.
- Model the overpressures produced in the credible worst-case explosion scenario, including failure of ventilation controls.
- Understand the potential solutions at the surface fan to protect it from these overpressures or to allow it to be restarted quickly.

The particular mine which was the subject of this study is a high-wall operation using the longwall retreat technique. Each longwall block is effectively accessed via new mine entries driven off the highwall. The conveyor which removes the coal from the face exits the underground via one of these highwall entries (via a short concrete culvert). The same conveyor drift is also one of the mine exhausts, so has a primary surface fan off at 45° to the side of the culvert with a coffin seal at the end of the culvert to reduce short-circuiting of intake air directly into the fans. There were a total of 5 surface fan installations at this mine: four off highwall portals and one off a surface exhaust shaft, see Figure 4.

It was decided to model potential longwall face / goaf explosions with the face position at 50%, 75%, and 100% extraction. The
100% extraction location meant the face was closest to the high-wall and therefore to the primary fans (about 150 m separation). An additional model for 0% extraction (i.e. at longwall start-up) was also examined.

Note that this paper reports only the modelling aspect of the work completed; other important aspects of this work such as a review of ventilation controls and a review of the fan / conveyor drift geometry / layout are not reported here.

DEFINITIONS AND GLOSSARY

There is frequently inconsistent use of important explosive and pressure piling terms in the various literature. For the purposes of this paper, the following definitions, largely taken from Bjerketvedt, Roar Bakke and van Wingerde (1997) and Zipf, Sapko and Brune (2007) are used:

**Table 1. Glossary of terms**

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Blast wave</strong></td>
<td>The air wave physically set in motion by an explosion</td>
</tr>
<tr>
<td><strong>Burning rate</strong></td>
<td>The amount of fuel consumed by the combustion process per second</td>
</tr>
<tr>
<td><strong>Flame speed</strong></td>
<td>The absolute velocity of a flame front relative to a stationary observer</td>
</tr>
<tr>
<td><strong>Burning velocity</strong></td>
<td>The relative velocity of the flame front with respect to the velocity of the unburnt fuel immediately in front of the flame front</td>
</tr>
<tr>
<td><strong>CJ pressure</strong></td>
<td>The Chapman-Jouguet detonation pressure</td>
</tr>
<tr>
<td><strong>CV</strong></td>
<td>Constant volume (an explosion which occurs within a fixed volume vessel)</td>
</tr>
<tr>
<td><strong>CV pressure</strong></td>
<td>The “ending” pressure produced when an explosion occurs in a fixed volume</td>
</tr>
<tr>
<td><strong>Deflagration</strong></td>
<td>A rapid combustion (explosion) with burning velocity (note: not flame speed) less than the speed of sound (1193 kph or 331 m/s)</td>
</tr>
<tr>
<td><strong>Detonation</strong></td>
<td>A rapid combustion (explosion) with burning velocity (note: not flame speed) greater than the speed of sound (1193 kph or 331 m/s)</td>
</tr>
<tr>
<td><strong>Dynamic pressure</strong></td>
<td>The pressure of a moving fluid (e.g. air) if were to be stopped against a wall</td>
</tr>
<tr>
<td><strong>Overpressure</strong></td>
<td>The peak value of pressure (e.g. the pressure wave) above the normal value of pressure at that location, i.e. the increase in pressure rather than the absolute pressure. For example, an over-pressure of 800 kPa from a pre-blast absolute pressure of 100 kPa (about sea level pressure) would create an absolute blast peak pressure of 900 kPa.</td>
</tr>
</tbody>
</table>

**Reflected wave**

When a shock wave strikes a solid surface, part of the energy of the shock wave induces a reflected wave, which can result in very high pressures at that location, but with lesser energy and pressures in the continuing shock wave.

**Shock wave**

A fully developed pressure wave of large amplitude, across which the density, pressure and particle velocity change dramatically.

**Stoichiometric composition/mixture**

The ratio of fuel and air which results in no excess fuel or excess air being left at the end of the reaction. For methane at standard temperature and pressure, this is 9.5% CH\textsubscript{4} by volume, or 67.8 grams CH\textsubscript{4} per m\textsuperscript{3} mixture. In most cases, the peak pressures from an explosion are obtained when the mixture starts at the stoichiometric value.

FUNDAMENTAL THEORY AND REVIEW OF CURRENT WORK

**Consequences of an explosive gas release or presence**

When there is a flammable gas release or presence, the consequences can be (Bjerketvedt, Roar Bakke and van Wingerde, 1997):

- Nothing, if there is no ignition source and the gas dilutes away.
- Fire, if the gas immediately catches fire on exposure to the air. In this case, the fuel and the oxygen are mixed during the combustion process.
- Explosion, if the gas / air mixture builds up to a flammable cloud, and is then ignited. In this case, the fuel and oxygen are mixed before the combustion process starts.

For a stoichiometric mixture of methane and air at 25°C and 101 kPa, the increase in pressure at constant volume is 8.94 times starting pressure and the increase in volume at constant pressure is 7.72 times starting volume (Zipf, Sapko and Brune, 2007). See Figure 2.

**Potential pressures from a combination of methane and coal dust explosions in closed vessels**

Unlike methane, coal dust does not have a similar “rich” or upper concentration limit beyond which the mixture becomes non-explosive. Coal dust reaches a maximum explosive pressure at concentrations of about 200-300 g/m\textsuperscript{3} (Figure 3). The energy release from a coal dust explosion is only limited by the available oxygen in the reaction vessel or the sealed area of a coal mine, if enough dust is available.

**Potential pressures from a methane/coal dust explosion in tunnels**

Zipf, Sapko and Brune (2007) describe the potential development of an explosion in a tunnel initially completely filled with an explosive mixture, from a slow deflagration to a rapid deflagration to a detonation including the potential for very damaging reflected
waves. The four possible stages are: “slow deflagration, fast
deflagration, detonation and reflection of a detonation wave from
head-on impact with the closed vessel”.

“Above each stage of combustion is a pressure profile along the
tunnel. Upon ignition, the initial laminar flame speed is only
3 m/s; however, a slow deflagration accelerates, and the turbulent
flame speed might increase to about 300 m/s (the “run up”). The
pressure in the burned gas behind the flame front increases to the
908 kPa CV explosion pressure. The combustion front acts as a
piston, compressing the unburned gas in front of it. The leading
dge of this acoustic wave propagates at the local sound speed of
about 341 m/s. In between this wave front and the name front, the
unburned gas acquires velocity and the static pressure inside this
region will increase. This pressure increase ahead of the flame
front is termed ‘pressure piling’. (italics author’s comment).

Zipf, Sapko and Brune (2007) goes on to note that the peak
pressures due to pressure piling will be higher than the CV value,
that peak pressures in a detonation can be up to about 1.76 MPa
(the CJ (Chapman-Jouguet) detonation pressure) and reflected
over-pressures could be 4.1 MPa or higher. However, in all cases,
these transient pressures will quickly equilibrate to the
908 kPa CV explosion pressure as before”.

The important point for explosion modelling in this study is that
the explosion in the portion of the mine that initially contains the
explosive gas mixture could reach catastrophically destructive
transient pressures, but as soon as the blast has run out of fuel. The
pressures within this zone will revert to 908 kPa overpressure.

Pressure piling: what is it?

There has been some use of the term “pressure piling” already in
the above discussion. However, this is not the only (or even most
common) way in which the term is used. There are at least three
contexts in which the term “pressure piling” is used.

“Pressure piling”: Classical definition

Di Benedetto, Salzano, and Russo (2005) define pressure piling in
this way: “The phenomenon of explosion of flammable hydro-
carbon-air mixtures in two or more interconnected compartments
is commonly defined as ‘pressure piling’ and it is characterised by
a pressure peak higher than the thermodynamic value”.

This is a similar definition to that used in AS2380.2 (1991) which
defines pressure piling as “a condition resulting from ignition of
precompressed gases in compartments or sub-divisions other than
those in which ignition was initiated”.

There are major differences between the classical pressure piling
situation in closed vessels and the sense in which the term has
more recently been used in coal mine airways:

- At the time of ignition in a coal mine, only the “initial vessel”
  (e.g. an open longwall goaf or the longwall face itself) has
  explosive gas in it. All other “vessels” have non-explosive
  mixtures in them (although the initial explosion could partially
  push an explosive mixture into some of them, although this is
  not a major mechanism given the speed of the flame front)

- In a conventional pressure piling situation, there are no
  additional “vents” except for the two interconnected and gas-
  filled vessels: the first vessel can only vent into the second, and
  the second vessel must vent back into the first (unless the
  vessels themselves fail or the vessels are vented to elsewhere).
  In an operating mine, there are many points at which the
  “vessels” (airways) can vent into other “vessels” (airways) and
  also more than one vent to atmosphere via the shafts or portals.

- In a coal mine, there is the risk of a gas explosion proceeding to
  a coal dust explosion, a different scenario to pressure piling
  which is based on gas explosions only.

- The term pressure piling is not used, in this sense, with respect
to detonation-type (CJ) explosions only constant volume (CV)
explosions.

“Pressure piling” in a long duct

As used by Zipf, Sapko and Brune (2007), the term “pressure
piling” can indicate the increase in pressure as an explosive
mixture in (say) a tunnel is ignited at one end of the tunnel, and
the flame front accelerates along the tunnel pre-compressing the
explosive gas in front of it and hence increasing the pressures
from the initial starting values at the site of the ignition. There is
no need for “two vessels” in this sense of the term, merely one
long vessel.

“Pressure piling” in non-explosive venting

The term “pressure piling” has also been used to describe the
short-term pressure increase in a duct system well away from an actual explosion and in a region where there is no explosive gas. In this situation, the pressure piling is not due to an explosion at the pressure piling point itself but is due to the increase in pressure away from the explosion site where the high pressures in the shock wave meet obstructions in the escape paths resulting in a high-pressure reflected shock wave and pressure concentration. It is important to remember that each time a shock wave produces a reflected wave, the energy of the original shock wave is divided into the continuing shock wave and the reflected shock wave, so that the energy of the continuing shock wave reduces.

Overpressures and pressure piling

Whether the overpressure produced at the surface fan location from the blast is due to the dissipation through the workings of the explosion overpressure at the original blast site, or due to the pressure concentration due to shock wave reflections in the airway at the fan location (or a combination of both), if this overpressure cannot be relieved ahead of the fans (e.g. by blast doors). Then the overpressure will destroy the fan elbow as well as catastrophically damaging the main fans and rendering the primary ventilation system inoperable for some time. If the blast panels “do their job” then the elbow and fan will survive and the blast panels should be able to be easily replaced allowing the fan to restart very quickly.

Implications for explosion modelling in coal mines to protect primary fans

With regard to understanding overpressures near the primary fans, the objective of this exercise is not to try to estimate the damage to the area of the mine originally filled with explosive gas (e.g. long-wall face) or even to other production-related underground infrastructure (conveyors, regulators, overcasts, etc). Rather the issue is to estimate the potential overpressure spike (“pile”) at the exhaust shaft collar or shaft portal where surface primary ventilation fans are located and to assess what mitigation strategies are required to ensure the primary fans can re-establish the primary ventilation immediately (or in a very short time) after a blast, to prevent further more damaging blasts and / or to facilitate self-rescue from those who survived the blast and / or to facilitate safe aided rescue by mine rescue teams.

BLAST RELIEF TO PROTECT PRIMARY FANS

Unfortunately, the research and standards for pressure piling and pressure relief applicable for industrial facilities are limited to situations where the length of the enclosure is not more than about 20 times the diameter of the enclosure. This makes most of this theory of little value for mines at least within the zone originally filled with explosive gas, as this can be subject to detonation as well as pressure piling.

The MSHA (USA) requirements for blast relief on primary fans are set out in MSHA CFR 75.310 and further discussed by Conn and Verakis (1993).

They note that the fan can sustain explosion damage by:

• Explosion damage from an explosion wind.
• Debris propelled by the wind.
• A shock / detonation front.

The important thing here is that it is not only the “pressure piling” effect that can damage the fan; in particular flying debris is an important potential source of damage.

EXPLOSION MODELLING SOFTWARE

The two gas explosion models used by NIOSH in the Zipf, Sapko and Brune (2007) report were AutoReaGas, available from Century Dynamics in the United Kingdom and FLACS (Flame Acceleration Simulator) available from GexCon of the Christian Michelson Research Institute in Norway (of which Bjerketvedt, the lead author of the Gas Explosion Handbook Bjerketvedt, Roar Bakke, and van Wingerden, 1997 was an employee). AutoReaGas and FLACS are specialised computational fluid dynamics (CFD) models for numerically solving the partial differential equations governing a gas explosion. These models are used extensively in the oil, gas, and chemical industries to assess risks, consequences, and mitigation measures for various gas explosion scenarios. In particular, they have seen application to offshore oil and gas production facilities since the Piper-Alpha oil platform disaster that occurred in the North Sea, UK in 1988. A few European research groups have made attempts to use these models to study gas explosions in mines, but to date such work is very limited.

Zipf, Sapko and Brune (2007) performed comprehensive modelling but this was to examine local pressures on seals using relatively simple geometries and was not to model the “dissipation” of the high pressures through to surface, or the “domino effect” as ventilation controls such as regulators and overcasts fall.

The complexity of most underground mines, along with their ever-changing geometry, means that the use of such special purpose CFD programmes for explosion modelling in operating mines is not practical. The following case study uses an explosion modelling feature within the ventilation modelling software Ventsim™, key details of which are described below.

CASE STUDY

Credible scenario for explosion

There are no doubt a large number of potentially credible explosion scenarios for any coal mine. However, the mine which is the subject of this case study adopted the following credible scenario:

• Gas explosion on the longwall face and / or in an open goaf behind the longwall face.

• At the time of ignition, there are no other explosive gas mixtures in the mine. This would be typical of a well-operated mine before any secondary explosions occur.

• Whether the explosion is a deflagration or a detonation is not relevant in that this only determines the local transient peak pressures within the zone of explosive gases (or somewhat further along the flame path, i.e. the longwall face). Immediately after the explosion, gas pressures return to the CV values and it is this value that must then be dissipated through the rest of the mine, and eventually to the surface.

• In addition, since a detonation does not disturb the air mass in front of it and moves at 1800 m/s (Mach 5.3) (Zipf, Sapko and
Brune, 2007), if explosives gases were to extend through the entire mine to the surface and this mixture was to detonate, then blast relief panels would be useless as there is no local pressure increase at the fans to activate the blast relief panels, until the detonation itself arrives with its catastrophically destructive over-pressures.

• The gas explosion occurs on the longwall face, so that the over-pressure is produced at this location. Modelling can therefore be considered to assume an instantaneous high pressure and volume on the longwall face, attenuating as this overpressure moves rapidly through to mine to all lower-pressure regions, causing seals to fall, and the pressure-relief path through the mine to then change (continuously as seals continue to fail), etc.

Assumptions of the initial gas volume and concentration and initial pressure / or explosion modelling

Given the above and the fact that it is overpressures at surface ventilation connections that are the object of this case study, the following key assumptions were adopted:

• The Pike River Enquiry notes that the initial explosion was perhaps up to 2000 m³ of methane. This was taken as meaning pure CH₄ volume plus associated diluting air. In the case study, the longwall face was about 330 m long and about 3.5 m high, and assuming the depth available for an explosive gas mixture is 10 m, the total volume of explosive gas mixture along the face would be in the vicinity of 11550 m³. If the mixture was 10% CH₄, this would be a volume of CH₄ of about 1155 m³ or a little over 50% of the estimated Pike River event.

• Peak pressures after the blast were set at the CV value of 8.96 times the starting pressure or 900 kPa absolute (assuming the underground workings are approximately at sea level). This pressure will exist along the longwall face and then be dissipated throughout the mine blowing out ventilation controls as the over-pressure expands outwards from the original blast site.

• This value of 900 kPa is high compared to the known or estimated peak values from other historical coalmine explosions, but is not incredible (Zipf, Sapko and Brune, 2007).

Modelling technique used for the case study

As noted above, there has been no “whole of mine” explosion modelling package developed for any type of complex underground facility such as a coalmine, to date. Any explosion modelling has used CFD techniques and been strictly limited in application. For this exercise, a new module within Ventsim (“ExplosionSim”) has been used for the modelling. However, it is important to understand that some simplifications have been made in the modelling technique. Ventsim has taken the following approach in the explosion module:

• Ventsim injects air at the user-defined fixed (explosion) overpressure into the zone of airways in which the user has designated the explosive mixture will be present. In the case of the case study, this is the entire length of the longwall face.

• Ventsim performs a simulation to determine the effective resistance from the explosion site through to surface.

• The programme then loops through every 0/1000th second, feeding a portion of the overpressure volume (0/1000) into the model through pathways leading from the blast zone. The remaining volume allows the next overpressure to be calculated.

• During each loop, the overpressure expansion radiates in all available directions such that pressure can be calculated at any location based on the ratio of expanded gas pressure to the original overpressure at the time the pressure wave reaches that location.

• A ventilation control “fails” when the overpressure on the control exceeds the user-nominated failure pressure. e.g. a “35 kpa” seal will fail at 35 kPa overpressure on either side of the control, assuming the user sets up the control to fail at this overpressure. Requiring the control to fail when the over-pressure on either side of the control reaches the failure pressure is more realistic and conservative than attempting to use a “modelled” differential pressure across each control, given the blast wave moves so quickly.

• Effectively, this means new resistances of the blast dissipation through to surface can then be recalculated minus the failed control.

• The blast dissipation routine is then restarted assuming the control will no longer be in place and the algorithm repeatedly progressively expands the blast overpressure volume further until no further controls will fail.

• The total time of blast dissipation occurs when the remaining blast overpressure falls below 10 Pa (0.01 kPa residual overpressure). This is reported in the message box.

As noted earlier, peak pressures within the initial explosive gas-filled zone can reach as high as 1.8 to 4.5 Mpa. However, even in the event of a detonation, the peak pressure within (or outside) the initial zone filled with explosive gas, a very short time after the flammable gas is consumed, will not exceed 9 times the initial local underground absolute pressure, or about 900 kPa. “Pressure piling” due to the obstructions in the airways including at the fan location can still occur, but it will only “pile” on top of the already dissipated pressures reaching the collar. Ventsim does not attempt to calculate the potential local pressure spikes due to reflected shock waves.

Results of modelling at the case study

As with any modelling exercise, it is important to understand the objectives of the model as this governs whether it is “fit for purpose”. The two critical issues for the case study mine were to understand:

• Which primary fans could “fail” in the event of a gas explosion on the longwall face, and

• For those fans, what measures could be taken to protect the fans or to ensure they can be restarted as soon as possible.

In addition to the potential for the ventilation controls in the mine to fail and the primary fans to fail if there is insufficient blast relief, a highwall operation such as the case study also has the potential for any structures projecting from the portals to fail. For example, at the case study mine, a concrete culvert (containing the conveyor and associated coffin seal) projects from the mine entry and also provides a suitable mating point for the main fans. Predicted peak overpressures at the five primary fan locations are summarised in Table 2. To estimate potential peak overpressure at
each fan location, column B in Table 2 assumes that the fan portals have not failed (coffin seals are intact) and blast relief doors have not operated, i.e. while other seals in the mine fail are allowed to fail at their nominal pressure rating (e.g., 14, 35 and 140 kPa), the fan portal “seals” are assumed to not fail in this modelling. This is to estimate the potential peak overpressure value at the fan location without pressure relief.

The number and location of seals within the mine workings that fail is complex, determined by how fast the original blast over-pressure can dissipate through to surface, and the amount of “expanding volume” available for pressure relief as the various seals fail, as well as by the strength of individual seals.

In the case of portal fan MG8, peak wind speeds (and hence velocity pressures) will occur if the structure or blast relief doors at the fan do “blow out” (fail). To estimate the potential upper severity of the wind blast at the fan portal, modelling of values in column B was performed where it is assumed the portal structure has “blown out”, i.e. failed. This is quite different to the situation in column A where to estimate the potential peak overpressure at the fans, it is assumed the structure around the fans has not failed. It is critical to note that these wind speeds assume the entire conveyor drift cross-sectional area is available for flow, which is not the case. Likely wind speeds are possibly up to twice those modelled.

For comparative purposes, the “equivalent” cyclone / hurricane wind speed is also shown in the table. Wind speeds cannot be directly compared to “cyclone ratings” as the air density in a mine explosion overpressure situation is much higher (and therefore more damaging) than air density in a cyclone. However, comparisons between velocity pressures of cyclones and explosive overpressure releases should be more comparable in terms of damage.

**Potential controls**

It is important to note that the risk control strategy recommended for this operation, and required by the regulators, assumes the primary ventilation circuit can be re-established very quickly after the primary explosion, so that any secondary explosions can be avoided, or the fuel content of any secondary explosion is kept low by dilution with fresh air so that they do not (as in the case of Pike River) create more damaging secondary explosions than the original primary explosion.

One complication that makes this highwall operation unusual is that the exhaust (portal) in which the fan is located is also the conveyor drift, and any major explosion underground will produce not only high overpressures in this drift, but (unlike most other coal mine exhausts (shafts) which are “empty”) is also likely to result in damage or destruction of the conveyor and / or “piling up” of the destroyed conveyor belt and its steelwork potentially near the portal (particularly in the scenario where the longwall is fully retreated). This drift with all its infrastructure would also provide an ample supply of flying debris some of which will be at very high speeds.

The issues here are that the damaged conveyor partially blocking the drift could produce even higher overpressures at the portal than predicted (by “bottling up” the pressures) and also provide ample projectiles that could easily pass through the fan and destroy it.

Possible controls to help ameliorate this issue included:

- Design pressure relief blow-out panels at the portal to keep overpressures and wind speeds “as low as reasonably achievable” (ALARA) and comply with Mine Safety and Health Administration’s Code of Federal Regulations (MHSA CFR) 75.310.
- Being able to safely close off the conveyor portal irrespective of its condition.
- Using a combination of other fans in the mine to provide a temporary ventilation circuit that ventilates as much of the mine as possible.

Each of these strategies requires a plan to be developed and risk assessed to ensure the work can be carried out even with a potential explosive mixture of unknown volume present underground.

Note that even if the pressure relief panels do operate, the portal will still experience high overpressures due to the finite rate at which the blow-panels can relieve the oncoming pressure wave from the explosion. The purpose of the blow-out panels is not to eliminate the potential for overpressures, but to:

- Reduce the peak overpressures, and
- Where the blow-out panels are located in the direct line of the blast (the normal situation), to allow flying debris to escape without being forced into the fan inlet and fan impeller.

Assuming the fan is undamaged in the explosion or can be rapidly recommissioned, provision must be made to allow the blast relief panels to be reinstalled assuming an explosive mixture remains underground without creating excessive risk for the repair crew.

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**Table 2. Predicted peak pressures at all five surface fan locations and peak wind speeds MG8. Note that there are significant changes to the primary ventilation circuit (including primary fan relocations) between LW8 at 0% and the other three scenarios.**

<table>
<thead>
<tr>
<th>LW8 % extraction</th>
<th>Blast dissipates (seconds)</th>
<th>Number of failed seals</th>
<th>Peak overpressure at each fan location, kPa</th>
<th>Peak velocity pressure, Pa, MG8 fan and equivalent category cyclone</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>A: Assumes blast relief does not operate</td>
<td></td>
<td>B: Assumes blast relief has operated</td>
<td></td>
</tr>
<tr>
<td>LW8 extract</td>
<td>Blast dissipates</td>
<td>Peak overpressure</td>
<td>Surf fan</td>
<td></td>
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<tr>
<td></td>
<td></td>
<td>at each fan location</td>
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<tr>
<td></td>
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<td>MG8 fan</td>
<td>MG10 fan</td>
<td>MG11 fan</td>
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<td>MG3 fan</td>
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<td></td>
<td></td>
</tr>
<tr>
<td>0%</td>
<td>20</td>
<td>24</td>
<td>7.4</td>
<td>n/a</td>
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<td></td>
<td></td>
<td></td>
<td>n/a</td>
<td>6.2</td>
</tr>
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<td></td>
<td></td>
<td></td>
<td></td>
<td>6.9</td>
</tr>
<tr>
<td>50%</td>
<td>11</td>
<td>24</td>
<td>13.7</td>
<td>0.0</td>
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<td></td>
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<td></td>
<td>n/a</td>
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<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>9400 Pa (&gt; &gt; category 5)</td>
</tr>
<tr>
<td>75%</td>
<td>5</td>
<td>38</td>
<td>23.5</td>
<td>0.0</td>
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<td></td>
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<td>n/a</td>
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<td></td>
<td></td>
<td>12500 Pa (&gt; &gt; category 5)</td>
</tr>
<tr>
<td>100%</td>
<td>6</td>
<td>25</td>
<td>52.5</td>
<td>0.0</td>
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<td>n/a</td>
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<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>34000 Pa (&gt; &gt; category 5)</td>
</tr>
</tbody>
</table>
SUMMARY AND CONCLUSIONS

The explosion scenario examined in this case study is for a blast igniting a flammable gas mixture filling the entire longwall face, but not elsewhere in the mine. It assumes no secondary gas explosions and no triggered coal dust explosion.

Peak pressures reached when a volume of gas explodes in a coalmine can reach up to 4.5 MPa due to a combination of: the explosion pressure itself, the “enhanced” effects of pressure piling, potential detonation (for supersonic explosions) and shock wave reflections.

These “enhanced” pressures, except for reflected shock waves, only occur within the volume originally containing the explosive mixture.

These “enhanced” pressures are transient, i.e. as soon as the explosion is over, the peak pressures within the volume originally containing the flammable mixture reduce to about 900 kPa.

Extreme destruction can therefore occur within the volume originally containing the explosive mixture.

Outside of the volume containing the explosive mixture, the peak non-transient pressure reached will be 900 kPa and will reduce as the now exploded gases expand into the remainder of the mine workings, including via failed ventilation controls.

A primary ventilation fan can sustain explosion damage by:

- Explosion damage from the airblast (wind) produced by the blast.
- Debris propelled by (carried along with) the wind, or
- A shock/detonation front.

Figure 4. Case study mine plan of operations

Figure 5. LW8 at 100% (scenario D in Table 2): 25 seals fail (red). Numbers are peak explosion overpressure kPa
The impact of reflected shock waves is to increase the pressure at the location of the obstruction, but reduce the energy and pressure of the continuing wave.

Only the potential for airblast overpressures and debris damage have been considered for this case study modelling.

The number and location of seals that fail is complex, determined by how fast the original blast over pressure can dissipate through to surface, and amount of “expanding volume” available for pressure relief as the various seals fail, as well as by strength of individual seals.

Table 2 provides estimated peak (worst case) overpressures in the case study operation and Figure 5 provides the expected ventilation control failure.

Table 2 also provides estimated peak velocity pressures at MG8 assuming an “open” drift (without any deduction of cross-sectional area for conveyor). This is an indication of the potential destructive force of the wind on the conveyor structure in this region. A comparison with cyclone “ratings” is also provided.

Potential solutions to this issue at this mine include having a contingency plan to recreate a viable primary ventilation circuit without the highwall fans on this maingate, and, in the medium term, to not put highwall fans servicing future longwall panels in the conveyor drift for that panel.

REFERENCES


CFD simulation of methane roof layering in underground coal mines

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ABSTRACT

This paper presents 3D numerical simulations of methane roof layering in a horizontal tail-gate of a retreating longwall mine using CFD code ANSYS Fluent 12.0. This study is aimed at investigating the effect of air velocity on methane layering in underground coal mines. Simulations were performed at constant rate of methane emission and varied air velocities using three commonly used turbulence models, viz. Spallart-Allmaras, K-ε and K-ω (SST). The study revealed that methane layering is greatly influenced by air velocity. Layering of significant length and thickness occurred in the case of poor ventilation; however, a reduction in the layering length with progressive increase in the air velocity was noticed. The lengths of methane layer obtained through simulations were compared with those estimated from the experimental data plot of Bakke and Leach (1960) corresponding to the layering numbers at different air velocities, and were found to be reasonably in good agreement.

INTRODUCTION

Methane (CH₄), which naturally occurs in coal seams, is released in significant quantities into underground mine workings when coal seams are disturbed by mining activities. Methane is a highly inflammable gas and accumulation of the same in underground coal mines always poses threat of explosion due to its ignition. In recent years, increase in methane gas emission into underground coal mines due to increased coal production has become a big issue and increased the risk of explosion hazards in mines because of methane layering problems. Being a lighter gas with specific gravity of 0.554, methane has the propensity of streaming in the form of a layer near the roof level of underground coal mines (Creedy and Phillips, 1997; Torano et al., 2009). In underground coal mines, the layering is more pronounced where the rate of methane emission is quite high and ventilation is poor to cause dispersion of the methane. The methane layer in a mine airway usually travels up the dip along with the air current and the movement mainly depends on the velocity of air current and roughness of the surface. According to Raine (1960), only the methane accumulation along the roof whose length is greater than the width of the airway in which it is formed should be considered as a ‘methane layer’. The length of methane layer is defined as the distance from the source to the point where the mean concentration of methane in the layer is 5% (Mishra, 1986).

In the early sixties, the analytical and experimental works carried out at the Safety in Mines Research Establishment, England led to quantification of the effect of important parameters, viz. velocity of air, rate of gas emission, width and inclination of airway, relative density of the air and gas, roughness of the roof on methane layering (Bakke and Leach, 1962). Till today, the pioneering work carried out by Bakke and Leach during the 1960s in the UK on methane layering is being relied on and referred by many researchers. Bakke and Leach (1960, 1962) observed that the layering phenomenon depends on the dimension less the combination of independent variables and proposed the following relationship for calculation of a dimensionless methane layering number which indicates the stability as well as the length of methane layers (McPherson, 1993):

$$L = \frac{v}{g \Delta \rho W} \left\{ \frac{Q}{\rho W} \right\}^{1/3} \tag{1}$$

where $v$ is the average air velocity (m/s), $g$ is the acceleration due to gravity (9.81 m/s²), $\Delta \rho$ is the difference in relative densities of the two gases ($1 - 0.554 = 0.446$ for air and methane), $Q$ is the rate of methane emission into the airway (m³/s) and $W$ is the width of the airway (m). In level airways, a minimum layering number of 5 is desirable for causing a sharp decrease in the length of the layer (Mishra, 1986; McPherson, 1993). However, in inclined airways, the desirable layering number depends on the direction of airflow. In case of ascensional ventilation, the desirable value for $L$ is 8, and in case of descensional ventilation, it is less than that for horizontal airways (Mishra, 1986).

From Equation 1 it is evident that air velocity is the most important parameter governing the layering number and hence, the length and mixing characteristics of the layer (McPherson, 1993). The tendency of streaming is reduced at turbulent airflow and rough roof with obstructions such as roof bar etc. The air current travelling down the dip breaks up streaming of methane and carries the methane down the dip. However, the tendency of streaming is greater with an up the dip air velocity and the required critical velocity to break the methane layering becomes higher. Therefore, the critical velocity for such breaking depends on the dip and varies from 0.04 m/s for a slope of 1 in 100 to 1.45 m/s for a slope of 1 in 2.8 in smooth airways (Mishra, 1986).

Several methods are used to break methane layering in underground coal mines. However, a good safety practice to be followed in coal mines to break methane layers and reduce explosion risk is by rapidly diluting methane to safe concentrations through adequate ventilation. Where it is impracticable to dilute methane below safer limit by ventilation only due to very high gas emission, pre-mining methane drainage should be practiced to minimise emission of methane into the mine atmosphere. Further, regular monitoring of methane concentration as well as adequate measures for dispersion of methane layers in underground coal mines should be employed.

Although, some research has already been done on methane layering, there is still a lot of scope for conducting further research in this area and the outcome of which will be very useful for the
mining industry in minimising the hazards caused due to methane layering. Keeping these in mind and considering the difficulty of carrying out this study in actual mining conditions, this research work focuses on the computational simulations of methane layering using computational fluid dynamics (CFD) software, which is a simpler tool to simulate the streaming of methane in complex mining conditions.

**METHODOLOGY**

In this study, 3 dimensional numerical simulations of methane roof layering in tailgate of a retreating longwall mine were performed using ANSYS Fluent 12.0 software. The layout of a typical retreating longwall panel is shown in Figure 1. The simulations were carried out at varied airflow rates assuming constant rate of methane emission into the tailgate 30 m³/min and the lengths of methane layers were measured. The length of methane layer was considered as the distance from the goaf edge (source of methane emission) to the point where the mean concentration of methane in the layer was 5% and it was estimated from the contours of methane concentrations obtained from CFD simulations. The layering numbers at different air velocities were computed using the empirical relation proposed by Bakke and Leach (Equation 1) and the corresponding methane layer lengths were estimated from the graph produced from the experimental data of Bakke and Leach (1962) for level airways and compared with those obtained from the CFD simulations.

**Geometry creation and meshing**

A three-dimensional model of the tailgate of a retreating longwall mine was created for simulation in this study. At the selected minimum air velocity of 0.5 m/s and assumed methane emission rate of 30 m³/min, the layering number was computed to be 0.65 using the Bakke and Leach (1962) relationship and the corresponding layering length of 59 m was estimated as maximum. Therefore, a horizontal tailgate of 4.8 m by 2.4 m cross-section and 100 m length starting from the goaf edge as well as a part of the longwall face was considered for the geometry. The meshing of the geometry was done using hex mesh cells. Initially, a grid independence study was conducted with three sets of cell numbers, viz. 1.2, 1.8 and 2.4 million and the simulation results in terms of methane concentration were compared to ensure a mesh independent solution. No significant difference in the simulation results was observed with cell numbers of 1.8 and 2.4 million and therefore, a grid of around 1.8 million cells was finally chosen for meshing of the geometry. The meshed geometry of the tailgate and a part of the longwall face is shown in Figure 2.

**RESULTS AND DISCUSSION**

The simulations were performed at varied air velocities ranging from 0.5 to 4.0 m/s at an interval of 0.5 m/s using three turbulence models and the results were analysed. The variation of methane concentration at different air velocities were analysed from the contours of methane volume fraction. However, the contours of methane concentration at the lowest and highest air velocities of 0.5 and 4.0 m/s are shown in Figures 3 and 5 respectively. Similarly, the variation in methane concentration at 10 m intervals from the goaf edge were analysed from the contours as shown in Figures 4, and 6 at air velocities of 0.5 and 4.0 m/s respectively. From these contours, it was observed that the concentration of methane decreases with increase in air velocity. Also, a decrease in methane concentration with distance from the goaf edge was noticed due to dilution with air.

From the simulation results it was also noticed that the highest turbulence effect was caused by the Spalart-Allmaras model followed by the K-ε and K-ω models. As a result, highest mean methane concentration at different segments (10 m intervals) was
found in the contours of K-ω model and the lowest mean concentration was obtained in the case of Spalart-Allmaras model, which can be clearly seen from the Figures 4 and 6. However, such variation in the mean methane concentration values was found to be significant up to an air velocity of 2.5 m/s and least variation was noticed at the highest air velocity of 4 m/s, see Figures 5 and 6.

The variation in the length of methane layer with air velocity and layering number shown in Figure 7 clearly shows that the simulation results follow the trend similar to the trend obtained by Bakke and Leach through experimentation. Although, for a particular air velocity, different models predicted the layering length differently, a decrease in methane layering length with increase in air velocity and layering number was generally noticed. Comparatively, the layering lengths predicted by the Spalart-Allmaras and K-ε models closely matched. A wider variation in the layering length values predicted by the models was noticed, at higher air velocities (> 2.5 m/s); however, a lower variation was noticed at lower air velocities. At the highest air velocity of 4 m/s, methane layering lengths predicted by the

models varied in the range of 3-17 m. Among all the models we used for simulation, while K-CD and Spalart-Allmaras models respectively over estimated and under estimated the layering length, the K-ε model predicted the layering length values between the two models. The under estimation of the layering length by Spalart-Allmaras model may be due to its greater dispersion effect. Since, among all the three models, K-ε model predicts the layering lengths that closely match with the results of Bakke and Leach, it may be considered as the preferred model for CFD simulation of methane roof layering phenomenon.

Further, disparities between the experimental results and present simulation results may be due to adopting a different geometry and methane injection point from that used by Bakke and Leach (1962). Lastly it may be mentioned here that although there are variations in the experimental and simulation results, the CFD software can be used as a tool to predict the methane layering behaviour in underground coal mines with a fair degree of accuracy.
CONCLUSIONS

The understanding of methane roof layering phenomenon is essential for implementing preventive measures to avoid explosion hazards in underground coal mines. This study demonstrated that the methane roof layering is mainly affected by the air-flow rate. This study also revealed that the variation of methane layering with air velocity computed through CFD simulations as well as experimentation follows the same trend. Also the lengths of methane roof layering computed from the simulations closely agreed. This shows that CFD software is a very useful tool for analysing the methane roof layering phenomena and estimating the optimum airflow rate with reasonable accuracy for effectively reducing the methane layering in underground coal mines. However, an attempt should be made to experimentally validate the simulation results and see how well the simulations predict experiments.

REFERENCES

Introducing a 0.1 mg/m³ limit for DPM in Western Australia:
What are the impacts on the mine ventilation system?

M.A. Tuck  Associate Professor of Mining Engineering, School of Science, Information Technology and Engineering, Faculty of Science, Federation University Australia, Australia

ABSTRACT

Following the June 2012 upgrade by the International Agency for Research on Cancer to ‘carcinogenic to humans’, the Department of Mines and Petroleum in Western Australia has published a guideline for the management of diesel exhaust adopting a limit of 0.1 mg/m³ (8 hour time weighted average (TWA) for elemental carbon. This paper analyses the impact of this guideline on mine ventilation systems by analysing a typical diesel fleet employed at a small underground gold mining operation to evaluate the impact of the guideline. In particular the impact on dilution airflow rates is investigated, as well as the impact of utilising different tier level diesel engines. The paper also questions whether dilution ventilation is the optimum way of achieving the guideline level and reviews some of the existing and possible future technologies which can reduce the emission of diesel particulate matter in underground workings.

INTRODUCTION

In June 2012 the International Agency for Research on Cancer upgraded the status of diesel particulate matter to ‘carcinogenic to humans’ IARC (2012). In response to this the department of Mines and Petroleum in Western Australia (WA) has published a guideline for the management of diesel exhaust adopting a limit of 0.1 mg/m³ (8 hour TWA) for elemental carbon DMP (2013). This paper analyses the impact of this guideline on mine ventilation systems by analysing a typical diesel fleet employed at a small underground gold mining operation to evaluate the impact of the guideline. In particular the impact on dilution airflow rates is investigated, as well as the impact of utilising different Tier level diesel engines. The paper also questions whether dilution ventilation is the optimum way of achieving the guideline level and reviews some of the existing and possible future technologies which can reduce the emission of diesel particulate matter in underground workings. The baseline on which comparison is to be made is with the previous diesel regulation which required a minimum airflow for diesel exhaust dilution of 0.05 m³/s kW rated power or 5 m³/s per 100 kW of rated diesel power.

BASE MINE DATA FOR THE STUDY

The base mine data used in this study is summarised in Table 1 and lists a typical set of diesel equipment that may be found at a typical small scale (less than 200,000 oz. of production) gold mine in Australia using an open stoping method of production. It should be noticed that the light vehicle fleet of Toyota Land Cruisers and similar vehicles are not included on the list, neither are other small diesel engines under 35 kW rated power. It is quite common to not include these when developing airflow requirements, a matter of debate raised by Brake and Nixon (2008). Table 1 also includes the airflow requirements for each item of diesel equipment calculated using the previous 0.05 m³/s kW regulation.

Thus under the previous guideline, the airflow required to cover the diesel dilution requirement was 235 m³/s. This type of mine utilises a large proportion of auxiliary ventilation within the stoping and development areas and volumetric efficiencies are not always high for a mine with diesel exhaust as its primary ventilation concern. This type of mine may well have to double this airflow at a minimum to provide acceptable environmental conditions within the mine. For the increasing number of Australian mines developing or already having heat issues the airflow can increase further.

Table 1: Base diesel fleet details for the case mine and airflow requirements using the previous requirement of 0.05 m³/s/kW

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Engine Power (kW)</th>
<th>Required Volume (m³/s)</th>
<th>Number of units</th>
<th>Total Volume (m³/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Atlas MT 5020 truck</td>
<td>485</td>
<td>24</td>
<td>1</td>
<td>24</td>
</tr>
<tr>
<td>Cat AD45B Truck</td>
<td>438</td>
<td>22</td>
<td>5</td>
<td>110</td>
</tr>
<tr>
<td>Cat R2900G Bogger</td>
<td>321</td>
<td>16</td>
<td>1</td>
<td>16</td>
</tr>
<tr>
<td>Cat R1700G Bogger</td>
<td>263</td>
<td>13</td>
<td>3</td>
<td>39</td>
</tr>
<tr>
<td>Volvo L120F IT</td>
<td>180</td>
<td>9</td>
<td>3</td>
<td>27</td>
</tr>
<tr>
<td>Cat Grader</td>
<td>138</td>
<td>7</td>
<td>1</td>
<td>7</td>
</tr>
<tr>
<td>Normet Charge Unit</td>
<td>120</td>
<td>6</td>
<td>2</td>
<td>12</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>235</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

ANALYSIS METHOD

Diesel dilution rates as a given volumetric flow rate per rated engine kilowatt are typically used as guidelines or statutory limits depending on the country or state a mine operates in Brake and Nixon (2008). Typically these dilution airflows can range between 0.04 to 0.1 m³/s per rated kW. These dilution rates are based on gas dilution rates using a standard diesel fuel and are based on the dilution requirements for Carbon Monoxide (CO) and Nitrogen Oxides (NOₓ). As such, there is no guarantee that if these are applied that the diesel particulate concentrations will be less than the guideline required for Diesel Particulate Matter (DPM), in the case of Western Australia 0.1 mg/m³ for an 8 hour shift. It should be noted that in Australia a typical shift is 12 hours in length, with a typical two and one roster, and the time weighted average exposure for DPM reduces to 0.05 mg/m². For the purposes of this paper an 8 hour shift will be assumed to simplify the analysis. This has little effect on the conclusions.
Diesel engines are made by a number of manufacturers and can be certified for use in mines in a number of ways. In the United States of America and Canada diesel engines are tested for underground use by the Mine Safety and Health Administration (MSHA) and CANMET respectively. In the United States, the MSHA provides two ventilation rates, the ‘ventilation rate (VR)’ for gaseous emissions, and the ‘particulate index (PI)’, for particulate emission based on the total carbon content. The Particulate Index is the airflow requirement needed to dilute DPM to 1.0 mg/m³, whilst the MSHA Part 7 Ventilation Rate is the airflow required to dilute Carbon Dioxide (CO₂) to 5000 ppm, CO to 50 ppm, NO to 25 ppm and NO₂ to 5 ppm.

The method used within this study uses the VR and PI to evaluate diesel exhaust emissions as described by Haney (2012). Equations have been developed to enable the airflow for both PI and VR related to the rated diesel power of a machine. These equations are based on MSHA engine testing and approvals and are as follows:

**For Ventilation Rate:**
\[ Q = 0.035 \text{ m}^3/\text{s}/\text{kW} \text{ for Tier 1 and 2 engines} \]
\[ Q = 0.023 \text{ m}^3/\text{s}/\text{kW} \text{ for Tier 3 and 4 engines} \]

**For Particulate Index:**
\[ Q = 0.098 \text{ m}^3/\text{s}/\text{kW} \text{ for Tier 1 and 2 engines} \]
\[ Q = 0.124 \text{ m}^3/\text{s}/\text{kW} \text{ for Tier 3 engines} \]
\[ Q = 0.010 \text{ m}^3/\text{s}/\text{kW} \text{ for Tier 4 engines} \]

It should be noted that Tier 3 engines produce more particulate matter than Tier 2 engines, however in terms of gaseous pollutants there is an improvement by moving from Tier 2 to Tier 3 engines. Another point to note is that the airflow calculated for PI is based on a value of a 1.0 mg/m³ concentration from the tailpipe. The proposed Western Australian standard is for 0.1 mg/m³ in the general body, thus the airflow for PI needs to be multiplied by 1/0.1 or 10 assuming an 8 hour TLV-TWA.

**RESULTS**

The airflow values for VR and PI for the engine Tiers were applied to the equipment listed in Table 1. Table 2 shows the results generated for VR.

For PI the results are presented in the following tables. Table 3 gives the results for PI based on the MSHA 1.0 mg/m³ tailpipe emission; Table 4 lists the results amending the results in Table 3 to the Western Australian requirement for a general body DPM concentration of 0.1 mg/m³. Table 4 amends the results in table assuming that diesel exhaust filtration that is 80% efficient is applied to the exhaust.

Table 6 amends the results of Table 5 by multiplying by the number of the various units in use and reports the total diesel emission airflow required ignoring the additional requirements of auxiliary ventilation, bypass fan airflow requirements and ventilation circuit efficiency.

**DISCUSSION**

Comparison of Table 1 and Table 2 shows that the previous 0.05 m³/s/kW requirement in Western Australia provides greater airflow for dilution of gaseous diesel emissions than determined using the VR formulas, indicating that in terms of gaseous products the limit was a reasonable level.

Comparison of Tables 1, 3, 4, 5 and 6 shows that the required airflow to dilute particulate matter is very dependent on the Tier level of the engine under analysis. Excepting Tier 4 level engines, the airflow requirements for PI are greatly higher than those from the previous 0.05 m³/s/kW requirement, also a point to note here as...
Table 6. Total airflow requirements based on fleet size from table 5 ignoring the additional requirements of auxiliary ventilation bypass fan airflow requirements and ventilation circuit efficiency

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Engine Power (kW)</th>
<th>PI Tier 1/2</th>
<th>PI Tier 3</th>
<th>PI Tier 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Atlas MT 5020 truck</td>
<td>485</td>
<td>95.06</td>
<td>120.28</td>
<td>48.5</td>
</tr>
<tr>
<td>Cat AD45B Truck</td>
<td>438</td>
<td>429.24</td>
<td>543.12</td>
<td>219.0</td>
</tr>
<tr>
<td>Cat R2900G Bogger</td>
<td>321</td>
<td>62.92</td>
<td>79.61</td>
<td>32.1</td>
</tr>
<tr>
<td>Cat R1700G Bogger</td>
<td>263</td>
<td>154.64</td>
<td>195.67</td>
<td>15.78</td>
</tr>
<tr>
<td>Volvo L120F IT</td>
<td>180</td>
<td>105.84</td>
<td>133.92</td>
<td>54.0</td>
</tr>
<tr>
<td>Cat Grader</td>
<td>138</td>
<td>27.05</td>
<td>34.22</td>
<td>13.8</td>
</tr>
<tr>
<td>Normet Charge Unit</td>
<td>120</td>
<td>47.04</td>
<td>59.52</td>
<td>24.0</td>
</tr>
<tr>
<td>Total</td>
<td>921.79</td>
<td>1166.34</td>
<td>94.06</td>
<td></td>
</tr>
</tbody>
</table>

Table 7. Amended table 5 accounting for the provision of exhaust treatment in Tier 4 engines

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Engine Power (kW)</th>
<th>PI Tier 1/2</th>
<th>PI Tier 3</th>
<th>PI Tier 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Atlas MT 5020 truck</td>
<td>485</td>
<td>95.06</td>
<td>120.28</td>
<td>48.5</td>
</tr>
<tr>
<td>Cat AD45B Truck</td>
<td>438</td>
<td>85.85</td>
<td>108.62</td>
<td>43.8</td>
</tr>
<tr>
<td>Cat R2900G Bogger</td>
<td>321</td>
<td>62.92</td>
<td>79.61</td>
<td>32.1</td>
</tr>
<tr>
<td>Cat R1700G Bogger</td>
<td>263</td>
<td>51.55</td>
<td>65.22</td>
<td>26.3</td>
</tr>
<tr>
<td>Volvo L120F IT</td>
<td>180</td>
<td>105.84</td>
<td>133.92</td>
<td>54.0</td>
</tr>
<tr>
<td>Cat Grader</td>
<td>138</td>
<td>27.05</td>
<td>34.22</td>
<td>13.8</td>
</tr>
<tr>
<td>Normet Charge Unit</td>
<td>120</td>
<td>23.52</td>
<td>29.76</td>
<td>12.0</td>
</tr>
</tbody>
</table>

Table 8. Amended Table 6 accounting for the provision of exhaust treatment in Tier 4 engines

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Engine Power (kW)</th>
<th>PI Tier 1/2</th>
<th>PI Tier 3</th>
<th>PI Tier 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Atlas MT5020 truck</td>
<td>485</td>
<td>95.06</td>
<td>120.28</td>
<td>48.5</td>
</tr>
<tr>
<td>Cat AD45B Truck</td>
<td>438</td>
<td>429.24</td>
<td>543.12</td>
<td>219.0</td>
</tr>
<tr>
<td>Cat R2900G Bogger</td>
<td>321</td>
<td>62.92</td>
<td>79.61</td>
<td>32.1</td>
</tr>
<tr>
<td>Cat R1700G Bogger</td>
<td>263</td>
<td>154.64</td>
<td>195.67</td>
<td>15.78</td>
</tr>
<tr>
<td>Volvo L120F IT</td>
<td>180</td>
<td>105.84</td>
<td>133.92</td>
<td>54.0</td>
</tr>
<tr>
<td>Cat Grader</td>
<td>138</td>
<td>27.05</td>
<td>34.22</td>
<td>13.8</td>
</tr>
<tr>
<td>Normet Charge Unit</td>
<td>120</td>
<td>47.04</td>
<td>59.52</td>
<td>24.0</td>
</tr>
<tr>
<td>Total</td>
<td>921.79</td>
<td>1166.34</td>
<td>94.06</td>
<td></td>
</tr>
</tbody>
</table>

Table 9. Amended Table 8 to elemental carbon assuming EC = 0.6 x TC

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Engine Power (kW)</th>
<th>PI Tier 1/2</th>
<th>PI Tier 3</th>
<th>PI Tier 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Atlas MT5020 truck</td>
<td>485</td>
<td>57.04</td>
<td>72.17</td>
<td>29.1</td>
</tr>
<tr>
<td>Cat AD45B Truck</td>
<td>438</td>
<td>257.54</td>
<td>325.87</td>
<td>131.4</td>
</tr>
<tr>
<td>Cat R2900G Bogger</td>
<td>321</td>
<td>37.75</td>
<td>47.76</td>
<td>19.3</td>
</tr>
<tr>
<td>Cat R1700G Bogger</td>
<td>263</td>
<td>92.79</td>
<td>117.40</td>
<td>47.3</td>
</tr>
<tr>
<td>Volvo L120F IT</td>
<td>180</td>
<td>63.30</td>
<td>80.35</td>
<td>32.4</td>
</tr>
<tr>
<td>Cat Grader</td>
<td>138</td>
<td>16.23</td>
<td>20.53</td>
<td>8.28</td>
</tr>
<tr>
<td>Normet Charge Unit</td>
<td>120</td>
<td>28.22</td>
<td>35.71</td>
<td>14.4</td>
</tr>
<tr>
<td>Total</td>
<td>553.07</td>
<td>699.81</td>
<td>282.2</td>
<td></td>
</tr>
</tbody>
</table>

reported by Haney (2012), one of the main differences between Tier 3 and Tier 4 engines is the inclusion of a diesel particulate filter in the exhaust, thus the values for Tier 4 engines shown in Tables 5 and 6 may be viewed as suspect and may be better reported as shown in Tables 7 and 8.

These results give rise to the conclusion that the new Western Australian guideline will result in higher airflow requirements for DPM dilution to meet the new guideline. At this point there is a need to introduce another variable. The Western Australian guideline is based on 0.1 mg/m³ of elemental carbon, whilst the MSHA calculations are based on the total carbon content. The relationship between total carbon and elemental carbon depends on a wide range of factors, including type of diesel fuel, engine and filtration maintenance, machine operation and a number of other factors. To illustrate this, the results from Table 8 have been further amended assuming that elemental carbon forms 60% of total carbon as shown in Table 9.

Inspection of Table 9 shows that the airflows under the new guideline are still higher than under the previous regulation but for the case of Tier 4 engines not much higher. The Western Australian inspectorate have been applying the guideline for almost a year, results of monitoring at mine sites have recently reported DMP (2014). The results indicate that the 0.1 mg/m³ standard is being attained at mine sites. This is not because Tier 4 engines are used at the mine sites, but rather that operation of diesel equipment at the mines is not continuous at the monitoring locations, possibly explaining the overall compliance measured to date.

CONCLUSIONS

Firstly the results presented are from a desktop study but show that from a gaseous emissions perspective that the new guideline has little impact on airflow requirements. However from a particulate perspective, the new guideline may well have the impact of increasing airflow requirements and therefore total ventilation costs. The paper only looks at diesel exhaust emissions and does not account for other ventilation pollutants. In most mines, requirements for air and the dominant airflow pollutant change both in time and space and ventilation flow requirements should always account for all pollutants found at a particular mine site.

Second, differently different Tier levels of engines have different impacts on the PI airflow. It is fair to state that if considering new or replacement equipment consideration should be given to the engine Tier level with Tier 4 level engines being the preferred type. A third conclusion is that engine Tier level is only one factor to account for and that filtration technology and efficiency also provide a key to reducing DPM exposure.

Fourthly, whilst it appears that gaseous emissions from exhausts are well covered by the new guideline, using different fuels such as biodiesels can impact on the rate of production of certain gases and a full monitoring program should be undertaken to assess this if alternative fuels are employed.

Filtration of DPM has improved over the years; consideration should also be given to other methods of achieving the goal of reducing DPM emissions in the respirable range from the tailpipe. An example of this may be acoustic agglomeration which is...
Diesel engines are the workhorse of the underground metalliferous mining industry in Australia and other countries, and perhaps further research should be undertaken into alternative power sources such as fuel cells and / or further investigation of existing technologies such as electric vehicles, either battery or cable systems.

As is normal within the industry, solutions to the problems faced at a particular point in time will emerge and will involve considering more than a single element of a particular problem.

ACKNOWLEDGEMENTS

I would like to express thanks to the Mine Ventilation Society of Australia for providing the catalyst to undertake this study. I would also like to thank Dr Jackie Tuck, my wife for putting up with the endless time I have spent on the study.

REFERENCES


MechCaL expands its footprint into Zambia

MechCaL has launched distribution of their products in Zambia in association with Best Line Mining Supplies as their agents. South African based fan and ventilation firm, MechCaL, announced in September 2015 that their renowned fans would be rolled out to the Zambian mining market. This marks a key point in the company’s on-going geographic expansion strategy which is aimed at increasing its presence in key growth markets.

“MechCaL has for quite some time had a policy of diversifying products and markets. The product diversification has progressed well but penetrating the smaller auxiliary fan market space outside of South Africa has been a challenge,” says MechCaL spokesperson and Managing Director, Jan du Plessis. “We have started supplying products to mines in the Zambian market and we are positive that, in future, the quality and energy efficiency of our products will be valued as the premium fan product in the rest of Africa.”

This move builds upon the preliminary research into the Zambian market that MechCaL has been conducting which includes a fan demonstration that was held in Kitwe in April 2015. A demonstration of MechCaL’s energy efficient 45Kw fan using an ISO test column to analyse the velocity, flow and pressure of the fan was carried out to showcase the remarkable power saving capabilities of MechCaLs’ products.

Gavin Ratner, Head of Marketing (at the time) at MechCaL, led the demonstration and said that the results of the demonstration made an impression on the Zambian delegates. “We conducted the test in the presence of mining representatives from the surrounding areas and their reaction to our products was very positive. This definitely opened the door for MechCaL to begin exporting products to service the Zambian market,” said Ratner.

In order to ensure smooth delivery and availability of fans to Zambia’s mining market, MechCaL has appointed Best Line Mining Supplies Ltd as their agent. According to Tom Hight at Best Line their move to join up with MechCaL will help alleviate demand for energy efficient products in the region. “We decided to become MechCal’s partner in both Zambia and the DRC due to the level of innovation in their products. Energy is becoming a concern in our region, with more demand than supply of power, coupled by a 30% energy increase in the 2014-2015 year. This has encouraged Best Line to assist with client assisted solutions to optimise the current mining operations,” says Hight.

“We appointed Best Line as they are extensively involved in ventilation in the sub-Saharan region. They are currently supplying the range of 15 kW, 37 kW, 45 kW and 75 kW MechCaL fans,” says du Plessis.

To date, MechCaL fans have been installed for Glencore’s Mopani Copper Mines Plc. To date 35 fans from across the MechCaL product range, including five of MechCaL’s new Jet Fans have been installed.

Hight believes that the outlook for MechCaL’s products in terms of Zambia’s mining industry is positive, as new innovative equipment for mines could play an important support role to the mines overcoming recent challenges in the region. “We have been hit hard in the copper industry along with all commodity prices that have crashed since June 2015. This has placed a lot of strain on the mining economy, coupled with increases in mineral royalties and energy tariff increases. However, we remain positive and are working closely with the mines to provide solutions to improve efficiency.”

According to du Plessis, one of the main challenges that they are faced with affecting exporting and installing fans for Zambian clients would be ensuring that they receive MechCaL’s latest innovative products. “We would definitely want to see more of our newer products from the Jet Fan range being supplied. We have developed an extensive range of products for the Jet Fan line from fans as small as 5kW all the way up to 45kW.”

“Supplying the best possible products to each and every client supports our goal of becoming the premier fan supplier in Africa. Zambia is one of the larger underground markets that we feel can be easily supplied from South Africa,” du Plessis concludes.

Visit www.mechcal.co.za for more information. Like them on Facebook at Mechcal.
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Visit www.mechcal.co.za for more information.

Like them on Facebook at Mechcal.
Managing mining costs - Discussions at mining indaba

Delegates at the 2016 Joburg Indaba breakfast seminar at the Johannesburg Country Club, in Auckland Park on 6th April all held the same view that the central theme to conversations in the mining sector at the moment is “commodities”.

Continuing to engage in critical conversations on the South African mining sector, representatives of the mining sector at the breakfast seminar agreed that commodity prices have nothing to do with the price of production.

The reality is that the only price that South Africa could control would be that of platinum, due to the fact that we control 70% of global production. For 90 years South Africa mined most of the world’s gold. Considering the constant rising trend in the Rand/Dollar rate, currently it looks like the best metal to mine, but for how long begs the question.

Mining policy consultant at Cadiz Solutions, Peter Major went on to state “The South African mining under-performance is no longer cyclical, it’s now secular.” Major backed up his view citing many factors that he believes have contributed to today’s bad state of mining.

• The subsequent loss of billions of Dollars in CAPEX due to the 2002 Nationalisation.
• Mining companies have been forced into debt within unknown partnerships required under the law of +25% BEE.
• Effects of a complex, burdensome and interpretive Labour Relations Act.
• Dramatic rise in total production costs from 2% to 20% due to Eskom price increases and stoppages.
• Union rivalry.
• Inadequate rule of law applied unevenly and often not at all.
• Breakup of the mining houses, as a result all gold mines are now on their own.
• Huge and ever growing social labour plans and requirements.
• New tax and royalty systems provoking less reason to mine underground.

The biggest problem as to why mining is now secular, Major believes, is the miners themselves. “We don’t have miners any more. We have gamblers and dreamers. Miners need to take control of their mines, their costs, and their people. They need to think how they can cut costs and increase productivity”.

The constant rise in the world’s population could be viewed as a plus for the metal sector. With more consumers, there is a larger demand for minerals. Along with increased population trends comes a rise in unemployment, and South Africa currently stands at eight million unemployed. Labour is available.

Delegates deliberated in round table discussions throughout the morning and presented the following outcomes.

• A strong belief that the Department of Mineral Resources (DMR) is an officially corrupt entity.
• It will require some bravery since the mining industry cannot control the cycle, the one thing that it can control is costs.
• To weather the storm and be competitive in terms of productivity, the mining industry must drive innovation.
• Mining entities must bring costs back down to where the commodity price is.
• To be more productive, mining must look to incentivising labour and implement cost management structures.
• The industry wishes to share lessons amongst themselves, and to drive detailed discussions with Government, albeit stemming frustrations on the lack of any outcomes from last year’s five-week long Mining Phakisa workshop. Apparently the report is sitting in the Presidency, presumably alongside the Mining Charter.

Bernard Swanepoel Chairperson of the Joburg Indaba wrapped up by saying “There is a newly found boldness in the mining sector and we will continue to speak the truth to both power (Government) and to each other. This is the only way we can move forward”.

Wynand van der Merwe
www.wynandvandermerwe.com
MVSSA Election of members of Council

Visit the MVSSA web site (www. MVSSA.co.za) to follow or partake in the election of members of Council for the 2016 to 2017 session.

The election process has been scheduled as follows:

15th April to 9th May 2016  Nominations
23rd May to 10th June 2016  Voting
24th June  MVSSA Annual General Meeting

Members in good standing are invited to partake in the election and to attend the AGM.

Details are available on the MVSSA web site.

Alternatively watch your e-mail.

The Mine Ventilation Society of South Africa will hold its 2016 annual conference titled:

The increasing relevance of the Mine Ventilation Profession in a highly regulated and cost constrained environment

Dates: 1st - 2nd September 2016
Venue: Emperor’s Palace Ekurhuleni

For further information or delegate booking forms go to www. MVSSA.co.za
Or email: secretary@mvssa.co.za
Or tel: +27 11 482-7957
BBE Laboratory

Analytical Services

BBE Laboratory is a specialist airborne pollutant analytical laboratory providing the following services to all sectors of the South African mining industry:

- Occupational hygiene filter preparation
- X-ray diffraction [XRD] analysis
- Fourier transform infrared [FTIR] analysis [KBr pellets and DoF]
- KBr pellet preparation

What sets BBE Laboratory apart?

- SANAS accredited laboratory [XRD, IR, KBr pellets and weighing]
- Participate in national and international proficiency schemes [NIOH and WASP]
- Employ top qualified staff
- Utilise state-of-the-art equipment
- Provide a customised real-time internet-based service
- Bar-coded filter tracking
- Provide courier services for fast sample turnaround
- Have close links with leading national and international role players

BBE Consulting

Occupational Hygiene Services

As an internationally recognised company in the field of mine ventilation and refrigeration, BBE Consulting has expanded its expertise to provide the following specialist occupational hygiene consulting services:

- Legal compliance audits
- Mine standards and systems
- Operational compliance interface with managerial standards, systems and/or procedures
- Voluntary and mandatory occupational hygiene Codes of Practice [in terms of sections 9.1 and 9.2 of the MHS Act]
- Occupational hygiene risk assessments [in terms of section 11 of the MHS Act]

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