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- ◆ Sealing waste areas of mines where Radon may be present
- ◆ Practices for conveyor roadway segregation and designation of escapeways in Australian underground coal mines
- ◆ Energy saving optimisation of auxiliary ventilation systems in hard rock development headings
- ◆ Comparison of diesel particulate matter ambient monitoring practices in underground mines in Australia, the United States and South Africa



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Cover Picture:



Photo taken by Shaun Levings

Editor's Comment

Coal mining explosions

The history of coal mining is punctuated with the frequent occurrence of tragic multi-fatality explosions.

Many coal mine disasters have been caused by methane and/or coal dust explosions. Methane gas and coal dust are both produced during normal underground mining operations.

The most recently reported incidents were on 2nd and 3rd December 2016 where at least 38 workers lost their lives in two separate accidents. In the first incident, 21 people died after a coal mine blast in the north-eastern province of Heilongjiang and the next day 17 people died in a coal mine explosion in northern China's Inner Mongolia region.

The prevention or the limiting of the extent of explosions is best achieved by focusing on the source of the explosion, i.e. by preventing the ignition of methane that may be generated during the mining operation and by reducing the generation of coal dust.

Despite a history of in-depth research effort into the development of preventive and protection measures against coal mine explosions, spanning

well over a century, disasters still occur and the following quotation remains equally valid today.

"Of all the risks inherent in coal mining, the one most feared by coal miners is an explosion. Explosions are not the biggest cause of loss of life. On a statistical basis, explosions may be amongst the less frequent events in mining causing loss of life but, apart from fires and flooding, there is no other cause capable of wiping out the entire workforce below ground at the time." These words were written by Joseph Dickson, an Inspector of Mines in the Lancashire coalfield, United Kingdom, in 1850.

Table 1 lists the top 11 coal mine disasters, ranked according to the number of fatalities, and the most likely cause of the explosion.

Because coal mine explosions have such devastating consequences, research into ways of preventing their occurrence has been conducted in many countries and over many decades.

In general the controls developed through the research can be categorised as the prevention of accumulation of methane through good ventilation practice, elimination of frictional sparking by the

Prof Jan du Plessis
Honorary Editor



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Table 1. Major coal mine disasters associated with explosions

Rank	Fatalities	Year	Name of Mine	Location	Most likely cause of disaster
1	1549	1942	Benxiyu Colliery	China	Ventilation system shut down
2	1099	1906	Courrieres mine	France	Miner's lamp and mishandling of mining explosives
3	687	1914	Mitsubishi Hojyo	Japan	Methane explosion
4	680	1960	Laobaidong	China	Methane explosion
5	458	1963	Mitsubishi Hojyo	Japan	Coal dust explosion
6	439	1913	Senghenydd	UK*	Electric sparking from signals
7	426	1972	Wankie Colliery	Zimbabwe	Dynamite in u/g magazine
8	388	1866	Oaks Colliery	UK	Blasting during rock excavation
9	375	1965	Dhanbad	India	Intentional act of ignition
10	372	1975	Dhanbad	India	Coal dust explosion
11	362	1907	Monongah Coal	USA**	Electric arcs or open lights

* United Kingdom ** United States of America

use of water, minimising dust generation and dispersal, and using stone dust to inert coal dusts to prevent coal dust from participating in mine explosions.

The final line of defence remains the use of barriers to prevent a coal dust explosion from propagating further into a mine.

In order to control production costs, mines have become more efficient and production rates have increased through the use of mechanised mining methods.

The use of more powerful machines has increased the rate of methane liberation, the likelihood of frictional ignitions, and lead to the increased production of fine coal particles as well as an increase in fine coal deposition increasing the risk that they will participate in an explosion.

More than 25 years ago Dr Jurgen Michelis, working in Germany proposed a framework of measures to mitigate the risk of ignitions and explosions in coal mines.

The protective measures described by him are a combination of preventative and constructive measures.

Preventative measures are measures used to prevent methane and coal dust explosions from occurring in the first place. Measures described by him include the extraction of methane from the

seam prior to mining, dilution of any methane emission by good ventilation and continuous monitoring of methane levels.

Measures employed to prevent a large accumulation of methane being ignited include active on-board ignition-suppression systems.

Constructive measures are all measures that involve construction of some kind to safeguard against explosions or their effects. They include all explosions barriers (active and passive), explosion stoppings and refuge bays.

Despite extensive research and the best efforts of the coal mining industry the hazard remains. In South Africa the prevention of explosions for a generation does not mean the hazard has been eliminated and may well lead to complacency.

In presenting a brief history of some of the major explosions, immediately followed by a description of the available control measures I make a plea for these to be rigorously implemented so that history never repeats itself and the coal mining industry of today remains free of these tragedies.

The question of whether these controls are adequate to reduce the explosion risk to as low as possible still remains and only the future performance of our industry will prove or disprove this.



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Sealing waste areas of mines where Radon may be present

D.R. Chalmers¹, G. Durandt²

¹University of New South Wales, Australia, ²BHP Billiton, Australia

ABSTRACT

Many hazards are present in mines. Some of these involve the emission of gas into workings, and in several metal mines one of the more difficult to control is radon. Current control measures for this gas include flood ventilation so that radon is diluted to below the recognised safe limits. Whilst this is an effective control it means that as the mine expands or as waste areas are abandoned, the quantity of air provided to the mines increases. This paper explores the cross-application of existing proven coal mine goaf sealing techniques to control the ingress of oxygen to mined areas that are liable to spontaneous combustion into uranium mines as a novel approach to reducing airflow requirements for sealed areas emitting radon gas. It also explores the use of compressed air instead of nitrogen as a cheaper but effective option in this case.

Keywords: Sealing, radon, uranium, dilution, hazard control

INTRODUCTION

Managing hazards within a mine is of paramount importance to provide a safe, productive workplace. In some mines, this may be controlling the risk of spontaneous combustion, heat or gases. In other mines it may be geotechnical.

Those that pose a visible change in the environment are more easily monitored. Odourless, colourless gases, especially those emitting radioactive waves and or particles such as radon are not easily monitored.

The behaviour of sealed areas, leakage in and out of them and mechanisms to control the atmosphere within them is well understood in coalmines in Australia. The techniques employed to control the ingress of oxygen into and the egress of gasses from waste areas of coal mines may be applicable to the abandoned areas of underground metal mines where radon may be present. Having an understanding of both radon formation and control as well as an understanding of spontaneous combustion and controls to mitigate or prevent it from occurring may lead to similar controls being implemented in sealing unwanted sections of mines that currently are only ventilated to control radon. Also, it is envisaged that this will only be employed in areas where the stopes are backfilled as open stopes provide too high a risk for such an unproven technique in controlling radon.

By providing a safe manageable solution, this could lead to considerable savings in operating and capital costs, as additional air just to control radon would not be warranted.

Original paper presented at the the 10th International Mine Ventilation Congress 2014

This paper briefly discusses radon, spontaneous combustion in coalmines and active sealing techniques and suggests a modification to the active sealing method as a cheaper alternative.

Radon

Radon and its daughter products may pose a hazard in mines where the orebody contains uranium. The emanation of radon is a natural part of the radioactive decay process and radon also radioactively decays through several progeny. During the decay process for U^{238} , both α and β radiation are emitted. A simplified representation of this decay is shown in Figure 1 starting from Uranium²³⁸ and ending as Lead (Moreby and Chalmers, 2005). The rate of emission is variable as it is dependent upon the composition of the mineralisation. Radon²²² is a dense noble gas with a relative density of 7.995 and diffuses slowly into the surrounding air. It has a half-life of 3.8 days. Any large emanation would remain as a concentrated body within the air mass, expanding slowly as the air diffuses into it. Gamma radiation has been omitted as it cannot be controlled by ventilation.

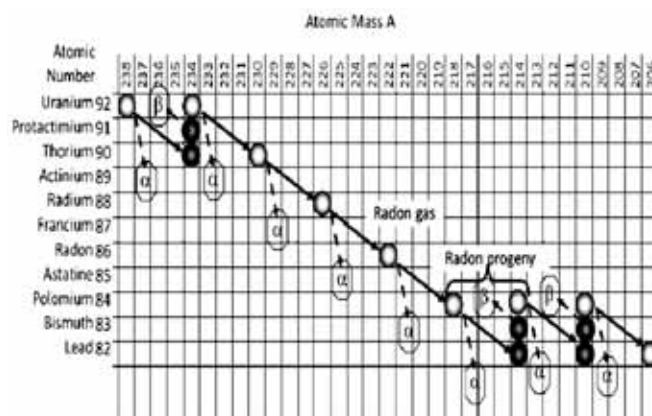


Figure 1. Uranium²³⁸ naturally occurring decay series

The main concern with this gas is the radioactivity of the daughter products that are solid products that each have a short half life and can be inhaled as a particulate and absorbed into lung tissue. Although radon decays, there is a rapid rise in daughter products until equilibrium is reached. This equilibrium is reached in about thirty hours (McPherson, 1996). This is represented in Figure 2. For clarity McPherson shows only the total decay curve for radon daughters.

Current controls for Radon

As low as reasonably achievable (ALARA) is a principle that is applied to safety systems. Some mines have set the limit for ionising radiation so that no person shall receive a dose greater than half of the annual exposure limit of 20 millisieverts. This is achieved through ventilation design, workplace systems and high ventilation quantities. However this should be noted that this would not be effective on gamma radiation.

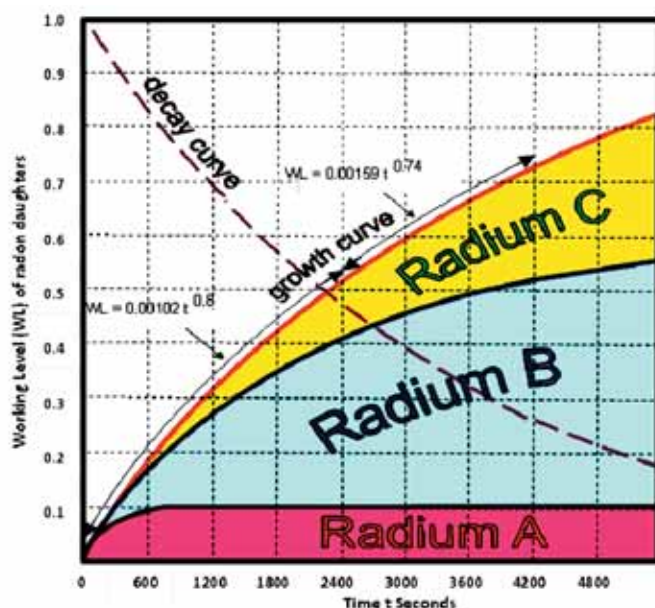


Figure 2. Growth curves of the development of radon daughters (McPherson, 1993)

The most important of these controls in mines where radon is present is the ventilation design. The essential design criteria for good radon control are dilution and residence time. If the mine footprint is relatively constant then the demand for air to provide this control will not increase greatly as the mine continues to produce. This is not the case if the mine increases its footprint and develops deeper into the orebody and surrounding rock. This increase in mine size will lead to higher demands for air, and the subsequent power to drive the required fans.

When areas of the mine are worked out they are still ventilated as the host rock, and remnant ore will still emanate radiation. For example, Olympic dam mine has a total ventilation quantity in excess of $5000\text{m}^3/\text{s}$.

Sealing sections of the mine to contain emanation areas would reduce this quantity considerably, however concerns that leakage from the potential area could contaminate fresh air airways.

If radiation and in particular uranium dust and radon gas are contained within a sealed area of the mine roadways, then the demand for air can be reduced by as much as $1500\text{m}^3/\text{s}$. This would be achieved by sealing entries into mined out areas, after the stopes have been filled. Currently these areas are flood ventilated to sweep any radon emanations into the exhaust.

Sealing and sealed areas have inherent problems and these need to be addressed to ensure that the implementation and operation of sealed areas do not pose a hazard to operations and are controlled in such a way as to prevent contamination of the mine airways.

To this end positive pressure sealing techniques similar to those used in some Australian coal mines has been proposed. This technique is explained in the context of spontaneous combustion management so that it can be understood and transferred to the context of radon control.

Spontaneous combustion can be a problem in some coal mines. In Australia, as an effective means of controlling waste areas in the mines, pressurised seals are employed to prevent the ingress of oxygen and the egress of gases from the void left after mining

This paper explores the possibility of employing these techniques in controlling sealed areas where radon may be a problem.

SPONTANEOUS COMBUSTION MANAGEMENT

Spontaneous combustion

Spontaneous combustion is the oxidation of a fuel in such a way that it slowly heats the fuel so that oxidation increases and the heat build-up continues until the fuel mass accelerates into thermal runaway. Some coals have a high propensity to spontaneous combustion and mining some of these coals can be problematic. Waste areas may contain large quantities of broken coal especially when longwall methods are employed to mine thick seams. With sufficient differential pressures across a waste area they can have the potential to provide sufficient oxygen to maintain oxidation and low flow to provide little to no heat removal. These conditions provide an environment where spontaneous combustion is likely to occur.

Void behaviour

The void left after mining has been completed is a finite volume. Convergence, rock falls, upsidence and subsidence can reduce this volume. Upsidence is the rise of strata once the overburden load has been removed. Any atmosphere that remains is subject to the gas laws. Changes in pressure and /or temperature along with the emanation of gases from the surrounding strata will influence the total volume of atmosphere that is present. If this volume exceeds the physical volume of the void space then the atmosphere will flow into the adjoining roadways. If the volume is less than the physical volume, then air from the adjoining roadways will be drawn in.

If the area has been sealed, then this behaviour of breathing in and out will still happen, however the rate of change is slowed by the resistance of the seals and other leakage paths.

Longwall goaves (waste areas in coal mines) are typically large, poorly compacted and contain voids of up to 70% of the extracted area. Small variations in barometric pressure can lead to large volumes of atmosphere being expelled or significant ingress of oxygen. As an example, a mined longwall block 4.5km long with a working height of 3.5m and a face width of 350m leaves an extraction zone of $5,512,500\text{m}^3$. If the convergence and roof collapse take up two thirds of this volume then there remain voids amounting to $1,837,500\text{m}^3$. Figure 3 shows the effect that changes of barometric pressure of up to $\pm 1000\text{Pa}$ would have on this volume.

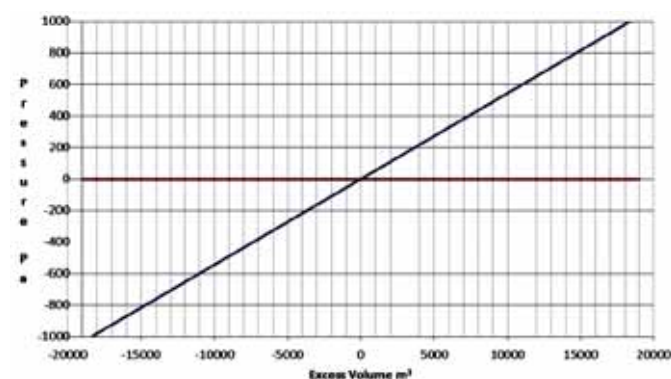


Figure 3. Void atmosphere volume with $\pm 1\text{kPa}$ pressure variation

The severity of barometric pressure change can affect the rate at which the atmosphere changes within the void. Figure 4 shows the rate of change in barometric pressure brought about by the movement of a tropical low pressure system moving into a mining region in Australia.

As the barometric pressure falls, the atmosphere in the void starts to leak from the sealed area. However the rate of leakage is dependent on the differential pressure and the resistance of the leakage paths. As the rate of change in the barometer may well be faster than the rate of change in pressure within the seals, there would be times that the sealed area would be breathing out even though the barometric pressure is rising.

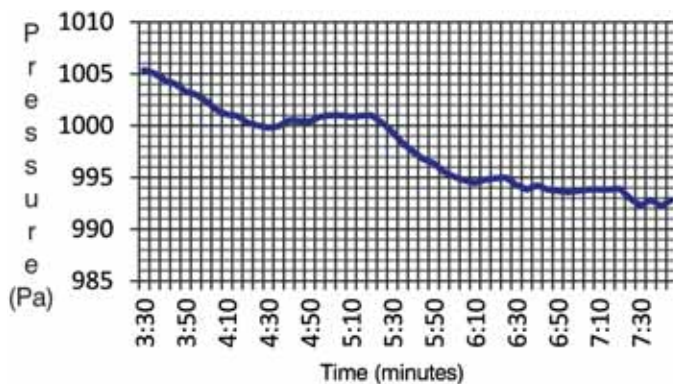


Figure 4. Barometric pressure change over a 4 hour period

Figure 5 shows the variation between the goaf pressure and the barometric pressure over a 24 hour period. The barometric pressure is taken from Rosslyn Bay weather station (BOM, 2010) as a tropical low pressure passed nearby.

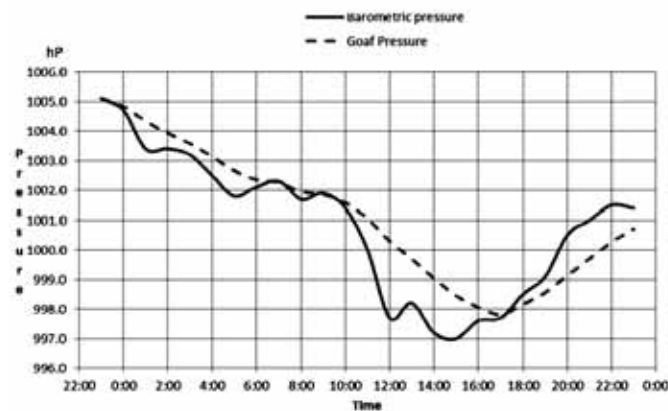


Figure 5. Goaf pressure and barometer

The goaf data was simulated using the void space of $1,837,500\text{m}^3$ sealed behind 30 seals at $20,000\text{Ns}^2/\text{m}^8$. As these seals would be in parallel the overall resistance of $22.22\text{Ns}^2/\text{m}^8$ was used to calculate the leakage and the subsequent pressure variation over an hourly interval. It was assumed that the goaf was at equilibrium at a starting pressure of 1005.1hPa .

The area between the dotted line, the goaf pressure and the solid line, barometric pressure shows that the sealed area would breathe out when the solid line is below the dotted line.

It can be seen that there are approximately 4 hours when the barometric pressure is rising that the goaf still breathes out.

Active sealing

With the higher pressures required to ventilate modern longwalls resistive, passive seals have not proven effective in all cases with the breathing and leakage providing sufficient oxygen to give rise to spontaneous combustion events. The use of pressure balance chambers also has been found to be ineffective or only partially successful (Brady, Burra and Calderwood, 2008) as they would not combat the fluctuations in barometric pressure. Positive pressure chambers were installed at Austar Coal mine and eliminated the flow of oxygen into the goaf (Brady, Burra and Calderwood, 2008).

These active seals work by providing a chamber between two seals built in parallel in the one airway and then injecting nitrogen into the chamber. The chamber is pressurised so that the pressure in the chamber is higher than the goaf and higher than the external roadway. The chamber isolates the goaf from fluctuations from the barometric pressure by topping up the chamber on a rising barometer and allowing leakage to lower the pressure as the barometer falls. Since the chamber is at the higher pressure then the leakage will always be from the chamber into the goaf and into the external airway. The only pressure that the goaf experiences is that of the chamber and by keeping it constant then no leakage will occur from the goaf.

Figure 6 shows a seal arrangement with the positive pressure chamber. The volume enclosed by the two seals is filled with nitrogen. By providing seals of different construction, allows for more leakage in one direction, however if the weaker one is on the inside and it fails, it is almost impossible to repair. Even if it fails then the leakage would still be into the goaf; however the volume of nitrogen that would be required would rapidly increase. A solution to this would be to make the inner one from a flexible membrane so that any convergence would leave the membrane unaffected and functioning. A seal arrangement with this membrane is shown in Figure 7.

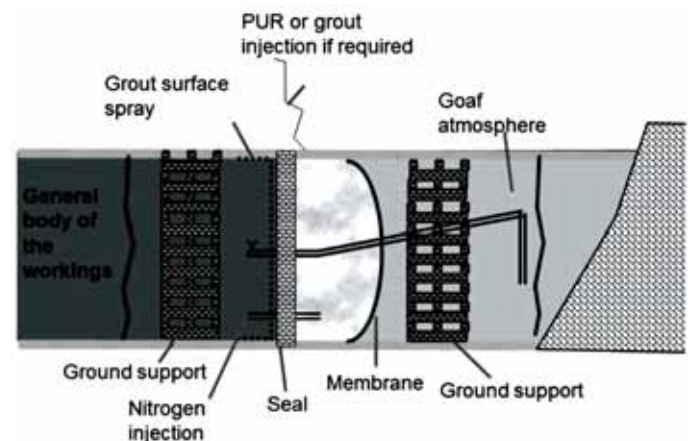


Figure 6. Active seal design with rigid seals (Moreby and Chalmers 2014)

APPLICATION TO METAL MINING

Similarities and differences

Any void that contains an atmosphere will be subject to a change in the barometric pressure unless isolated by a pressure barrier. The voids in coal mining tend to be tabular and extend for several

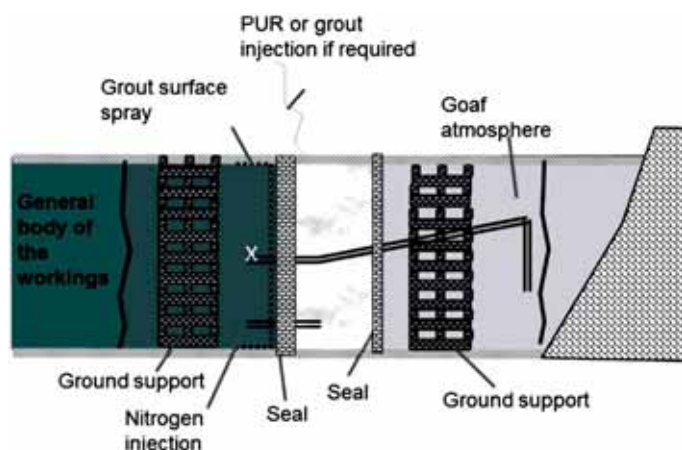


Figure 7. Active Sealing with membrane (Moreby and Chalmers, 2014)

kilometres. A metal mine consists of declines, shafts, drives, cross cuts, ore passes raises and stopes. Typical airways in a coal mine range between 2-4m high and 4.5-5.5m wide, and can extend to 7km in length, however metalliferous mine drives range from 4x4m to 7x7m and are typically only 20m-400m in length.

Prior to backfilling, the size of the void space in a stope can be considerable and as such would exhibit large variations in volume with the changing barometric pressure.

The stopes may be backfilled with a cement fill reducing the void volume, and goaves are generally designed to collapse partially filling the voids.

Entries into a goaf, headings and cut-throughs could be as many as 60 seals. Since these seals are parallel paths then the overall resistance of the seals is considerably lower than that of an individual seal. Depending on the section of the metalliferous mine that is sealed, it may be as few as 5 seals.

Figure 8 displays leakage quantities for three types of flow $P=RQ$ laminar flow, Transitional flow $p=RQ^{1.8}$ and $P=RQ^2$ for turbulent flow. For an applied pressure of 1000Pa and an individual seal resistance of 20,000Ns²m⁻⁸, the quantities flowing for more than five seals would suggest a transitional to turbulent flow regime.

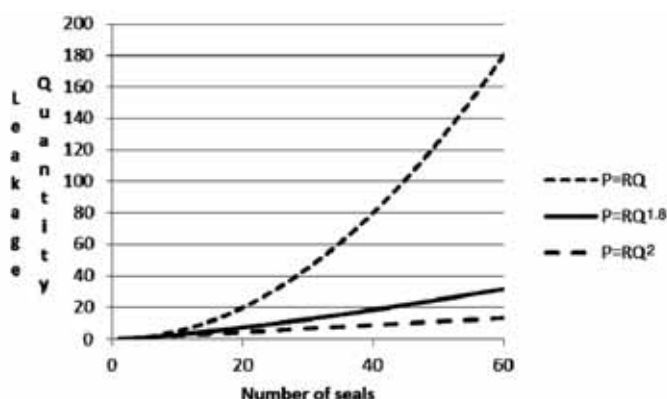


Figure 8. Leakage quantity vs number of seals

This would provide a more effective seal and the leakage would be considerably less. For seals of the same resistance and the same differential pressure the leakage across five seals would be approximately 17% of that of thirty seals in parallel. With a

barometric pressure drop of 1000Pa over 4 hours, and with five seals would result in a leakage of approximately 120ml/s. This would mean that the pressure in the sealed area would not vary much over such a fluctuation.

Hypothetical void space in a metal mine

The void left behind on a production level may consist of 20 drives 6m x 6m and 250m in length and have 20 draw points on each drive, extending 5m would be approximately 134000m³. This is considerably less than the coal mine goaf void previously discussed. This example excludes the stopes. The reason for the exclusion of the stopes is that they would be backfilled before any sealing operation would take place. Stopping ventilation while there were large voids would allow for a massive radon / radon daughter concentration to occur. It would also mean that the atmosphere that contained these contaminants would be considerable and large leakage volumes would be the result of barometric pressure fluctuations.

Eliminating the stopes by backfilling, the void of the remaining drives would be subject to the same effects and this is shown in Figure 9.

Although this variation would be small, there is the possibility that leaking atmosphere would bring radon into the intake air.

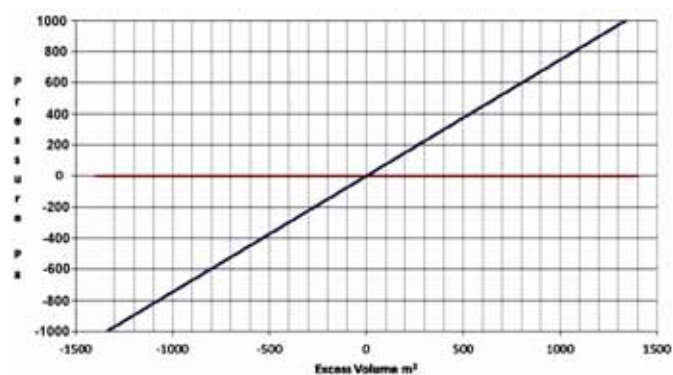


Figure 9. Volumetric change with barometric pressure variation

Reasons for active seals

In coal mining, the reasons for using active seals is to prevent the ingress of oxygen or the release of flammable or toxic gases. The prevention of oxygen is to prevent the formation of explosive atmospheres and to reduce the opportunity for a spontaneous combustion event.

In the proposed application, the risk of spontaneous combustion and the formation of explosive atmospheres are negligible. It is the control of radon and its progeny that is paramount. If passive seals were to be employed then the sealed area of the mine would be subject to the fluctuations of barometric pressure in exactly the same way that a coal mine goaf would behave.

A falling barometer would result in the volume contained within the seals breathing out with the possibility of contaminating the adjacent airways with an atmosphere containing radon. Leakage into intake airways could be problematic as it would be subject to fluctuations in barometric pressure and the emanation of radon.

A rising barometric pressure would allow for the ingress of air into the sealed area; however this should not cause any major concern

as it would only lead to the dilution of radon.

An active seal at all times provides a pressure that is higher than that of the void. All leakage from the chamber between the seals will be out of the chamber. It will leak into the sealed area and to some extent it will leak into the airway. By applying active seals to all roadways would eliminate the egress of radon and keep it contained.

Passive seals on such an area would be subject to leakage. With high barometric pressure fluctuations there would be the potential that the pressure inside the waste could rise above the pressure in the fresh air drive adjacent to it causing leakage to flow from the waste into the fresh air. An active seal would prevent any leakage by providing a higher pressure differential than that of the waste and leakage would only occur into the waste.

Implementation

In coal mines where this technique has been deployed, the balance chamber was filled with nitrogen. This would mean that the leaking atmosphere from the chamber would be nitrogen. This is done so that the goaf void remains inert. In the case of radon there is no need to use nitrogen, as radon is a noble gas and therefore the risks of explosion is non-existent, and unless there is reactive ground within the sealed area there is no risk of spontaneous combustion.

Building seal chambers and filling them with compressed air would create the higher pressure needed and any egress of the chamber's atmosphere that leaked into the intake air would be air.

If there was limited access to exhaust sides to create or maintain active chamber seals, then simple bulk-heads might be deployed. This would mean that the exhaust side of the sealed area would be subjected to the vagaries of the barometric changes and some leakage would occur. However, leakage into exhaust raises would be diluted by the ventilation current and would be away from personnel so that they would not be exposed.

Ore passes would need careful consideration. If they pass through the sealed area the piston effect would need to be assessed. Isolating them by sealing would require a bulkhead that could withstand the transient pressure load.

CONCLUSIONS

Although there are quite a few differences between coal and metalliferous mines, there are opportunities to utilise the innovations that each sector of the industry has developed.

Radon control is usually by ventilation dilution and as the mine extends the mine demand for air increases. As the mine develops and old sections of the mine are worked out, ventilation would still be applied to ensure that radon concentrations are controlled. By sealing these areas and ensuring that radon would not contaminate the airways adjacent to the seals, the ventilation demand may be lowered providing considerable savings to the operating budget. Even if the amount of air required remains static instead of having to increase; this will provide a cost per tonne saving to the mine.

Active sealing chambers supplied with compressed air can be utilised for this and provide positive pressure to control leakage.

The sealed volume for a metal mine will likely be much smaller than that of a coal mine and hence the leakage rate will be much less. It would be in the order of 100 litres/second without the use of active sealing. Active sealing would ensure that any leakage that occurs would always be into the sealed areas and into the exhaust air raise network.

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Psychrometry and Psychrometric Charts

The major requirement in mine ventilation work is that volume, mass flow, heat, energy and moisture content can be determined speedily. Further requirements are the rapid determination of relative humidity, dew point temperature, vapour-pressure. Many forms of charts are available which meet these requirements. But from all available types of charts it is felt that for ventilation surveys the original "Carrier" chart is still the most useful in construction in that with the commonly-used types of instruments (barometers and hygrometers) all data can be read off the charts directly without the use of intermediate co-ordinates. The disadvantage of the "Carrier" chart is however that it is constructed for use at one specific barometric pressure. This disadvantage has been overcome by the construction of various charts which cover a wide range of barometric pressures.



PSYCHROMETRIC CHARTS	
Chart No 1	Density chart
Chart No 2	Psychrometric chart for 80 kPa
Chart No 3	Psychrometric chart for 82.5 kPa
Chart No 4	Psychrometric chart for 85 kPa
Chart No 5	Psychrometric chart for 87.5 kPa
Chart No 6	Psychrometric chart for 90 kPa
Chart No 7	Psychrometric chart for 92.5 kPa
Chart No 8	Psychrometric chart for 95 kPa
Chart No 9	Psychrometric chart for 97.5 kPa
Chart No 10	Psychrometric chart for 100 kPa
Chart No 11	Psychrometric chart for 102.5 kPa
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Chart No 19	Psychrometric chart for 122.5 kPa
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Chart No 21	Psychrometric chart for 127.5 kPa
Chart No 22	Psychrometric chart for 130 kPa
Chart No 23	Sigma-heat chart, range 0-40°C
Chart No 24	Sigma-heat chart, range 20-60°C
Chart No 25	Enthalpy chart, range 0-40°C
Chart No 26	Enthalpy chart, range 20-60°C
Chart No 27	Moisture chart, range 0-40°C
Chart No 28	Moisture chart, range 20-60°C
Chart No 29	Water-vapour chart, range 0-40°C
Chart No 30	Water-vapour chart, range 20-60°C
Chart No 31	Compressed vapour chart

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Practices for conveyor roadway segregation and designation of escapeways in Australian underground coal mines

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ABSTRACT

Segregation of conveyor roadways is a ventilation practice that is being increasingly applied in Australia. It is largely driven by the legislated requirements in Queensland pertaining to separation of escape ways from the mine. This legal requirement was based on a recommendation from the Moura No. 2 Wardens Inquiry into the mine disaster that claimed 11 lives.

The report recommended “the introduction of a requirement for all underground mines to have one intake airway that is completely segregated from other parallel intake airways so as to provide two separate means of egress from the mine via an intake airway”. This recommendation seeks to assist mineworkers to escape from a mine after a fire by providing them with an airway that is free from smoke or contaminants. The concept of an airway being “completely segregated” is an ideal that is challenging to implement in practice when considering the effect of leakage.

The practice of belt segregation is concerned with the potential for fires in the belt roadway although there are numerous other potential fire sources in underground coal mines. A review of segregation practices has been conducted applying ventilation engineering principles. Four different scenarios have been analysed with regard to the effectiveness of the segregation stoppings in preventing potential fire contaminants migrating to other parts of the mine. The results show that in some cases the benefits of segregation to a person evacuating a mine in the event of a fire range from beneficial to detrimental.

This is influenced by many factors including the location of that individual in that particular mine. There is room for improvement in the design and implementation of escape ways in underground coal mines. Escape ways need to be planned and designated with regard to the potential sources of fire, pressure differentials between escapeways and the operational practicalities of maintaining the pressure differential between them.

Keywords: Escape, conveyor roadways, segregation, fires, emergency response

INTRODUCTION

The term “segregation” is used widely in the Australian underground coal industry today but this term cannot be found in any legislation relating to under-ground coal mining. The intent of this paper is to identify the origins, reason and purpose for segregation

and also to assess its practical effectiveness. It is worth noting that this paper is focused on the modelling of contaminants from fires on main conveyors and that there are other types of mine fires or “reasonably foreseeable events” that can cause the intakes of a mine to become contaminated. The reason for this is in response to the industry practice of belt road segregation as means of mitigating the risks to underground personnel in the event of a belt fire.

Maintaining separated intake escapeways is a legal requirement in Queensland and one that is often complied with by segregation of the main belt road from the main travel road. Mines inspectors ensure compliance with the legislation by enforcing the segregation of the belt road at individual sites by issuing directives to bring the segregation into compliance. For the mining operation the segregation of the belt road is often a nuisance as it limits access to the belt road. It can be a headache for the ventilation officer to control and maintain, as there is often operational requirements to breach the segregation stoppings for access purposes.

Mining operation invest considerable amounts of time and money in the installation and maintenance of these ventilation control devices. It is important that they are serving a purpose and reducing the risk to underground personnel. If they are not, then they are a waste of time and money and are providing a false sense of security to underground personnel and management at the mine.

HISTORY IN AUSTRALIA

The legal requirement for belt segregation started in the Queensland coal mining legislation in the 1970s. The Queensland Coal Mining Act 1925-1981 General Rules for Underground Coal Mines (Queensland Government, 1981) rr. 4.2 stated;

“(1) In an underground coal mine other than a mine existing at 1st July, 1978, provisioning shall be made for an intake airway other than a roadway containing a belt conveyor. This requirement shall apply to any part of such mine other than a panel or sub-panel where the method of working limits the number of roadways to less than three: Provided that in the initial development of a new mine the belt conveyor roadway may serve as the only intake airway for such time as is reasonably required to provide a second intake roadway.

(2) All belt conveyor roadways shall be segregated from other intake airways and from return airways”.

This rule was very specific about what was required and when it was required.

The current legislation in Queensland does not call for belt segregation. Section 296 of the Coal Mining Safety and Health Regulation 2001 (Queensland Government, 2001) calls for two

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intake escape ways to be established that are separated in a way to prevent any reasonably foreseeable event happening in one of the escape ways affecting the ability of persons to escape through the other escapeway. In Schedule 4 - Ventilation control devices and design criteria, the stoppings used for establishing separation are specified as being of substantial construction with no over pressure rating. This section of the regulations came about in response to a recommendation on p67 of the Wardens Inquiry Report for the Moura No. 2 Mine Explosion, which called for:

“the introduction of a requirement for all under-ground mines to have one intake airway that is completely segregated from other parallel intake airways so as to provide two separate means of egress from the mine via an intake airway”.

Despite the change in legal requirement the practice of belt segregation is often used to achieve the requirement for two intake escapeways.

It is also worth noting that s296 of the CMS and HR 2001 is part of Division 4 of Part 9 which deals with mine design and that it is an obligation of the Site Senior Executive and not the Ventilation Officer to ensure that this is in place. The Ventilation Officer is however usually tasked with the responsibility of ensuring this is compliant. This responsibility is often begrudgingly accepted, as it is a constraint that is in some cases a hindrance to the ventilation of the mine. It is not surprising then that the separation of escapeways is not planned and modelled to the same extent as the ventilation of production panels.

The Queensland Mines Inspectorate issued a Safety Bulletin titled, “Lessons of mine segregation must be applied” (Taylor, 2008). This Safety Bulletin explains the legislated requirements and the origin of these requirements. It also warns operations that non-compliance with this requirement will result in a directive to suspend operations being issued under section 167 of the Coal Mining Safety and Health Act 1999 (Queensland Government, 1999).

In NSW there is no requirement for two escapeways in intake air. Some mines have implemented a separate escapeway in intake air to part of the mine. The legal requirement for escapeways or means of egress is in cl 45 of the NSW Coal Mine Health and Safety Regulation 2006 (NSW Government, 2006)

(b) (iv) at least 2 means of egress from each production area or other part of the mine to the surface part of the mine so that, in the event of any roadway becoming impassable, another is always available.

INTERNATIONAL PRACTICE

USA

The United States has very prescriptive requirements on segregation of conveyor roadways. The principle that is applied is that no air from a conveyor roadway is allowed to ventilate a working face. Leakage through ventilation control devices is included in this requirement and results in the practice of ensuring the pressure in the conveyor roadway is always less than the pressure in the travel road. This is achieved in practice by the routine placement of regulators in the conveyor roadway and overcasts where necessary to dump conveyor roadway air to the return airway.

These requirements come from § 75.350 of the Title 30 Code of Federal Regulations (30 CFR) (Federal Government of USA, 2008), which include the following:

(a) The belt air course must not be used as a return air course; and except as provided in paragraph (b) of this section, the belt air course must not be used to provide air to working sections or to areas where mechanised mining equipment is being installed or removed.

(1) The belt air course must be separated with permanent ventilation controls from return air courses and from other intake air courses except as provided in paragraph (c) of this section.

The double fatality at the Aracoma Alma No. 1 Mine in 2006 after a conveyor fire filled the primary escapeway of a working section with thick smoke increased the focus placed on this section of the 30 CFR by MSHA.

South Africa

In South Africa the Regulations under section 98 of the Mine Health and Safety Act, 1996 (Act No. 29 of 1996) call for two means of egress from the mine (Government of South Africa, 1996). There is no requirement for the provision of two separate intake airways or for the segregation of belt roadways from intake air. The safety systems for dealing with mine fires in coal mines rely on:

- Emergency lifelines
- Establishing refuge chambers with borehole to the surface and small fan for fresh air supply
- Capacity to ream out refuge chamber boreholes for evacuation of personnel

United Kingdom

The UK contains similar legislative requirements to those required under the Queensland legislation but contains some more clarity on when the requirements are applicable. The Mines (Safety of Exit) Regulations 1988, Regulation 9 (Government of the United Kingdom, 1998) states the following:

Intake airways

9. The manager shall ensure that, apart from those persons who are going to or leaving their place of work at the beginning or end of a shift, not more than 50 persons are employed below ground in any part of the mine unless:

(a) there are two separate intake airways into that part of the mine which are connected only in such a way that in the event of a fire, transmission of the products of combustion from one airway to the other is prevented so far as is reasonably practicable; or

(b) there is one intake airway which is constructed of suitable fire resistant materials and is free, so far as is reasonably practicable, from the risk of fire.

In addition to this regulation there is a requirement for at least two means of egress to the surface, ie. an intake and a return.

DIFFERENT SCHOOLS OF THOUGHT

From looking at the different segregation practices used locally and around the world several different concepts emerge. These may

not be immediately clear so they are listed below:

1. No segregation of intake roadways
2. Segregation of the belt to prevent belt fire contaminants entering intake roadways
3. Segregation of the belt to prevent belt fire contaminants entering working faces
4. Provision of a separated intake airway for use as an escapeway

These four different approaches all have advantages and disadvantages and different levels of complexity with regard to implementation.

It is important to understand what is trying to be achieved before an appropriate arrangement can be adopted. Too often, it seems, segregation stoppings are installed purely from a compliance standpoint with little understanding or interest in the purpose or effectiveness.

MINE SCENARIOS

For the purpose of analysis of belt segregation four different mine layouts were used to model the effectiveness of the segregation. The first scenario, called Case 0 is a conceptual model and does not represent the workings of an actual mine. This was used so that any ventilation layouts and analysis results could be published without concern for confidentiality. The other three scenarios, called Case 1, Case 2 and Case 3 are based on the ventilation models from actual longwall coal mines in Queensland, Australia. For the purposes of confidentiality only the analysis results are published.

Methodology

Pressures

Pressure gradient plots were generated for each scenario. These display the relative static pressure in the mine roadway from the surface intake to the long-wall and along the return back to the main fans. The belt road pressure gradient was also plotted. The pressure gradients of any additional separated intake roadways were also plotted.

Contaminant Test

A 100ppm contamination was placed into the model inside the belt portal and then modelled to see where the contaminant would migrate throughout the mine. This test was applied to each of the scenarios and the results recorded. The models were then modified with all the segregation stoppings in the mine removed and the same 100ppm contamination test reapplied. This allowed the two results for the same mine to be compared. One set of results with segregation stoppings in place and one with the segregation stopping removed. This was used so the effectiveness of the segregation stoppings of the scenario could be measured. It is important to note that the numerical value of the contamination concentration in the results table is only relevant with respect to the 100ppm contaminant that was used for the test. It is primarily for comparison between models and between segregation and no segregation. For example, a 20ppm contamination in a primary escapeway may appear to be acceptable until you consider that if the contamination at the belt portal was 1000ppm then the

concentration in the escapeway would be 200 ppm. Table 1 displays the model results for each scenario with the segregation stoppings in place and also with the segregation stoppings removed.

Table 1. Contamination test - Modelled segregation stopping effectiveness

	Modeled Number of Segregation stoppings	Modeled Contaminant Concentration (ppm)					
		Belt Portal	LW face	Mains	Dev 1	Dev 2	Primary Escapeway Maximum
Case 0	157 0	100 100	14 31	76 28	16 30	14 27	18 or 4* 33 or 27*
Case 1	203 0	100 100	7 17	30 22	25 21	0 1	31 22
Case 2	106 0	100 100	53 53	15 1	1 7	1 0	6 31
Case 3	64 0	100 100	21 22	44 26	29 24	44 25	26 29

* Depending on which heading is classified the Primary Escapeway

The purpose of this test is to measure the effectiveness of segregation stoppings to reduce the spread of contaminants to other parts of the mine. No consideration has been given to the dynamic nature of a fire, the buoyancy and pressure differentials that are possible from an active fire.

Case 0

The ventilation layout for Case 0 is shown below in Figure 1.

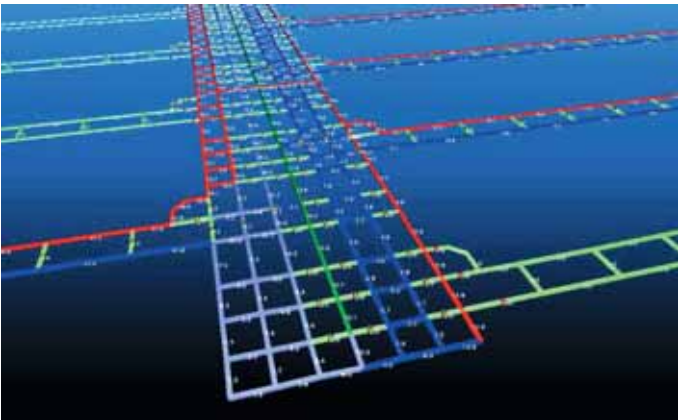


Figure 1. Case 0 - Ventilation layout

This fictitious mine consists of seven heading mains, 3 headings with flanking returns and a segregated belt road in the middle heading. As the mine does not exist there is not a designated primary escapeway. There are two results for the contaminant test for the primary escapeway shown in Table 1 depending on which set of intake roadways is adopted as the primary escapeway. The results show that with the 157 segregation stoppings in place the contaminant is directed predominantly into the mains development area with a concentration of 76ppm. The other three panels (including LW) modelled a contamination around 15ppm. The greatest benefit modelled with this arrangement of segregation was the contaminant concentration of 4ppm adjacent to the 76ppm in the mains. The 4ppm result came from the single

intake airway to the left of the mid-dle heading belt road in Figure 1.

The reason for the relatively low level of contamination can be seen in the pressure gradient plot for Case 0 in Figure 2. The heading that returned the result of 4ppm is referred to in Figure 2 as "Primary". This heading for the most part sat at a higher static pressure than the surrounding roads particularly the belt road where the contaminant was concentrated. This resulted in leakage paths flowing away from this "Primary" heading. The instances where the static pressure in this roadway drops below the belt road is due to the placement of segregated belt underpasses that were put into the model to allow for transport movements (operational requirement) and balancing of the intake airway pressures. It is this balancing that has caused the drop in static pressure to below that of the belt road in some instances. This could be mitigated in practice by the installation of machine doors at the segregated underpasses.

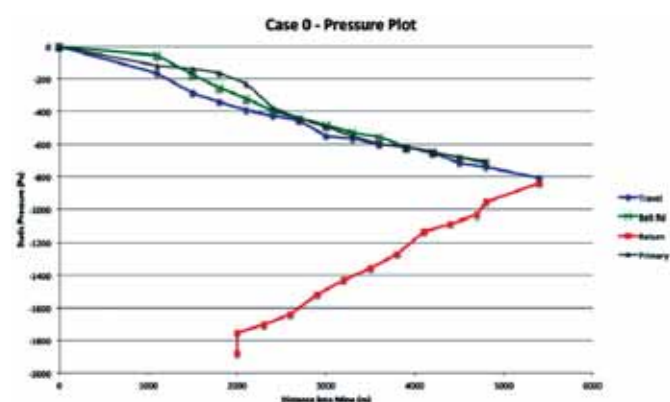


Figure 2. Case 0 - Pressure plot

The results in Table 1 also show that by removing all segregation from the model all inbye areas of the mine received similar contaminant concentrations of around 30ppm. This includes all working faces and escapeways.

Case 1

Case 1 is based on a longwall mine in the Bowen Basin in Queensland, Australia. The trunk conveyor of the mine is segregated on both sides from the surrounding intakes with 203 segregation stoppings. The primary escapeway of the mine is the main travel road. The contaminant test in Table 1 shows some very interesting and unexpected results. The highest contaminant results with the segregation stoppings in place were 31ppm in the primary escapeway, 30ppm in the mains development panel and 25ppm in one of the gateroads. Without the segregation stoppings in place the most significant result was the reduction of contaminant in the primary escapeway by 30% down to 22ppm. The longwall result increased from 7ppm to 17ppm without the segregation stoppings in place and the mains development and one of the gateroads both had reductions in the level of contaminant. As expected, the contaminant was more spread out and diluted without the segregation stoppings and more concentrated in particular areas.

The pressure gradient plot for Case 1 in Figure 3 shows the belt road at a higher pressure than the primary escapeway most of the time. The first 1000m the primary escapeway and the belt are in separate drifts so the leakage would be almost non-existent. The

time where the primary escapeway sits above the belt road in static pressure around the 2000m mark is due to a significant reduction in the number of main headings which causes the static pressure of primary escapeways to peak in this area. It is possible to see that the general pressure gradient trend of the segregated belt road is flatter than that of the primary escapeway causing the belt road to be at a higher static pressure than the primary escapeway. This results in the leakage of contaminant into the primary escapeway in the event of a belt fire. This scenario had 17 vehicle doors positioned along the trunk belt system. This highlights the need for vehicle and personnel access to the belt road and subsequent issues with quality of stoppings and leakage.

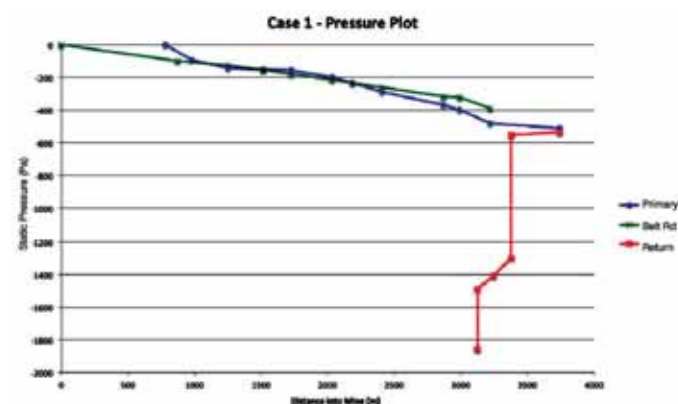


Figure 3. Case 1 - Pressure Plot

Case 2

Case 2 is based on a longwall mine in the Bowen Basin in Queensland, Australia. The trunk conveyor of the mine is generally segregated on both sides from the surrounding intakes with 106 segregation stoppings. The primary escapeway of the mine is the main travel road. This scenario involved the most elaborate layout for segregation of the belt road of all the scenarios analysed. Table 1 shows that the contaminant result for the longwall face is 53ppm regardless of whether the segregation stoppings are in place or not. The mains development panel showed a reduction of 15ppm to 1ppm with the removal of the segregation stoppings and another development panel showed a rise from 1ppm to 7ppm. The primary escapeway however showed a significant increase from 6ppm to 31ppm with the removal of the segregation stoppings. The pressure gradient plot for Case 2 shown in Figure 4 shows the extent that this particular operation has gone to get the pressure in the belt road to below the pressure in the primary escapeway (travel road).

The step in the pressure gradient for the belt road at the 2000m mark is due to the placement of a regulator and an air dump in the belt road. This does a good job to reduce the pressure of the belt road and largely prevents leakage of the contaminant into the primary escapeway. The infrequent instances when the belt road has a higher static pressure than the adjacent primary escapeway results in the low result of 6ppm. The air dump directs air out of the belt road into adjacent intake roadways. It is this air that is directed to the longwall and this is the reason for the 53ppm result for the longwall face.

Case 3

Case 3 is based on a longwall mine in the Bowen Basin in Queensland, Australia. The trunk conveyor of the mine is

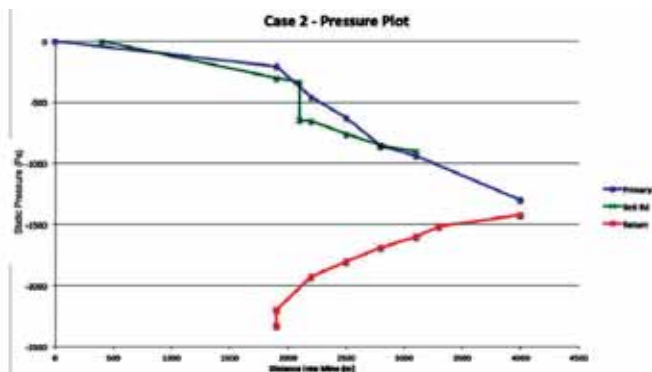


Figure 4. Case 2 - Pressure plot

generally segregated on both sides from the surrounding intakes with 64 segregation stoppings. The primary escapeway of the mine is a roadway separate from the main travel road and on the other side of the trunk conveyor. This scenario initially showed the most promise for having a simple design and maintaining a primary escapeway at a pressure above the adjacent belt road. The pressure gradient plot in Figure 5 shows that this is not the case. The belt road is generally at a greater static pressure than the adjacent primary escapeway. This is reflected in the results in Table 1 with arguable better results achieved with the segregation stop-pings removed from the model. The primary escapeway showed an increase in contaminant concentration from 26ppm to 29ppm with the segregation stoppings removed.

There are two ways this scenario could be dramatically improved. The primary escapeway loses significant pressure early in the mains due to a segregated belt underpass. This allows air to travel from the primary escapeway to the travel road. This could be easily corrected with the installation of a vehicle door. Additionally there is a 200Pa pressure drop in the belt road around the 2500m mark. This is due to a high resistance Ventilation Control Device (VCD) located in the belt road. This VCD would have very likely been installed to achieve compliance with belt segregation. The result is an increase in the static pressure of the belt road and the increased level of contamination of the primary escapeway in the event of a belt fire. In fact the primary escapeway suffers less contamination if this VCD is removed from the model. This is a good example of where compliance does not necessarily result in lower risk.

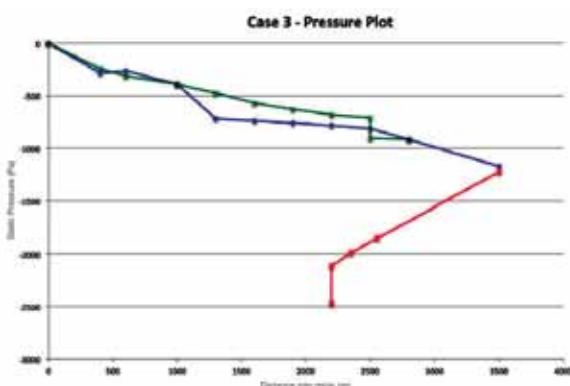


Figure 5. Case 3 - Pressure plot

CONCLUSIONS

A line of stoppings will not prevent contaminants from a belt fire entering a primary escapeway if the belt road is at a higher static

pressure than the primary escapeway. For the examples analysed, segregation of the belt road from all other roadways usually resulted in the belt road being at a higher pressure than surrounding intake airways.

Consideration needs to be given to the static pressure differential between separated escapeways. The only way to ensure that a contaminant does not enter the primary escapeway is to ventilate the mine such that the primary escapeway is generally at a higher pressure than the surround roadways. Ideally the primary escapeway should have the highest static pressure of any adjacent roadways.

Consideration should be given to establishing primary escapeways that are not the main travel road in the mine. This will allow for the following:

- Provide a primary escapeway for the full length of the main headings free from contaminants in the event of a fire in the belt road or travel road
- To better meet the requirements of s298 of the Qld CMS&H Reg 2001 for primary escapeways. Specifically "As far as practicable, free from the risk of fire"
- Ease of access to the belt road and less issues with damaged stoppings or leaking vehicle doors
- Improve early detection of heatings and small fires(ie. via smell)

Consideration should be given to putting more focus on reducing the level of risk to personnel than just being compliant. There are several examples from the modelling conducted where compliance is achieved but a more hazardous result is also achieved.

Based on the modelling work conducted the following observations have been made: In the event of a belt fire, segregation of the conveyor roadway will result in the smoke generated by the fire being concentrated in particular areas of the mine. This will usually be the mains but may be elsewhere, eg. Case 2 where the contaminant showed up greatest at the longwall face

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Energy saving optimisation of auxiliary ventilation systems in hard rock development headings

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ABSTRACT

This paper documents a theoretical investigation and design for reducing electrical energy consumption in development headings, while crucially maintaining the temperature, humidity and contaminant properties of the mine atmosphere at acceptable levels.

The proposed design replaces the traditional ventilation layout of a half level with a simple, branching network of glass reinforced plastic (GRP) ducting coupled to a single axial fan. This allows for a much lower delivery volume due to the fact that the fan now delivers fresh air directly to the working faces, thus removing as much heat as the larger volume of reused air does in the traditional system.

A variable speed drive will control the fans speed in order to maintain minimum standard ventilation volumes in each area as they extend into the virgin rock.

The combined effect of these design changes theoretically decreases total energy consumption by as much as 83% relative to the baseline.

Keywords: Energy, auxiliary ventilation, development, leakages, speed control

INTRODUCTION

Background and motivation

The present economic and energy climate has made it increasingly necessary, and potentially lucrative, to re-examine how the major power users of a mine might be improved or redesigned to achieve energy savings. Auxiliary ventilation in development headings accounts for a significant portion of a mine's total energy usage. Initial studies on the parameters governing these systems consistently found that as much as 75% of the typical energy consumption could be eliminated by reducing leakage, specifying smoother ducting, and increasing duct diameter. Reducing the speed, and thus the delivery and pressure of a fan in order to supply just enough air for temperature control also contributed greatly to theoretical savings.

A more detailed investigation and the eventual design required a practical awareness of the mining environment where these systems operate, that is, on half-levels.

Half level ventilation

A half level, as shown in Figure 1, consists essentially of a main

haulage development, an established crosscut leading to a raise- and winze-line (developing up and down the reef respectively), and a developing crosscut or travelling way.

Ventilation of the half level is achieved by a system of ducts and fans which force air to the blind haulages as they are extended into the virgin rock. This is done of course to provide personnel with a safe working environment and sufficient airflow to transport away the heat, dust and contaminants arising from drilling and blasting operations.

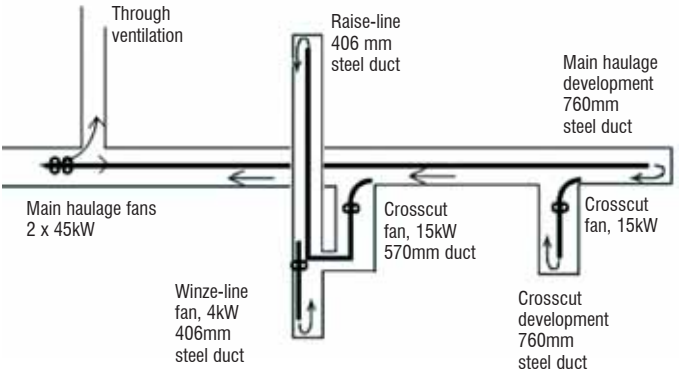


Figure 1. Ventilation of a typical half level

Note that each area has a dedicated fan and duct subsystem, and furthermore that the air is reused as it travels from the main haulage development to the crosscut, and then to the raise- and winze-lines, before rejoining with the through-ventilation of the mine. This process effectively concentrates heat and contaminants at each successive area, thus necessitating higher flow rates.

Typical flow rate and fan absorbed power by area are described in Table 1.

Table 1. Estimated flow rates and absorbed power of fans in a typical half level

Fan/Area	Fan volume m³/s	Rated power kW	Estimated absorbed power kW
Main haulage	10	90 (2 x 45)	77
Crosscut	5.5	15	13
Raise-line	5.5	15	13
Winze-line	2	4	3
TOTAL			106

Proposed design concept

The proposed design consolidates the system into a main duct, fabricated from fire retardant glass reinforced plastic (GRP), from which steel ducting branches off at the two Y-junctions, as shown in Figure 2. One branch ventilates the developing crosscut, while

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the other passes through the established crosscut and into the raise- and winze-lines, where the duct splits again.

A single 45kW high efficiency axial fan and variable speed drive (VSD) replaces the old 45kW fans at the intake. Regulators in the indicated locations will allow the air to be shared between the three working faces.

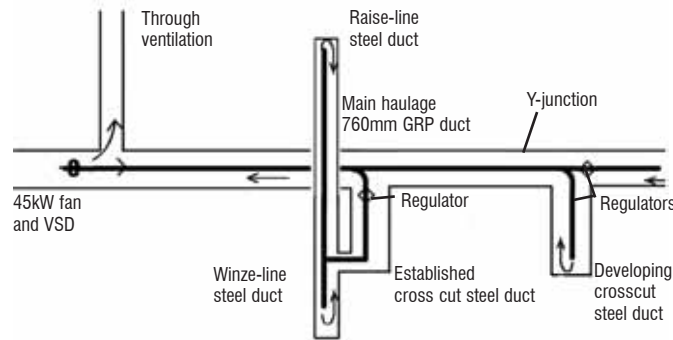


Figure 2. Proposed half level ventilation layout

Design criteria

Three design criteria define how much flow each area will receive in order to maintain a safe working environment. The chief criterion during normal operating conditions is a maximum wet bulb temperature of 29.5°C in any area.

During the so-called “re-entry period” after blasting, where no personnel are allowed underground, the volume must be such that the re-entry time does not exceed a stipulated value, in this case chosen to be 3 hours. The calculation method for re-entry is discussed in Simulation Methodology.

Provided these criteria are met, the lowest permissible flow in any area is limited to 0.15m/s per m² of face area (le Roux, 1990). Table 2 shows the resultant volumes of the third criterion.

Table 2. Required flows in the proposed design (0.15m/s/m²)

Location	Face area m ²	Required volume m ³ /s
Main haulage	10.9	1.63
Crosscut	10.9	1.63
Raise-line	4.3	0.65
TOTAL		3.9

THEORY

Ducted ventilation

A first principles approach (Meyer, 1998) determined which parameters out of diameter, friction factor, length, leakage and flow rate, have the greatest effect on fan absorbed power. A basic mathematical model, and later a more complex simulation model will make use of the most favourable and /or most practical of these parameters.

The relative effect on power absorption was determined by altering the following parameters one at a time through a range of selected values, while holding the rest constant:

- Duct diameter - D [m]
- Duct length - L [m]

- Atkinson's friction factor - k [Ns²/m⁴]
- Air flow - Q [m³/s]
- Fan static pressure - P [Pa]
- Air power - W [kW]

As a starting point, Atkinson's general resistance equation was considered. Firstly,

$$P = R \times Q^2 \quad (1)$$

where R is the system resistance:

$$R = (k \times C \times L \times \rho / 1.2) / A^3 \quad (2)$$

where C is the duct circumference, ρ is density.

Now, considering air power,

$$W = P \times Q \quad (3)$$

therefore,

$$P = (k \times C \times L \times \rho / 1.2 / A^3) \times Q^2 \quad (4)$$

and,

$$W = (k \times C \times L \times \rho / 1.2 / A^3) \times Q^3 \quad (5)$$

Friction factor and duct length therefore have a linear effect on air power. Flow rate has a cubic effect on power, as also described by the fan infinity laws.

$$W_2 = W_1 \times (Q_2 / Q_1)^3 \quad (6)$$

Diameter has an inverse effect on air power related to the fifth power - again supported by the fan laws.

$$W_2 = W_1 \times (D_1 / D_2)^5 \quad (7)$$

The relative effects of the parameters showed a high level of agreement with the illustrated trends in Figure 3.

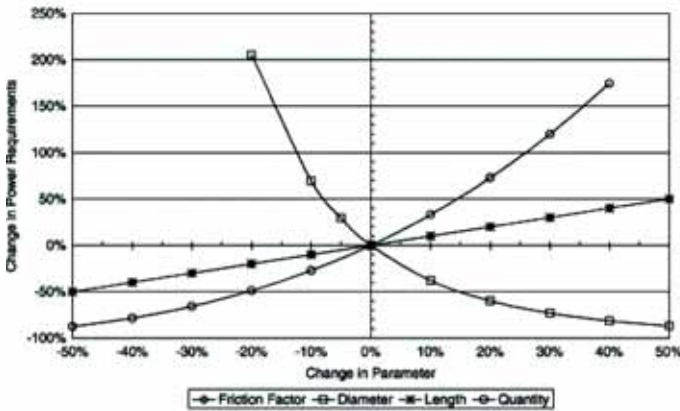


Figure 3. Parameter effect on fan motor absorbed power (Auld, 2004)

Mathematical model

A basic mathematical model created from these principles supported these findings. Notably, reducing the friction factor (k) from 0.0037 (typical steel duct) to 0.002 (smooth plastic duct) Ns²/m⁴ reduced the power usage in a consistently linear manner, in this case a reduction of about 45%.

Increasing the duct diameter improved power usage further still, with a change from the standard 760mm to 850mm resulting in a reduction of around 43%.

It was not intended for the study to be exhaustive, merely indicative, since the results matrix would have become very large, and because it was quickly identified that the most practically relevant parameters for an energy savings project are leakage, diameter and friction factor.

The problem of leakage was treated according to the principle illustrated in Figure 4.

When a certain discharge flow rate is required at the end of a duct, leakage works against this by necessitating a flow increase and a corresponding pressure increase, as shown. Essentially the fan must be specified to overcome the expected leakage and the pressure drop. So, if leakage can be reduced, a smaller fan may be chosen.

The model (which took into account the pressure and volume increase ratios as per the NCB leakage nomogram in Le Roux, 1990) showed that the high leakage percentage typical of many mines (up to 30%) was responsible for the majority of the system's losses and inefficiency. In one theoretical trial, reducing the leakage from 30 to 5% resulted in a decrease in absorbed power of 75%.

While such a radical improvement is unlikely in older steel ducting, plastic ducting has been shown to exhibit very low leakage.

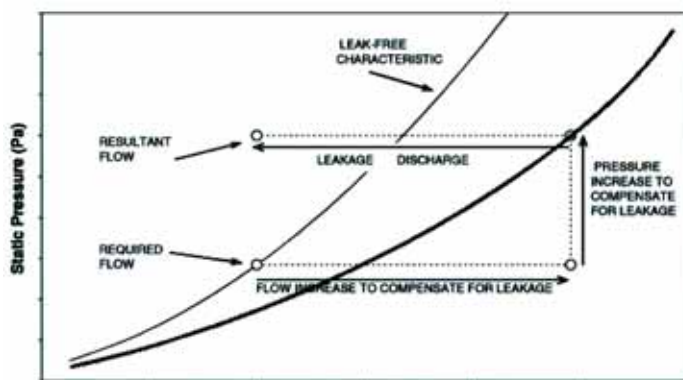


Figure 4. Leakage effect on resultant flow and pressure (Auld, 2004)

VUMA simulation

The promise of these potential savings led to the creation of a simulation model of the design concept (as per Figure 2) in VUMA, a programme capable of simulating complex thermodynamic and contaminant values alongside airflow.

Figure 5 shows the final VUMA model, with the haulages and ducts modelled as separate branches linked by periodic leakage paths. For the purposes of simplicity the raise- and winze-lines were consolidated and are referred to simply as the raise-line.

All input values had a high granularity, which helped to find the highest energy savings and efficiency, and the most cost effective solution.

The duct is branched, with three regulators positioned as indicated. During calibration of the model, the tuning of the regulators ensured correct sharing of the air between the three areas, according to the design criteria.

Haulage and development end heat sources, e.g. virgin rock temperature, rock drills, winches, electrical substations, locomotives

were included but for this particular model the inlet wet bulb temperature was set at 22°C, while in a real world trial the actual inlet temperature would obviously have to be used.

Modelling of the fan was done such that its volume output could be controlled as though by VSD. An earlier stage of design had identified this as the most effective means of delivering the correct (and lowest possible) airflow to match the three areas increasing demands during the half level life-cycle.

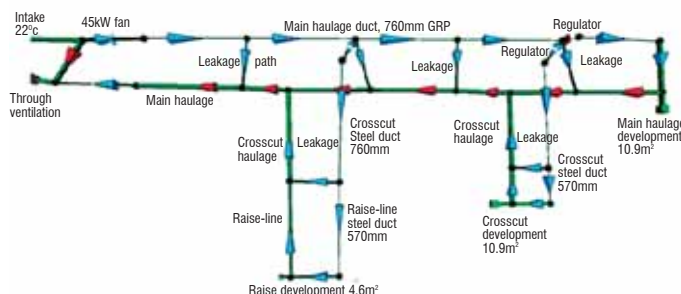


Figure 5. VUMA simulation screen capture

Simulation methodology - Volume flow

The duct length was increased at 30m increments at the main development and raise line, and 20m increments at the crosscut. This approximates a month's development per iteration.

The iterations began at a main haulage length of 400m and ended (after 8 iterations) at 640m, which is a typical length at which existing systems are dismantled and shifted down the main haulage.

For each iteration the VSD speed and regulator position were adjusted to provide just enough air-flow to ensure that the maximum reject wet bulb temperature was not exceeded. These results would reveal how often and to what degree these components would need adjustment during the life-cycle of the half level.

The fan static pressure, fan volume flow, face volume flow, and absorbed power were recorded. VSD percentage was manually defined and kept between 70 and 110%, outside of which typical VSDs become too inefficient (and at the higher end create the potential for motor burnout).

Simulation methodology - Re-entry time

Re-entry is essentially calculated by taking the excavated volumes of a given half level and dividing the results by the airflow serving each area. This time result must be multiplied by 8, to represent the 8 air changes which must occur before work may resume.

The re-entry is calculated as follows:

$$\frac{(\text{Excavated volume}) / (\text{face volume flow})}{(3600 \text{ seconds} / \text{hour}) \times 8} = \text{Re-entry time in hours.}$$

In existing systems, because the air is reused, the raise line is the last area to be cleared and thus dictates overall re-entry time.

In the proposed system, however, fresh air is delivered simultaneously to all three areas. Therefore the area with the highest re-entry time as per the calculation becomes the "bottle-neck" deciding the overall result.

RESULTS

Overview

The simulation procedure showed that a 45kW fan with VSD control and in-duct regulators is capable of delivering an adequate volume air throughout the life-cycle of the half level, and that this ensures the safe working environment as defined.

The 29.5°C_{wb} criterion was adhered to, as well as the minimum flow of 0.15m³/s/m².

Simulation iterations and power savings

Table 3 summarises the iteration procedure and indicates the power absorbed by the 45kW fan.

The minimum absorbed power is 15.5kW and the maximum 27.1kW. The average over the 9 month period is 18kW, which constitutes a reduction of 88kW, or 83%, compared to the existing system baseline of 106kW.

Table 3. VUMA simulation iterations, 45kW fan

Iteration	Working area	Length m	VSD %	Regulator Open %	Face Volume m ³ /s	Fan Volume m ³ /s	Fan Pressure Pa	Motor Power kW
1	Haulage X/cut Raise	400 20 60	75	15 25 80	1.8 1.9 3.0	8.1	1272	15.8
2	Haulage X/cut Raise	430 40 90	75	15 25 80	1.8 1.9 3.0	8.1	1272	15.8
3	Haulage X/cut Raise	460 60 120	75	15 25 80	1.9 1.9 2.8	8.0	1284	15.8
4	Haulage X/cut Raise	490 80 150	75	15 25 80	1.9 1.9 2.8	8.0	1275	15.7
5	Haulage X/cut Raise	520 100 180	75	15 25 80	2.0 1.9 2.4	7.9	1266	15.5
6	Haulage X/cut Raise	550 120 210	75	15 25 80	2.0 1.9 2.6	7.9	1282	15.5
7	Haulage X/cut Raise	580 140 240	80	15 25 80	2.0 1.9 2.6	8.0	1327	16.3
8	Haulage X/cut Raise	610 160 270	85	15 25 80	2.2 2.1 2.7	8.6	1635	21.5
9	Haulage X/cut Raise	640 180 300	90	15 25 80	2.4 2.2 2.8	9.1	1935	27.1

VSD and regulator control

The VSD percentage remained at 75% for the majority of the study, increasing to 90% over the last 3 months.

The change in speed could be managed with a built in schedule based on simulated predictions of volume requirement, or by feedback control with air flow instrumentation, or by manual adjustment of the set point.

The regulators needed little to no tuning. In order to reduce costs and complexity these should be manually adjustable. Adjustments

should be informed by airflow recordings at the rock face.

Future studies may investigate ways to optimise these components to further reduce resistance and thus absorbed power.

Leakage

Overall leakage was 20%, i.e. 2.2m³/s out of 8.2m³/s (average). This value is typical of steel ducting, but is most likely too high an estimation for plastic ducting.

Re-entry period

For Iteration 9, which represented the longest haulages, the raise-line and established crosscut proved to have the longest re-entry period at 2.65 hours (Table 4).

Table 4. Sample calculation of re-entry period

Area	Excavated Volume m ³	Flow rate m ³ /s	Re-entry hours
Main Haulage 1st segment	2180	7.4	0.65
Main Haulage 2nd segment	2180	4.6	1.05
Main Haulage 3rd segment	2616	2.4	2.42
Crosscut	1962	2.2	1.98
Raise-line and crosscut	3342	2.8	2.65

Note that the main haulage segments considered each have a different volume flow, which increase for each segment as the air drifts from the development heading back toward the fan.

The fan speed did not have to be increased to meet the re-entry criterion of 3 hours, meaning that the 29.5°C temperature criterion dominates the system for both normal operating times and re-entry.

PROPOSED INSTALLATION

Overview

The proposed system consists of the following:

- A760mm diameter main haulage duct constructed of ±3m sections of glass reinforced plastic, including two special “Y-pieces”
- Three louver-type regulators to fit the corresponding ducts.
- 760mm and 570mm steel duct sections. Supplied out of the mine's existing stock.
- A 45kW high efficiency axial fan.
- A 45kW VSD unit, PLC, and associated electronics, to be enclosed in a steel cabinet near the fan.
- Air velocity and wet-bulb temperature instrumentation for control feedback to the PLC.

Operation

The ventilation system will operate via a feedback loop consisting of the air velocity and temperature instrumentation and the PLC. Velocities will be measured at the three regulators, thus approximating the volume reaching each development heading, after taking into account an estimated leakage factor.

This does away with the need for sensors at the ends of the ducts, thus mitigating the risk of damage by mining activities (blast debris, vehicles, etc.) and from handling by personnel during duct extensions.

One temperature sensor must be placed in the hottest area (the first crosscut in this case) and linked to the same feedback loop.

Set point updates

The simulation showed that the regulators may be adjusted manually, with simultaneous volume checking at the face. This lowers the cost dramatically.

The PLC will make automated adjustments to the VSD such that the temperature and airflow as measured by the sensors (and adjusted for reasonable leakage) remains within preset tolerance bounds.

Installation

When an identified half level ventilation network is due to be shifted down the haulage, the new fan, VSD, PLC and a $\pm 400\text{m}$ length of GRP ducting will be installed, including the Y-junctions, regulators and velocity sensors. As the development grows (typically 3m/week) additional ducting sections will be added as normal.

Practicality in underground environments

The practicality of the system comes into question when considering the nature of the mining environment, particularly in the rough and ready conditions and the mistreatment of hardware for which development headings are notorious.

The VSD and air velocity sensors are particularly vulnerable to degradation in these conditions.

Also, since the airflow as perceived by the miners is reduced (even though cooling power remains unchanged) informal adjustments and even abuse of certain components is likely to occur. Personnel must be informed of the benefits of the new system and assured of safe working conditions.

Finally, the risk arising from dramatically reducing air delivery must be thoroughly mitigated before installation. This must be done through a detailed analysis of the half level, including the simulation of all physical parameters together with the VRT, heat loads, inlet temperature and operating machines.

CONCLUSIONS

The system

The proposed theoretical system is capable of providing fresh air at adequate cooling power to all the areas of the half level as defined in this study. The airflow can be adjusted automatically throughout the half level life cycle by means of the VSD and temperature and flow sensors, and is shared appropriately between working areas by means of relatively infrequent manual adjustments of the regulators.

Energy savings

By extrapolating the per-half-level savings of 88kW, a mine consisting of 100 similar half levels could realise a total energy saving of around 8.8MW.

As predicted, the main cause of reduction in power demand is not necessarily better ducting, but that the required volume of fresh air is so much less than the previously required volume of reused air, thus allowing all but one fan to be eliminated.

FUTURE WORK

Change management and training will become a major factor in the successful implementation and maintenance of the new design. Miners' concerns about cooling power and lower airflow will need to be addressed.

A pilot project is being planned and should reveal the actual energy savings potential, as well as the survivability of the system in adverse operating conditions.

An algorithm to optimise the balance between VSD speed and regulator position would further enhance the energy savings.

The durability of GRP ducting translates as a higher reusability over multiple development end life-cycles, which should mitigate the higher initial cost. A detailed study on this aspect is called for.

ACKNOWLEDGEMENTS

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Comparison of diesel particulate matter ambient monitoring practices in underground mines in Australia, the United States and South Africa

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ABSTRACT

Exposure to the microscopic particles in diesel engine exhaust can lead to serious health problems including incidence of cancers, heart disease and increased susceptibility to respiratory ailments of pneumonia, bronchitis, and asthma. Different ambient monitoring approaches being used in Australia, the United States and South Africa are examined. The options for the treatment and reduction of diesel emissions have become a major area of concern for many mine operators. The study compares various DPM ambient monitoring practices used currently in underground mines and measurements undertaken in the three countries are examined. DPM monitoring approaches have been available for some time based on shift average measurement practices but these have limitations in leading to understanding of DPM levels over short time periods. Real time monitoring produces data that is required for engineering evaluation exercises and often highlights mine situations where DPM levels are relatively high for substantial time periods.

Keywords: Diesel particulate, monitoring, exposure, best practices, underground mines

INTRODUCTION

Exposure to the microscopic particles in diesel engine exhaust can lead to serious health problems including the incidence of cancers, heart disease and increased susceptibility to respiratory ailments of pneumonia, bronchitis, and asthma. Different diesel particulate matter ambient monitoring approaches being used in Australia, the United States and South Africa are examined. The options for the treatment and reduction of diesel emissions have become a major area of concern for many mine operators. The basis for any complete Diesel Particulate Matter (DPM) compliance strategy should be a comprehensive baseline study of the DPM present in the mine atmosphere including ambient air monitoring, analysis of monitored data, and development of a realistic plan for ambient DPM reduction. It is important that studies are taken on a real time basis to allow important sources of DPM in the mine atmosphere to be prioritised.

NIOSH has been closely involved in development of instruments

for measurement of airborne DPM for more than 20 years. The earliest approaches focused on shift average determinations with development of the SKC approach. Two real time DPM monitors have been developed since. The first, the D-PDM was developed on the base of the successful Personal Dust Monitor (PDM) unit. The heart of the PDM is a miniaturised direct mass measuring sensor that measures mine dust. Changes were undertaken to the PDM (Gillies and Wu, 2008) to convert it to a DPM particulate submicrometer real time monitoring underground instrument which was named the D-PDM. The real time DPM unit continually reports levels of mine atmosphere submicrometer aerosol. The D-PDM results have been correlated with parallel SKC system DPM evaluations (Gillies, 2011). A phase of robustness and engineering testing has been undertaken to ensure the instrument can effectively assist mine management.

Another real time DPM measurement instrument, the FLIR Airtec, became commercially available in 2011 (Janisko and Noll, 2008; Noll and Janisko, 2007). It measures the Elemental Carbon (EC) component of DPM by a laser scattering approach. Both new instruments have been evaluated underground in robustness and reliability testing in coal and metal / non-metal mines.

The study compares various DPM ambient monitoring practices used currently in underground mines in Australia, the US and South Africa. Approaches from the three countries are examined. DPM monitoring approaches based on shift average monitoring have been available for some time but these have limitations in leading to understanding of DPM levels over short time periods. Real time monitoring produces data that is required for engineering evaluation exercises and often highlights mine situations where DPM levels are relatively high for substantial time periods. There are limits to the tools available to improve mine face conditions. One of these is increasing airflow ventilation in the working area. Another is to carefully control position of miners to upstream of working filters designed to capture DPM.

Modern large mines use hundreds of diesel vehicles. Real time DPM monitoring allows the industry to pinpoint high exposure zones such as those where various vehicles work in areas of constrained or difficult ventilation. Identification of high concentration zones allows efficient modification of local mine ventilation, operator positioning, work practices and introduction of exhaust filters and other engineering tools to reduce exposures.

AUSTRALIAN DEVELOPMENTS

In addition to personal exposure monitoring the real time D-PDM monitor has in recent years been used in many mines and educated operators in the control of their environment. The monitoring approach has application to all forms of diesel

powered mining. With its real time atmospheric monitoring ability, the D-PDM monitor has demonstrated that it can be used as an engineering tool to pinpoint high DPM exposure zones such as LW face moves or on development faces using diesel haulage cars. Isolation of high DPM concentration zones allows efficient modification of work practices to keep miner exposure within shift length exposure standards.

Ventilation considerations in handling DPM

One point of high DPM generation that is found in coal longwall (LW) moves occurs once every 12 to 14 months for a period of up to a month. The move relies on use of high powered shield transporters that produce high levels of exhaust pollutants of gases and DPM. Many mines find it a challenge to meet DPM recommended or target limits during all phases of operational moves. Issues that should be considered in optimising design of strategies to minimise atmospheric DPM include:

- Maximise air quantity where LW face equipment recovery, movement or installation occurs.
- Have all moving equipment (at least loaded machinery) travel in opposite direction to air flow.
- Ensure that air velocity exceeds machine speed to ensure a plume of exhaust does not hang over travelling equipment.
- Have parallel transport roads so that movement occurs in a circuit of loaded machines travelling inbye on one road and outbye on a parallel one.
- Ensure that miners are working upstream of machinery and particularly machinery that is working on faces loading or unloading.
- Divide available air so that majority is passing along the headings used by loaded machinery.
- Monitor DPM with real time instruments so that points where "Target" limits are not being met are identified and improvements are made during current LW move or planned for the next.

Mine atmosphere measurements of DPM in Australian mines have occurred systematically since the early 2000s. Regulatory guidelines are starting to emerge in Australia and the individual states are generally moving to acknowledge DPM metal / non-metal mine limits in use in the US of 0.2mg/m³ full submicrometer particulate matter, 0.16mg/m³ TC particulate and 0.10mg/m³ EC particulate. Examples of coal mine measurement real time DPM measurement are shown.

Monitoring of diesel particulate matter using the D-PDM

Mine A

Results from DPM monitoring using D-PDM instruments are shown from three coal mines. Mine A examined one 2.5 hour period as a 37 tonne dozer was brought in to pull the first shield on recovering a LW face as shown in Figure 1. About 50m³/s of air was measured on the LW recovery face. Between 14:45 and 15:32, the dozer attempted to pull out the first shield but was unsuccessful. It worked hard much of the time at maximum engine power.

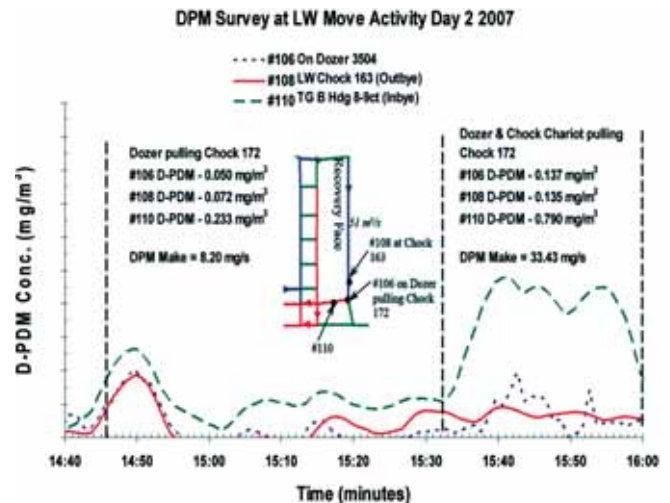


Figure 1. Submicrometer DPM in LW recovery face pulling shield

Between 15:32 and 16:00 a shield was chained to the dozer and together they successfully pulled the first shield while working hard. A general observation on LW moves was that some high aerosol readings were recorded due to the large numbers of diesel activities in working sections of the mine. This was contributed to by frequent vehicle movements or traffic jams. Miners should not be placed working inbye heavy vehicles working very hard, such as the dozer when pulling shields. For the LW move routes it is best if vehicle travels against airflow direction.

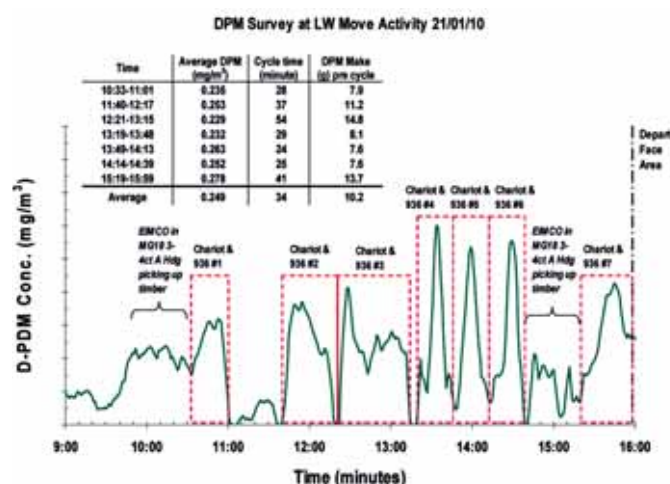
Mine B

Mine B monitored a highwall mine with no underground mains headings. Ventilation quantity was high. The main diesel activities were at the LW installation face. A total of seven Chock Shields were installed during the survey period as shown in Figure 2. For the seven peaks or higher levels of DPM, cycle time and DPM make were identified. The DPM makes varied from 7.6 to 14.8g / cycle with individual cycles time ranging from 25 to 54 minutes. This compared well with other mines' data. For example LW moves in one neighbouring mine showed DPM makes ranging from 3.0 to 22.4g / cycle and cycle times from 16 to 29 minutes for operations of Shield chariots (arrived, unloaded shields and departed) and EMICO 936 (into face, repositioned shields and out of face). The short cycle times in the other mine were due to the chariots only needing to travel half the length of the LW panel.

Mine C

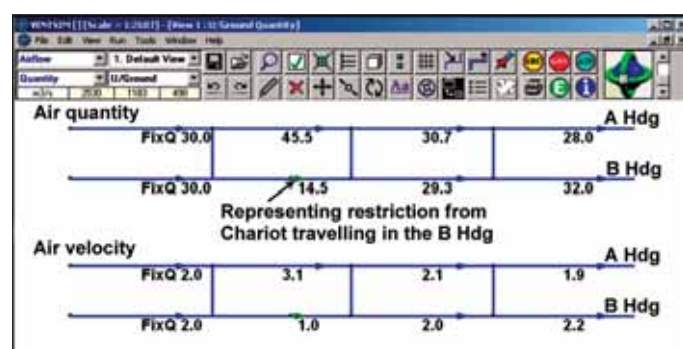
During LW Move real time DPM surveys at Mine C, one of the D-PDM units was placed on board a shield chariot to identify the exposure levels of operators. Significant DPM levels were recorded especially when the chariot was travelling in the new LW panel tailgate (TG) B Heading with chock loaded. The high DPM level exposure of the chariot operator in B Heading is contributed by the following:

- Chariot was working under load thus more exhaust generated.
- Chariot was travelling in the same direction as the ventilation air flow thus reducing the effective air velocity over the engine exhaust.
- Chariot was blocking much of the cross-sectional area of



B Heading thus increasing airway resistance and forcing more airflow through A Heading and as a consequence leaving less air available to dilute chariot exhaust.

A computer ventilation simulation (Ventsim) model was created to demonstrate the last point with chariot travelling in tailgate B Heading. Figure 3 shows the effect of the chariot in B Heading on the ventilation air split between A and B Headings.



A total of 60m³/s was available in the TG between A and B Headings and it was assumed that air split evenly between A and B Headings. A restriction of 67% of the cross-sectional area in the B Heading by the chariot loaded shield was assumed. This restriction had reduced airflow in B Heading from 30m³/s down to 14.5m³/s and air velocity from 2.0m/s to 1.0m/s. It should be noted that actual air split between A and B Heading near the installation face was measured at 28 and 32m³/s for A and B Headings during the surveys.

A chariot took one hour to travel from B Heading near the Recovery Face to the Installation Face (over about 3.0km (and thus at an average speed of 0.83m/s). Therefore, a relative velocity of 0.17m/s (about 2.5m³/s with 14.5m² area) across the chariot's engine exhaust can be calculated. The small amount of available air had caused buildup of DPM around the chariot while travelling in by carrying a shield which was evidenced by the high exposure level measured by the D-PDM unit on board.

UNITED STATES DEVELOPMENTS

The US DPM Personal Exposure Limit (PEL) was adopted in May 2008. This rule limits the Total Carbon (TC) on a shift average basis to 0.16mg/m³. Research is continuing by NIOSH on what is an equivalent or acceptable PEL limit in terms of EC. The Airtec DPM monitor measures the EC component of DPM by a laser scattering approach (Nolland and Janisko, 2007). Results from the Airtec can be compared directly with SKC system DPM evaluations. Both the D-PDM and Airtec new instruments have been evaluated underground in robustness and reliability testing in coal and metal /non-metal mines. Two Airtec units were used and results were compared against SKC method.

A DPM survey was conducted at Mine D, a metalliferous mine, during mucking. Downstream and upstream underground stations were selected to install real time Airtec units. Two SKC units were also installed next to the Airtec units to measure time weighted average DPM concentrations. Monitoring was conducted for 5.5 hours. Vehicle passing the points were monitored; data logged consisted of time, vehicle type and direction of travel. It was noted that almost 65 equipment units passed during the duration of monitoring. High EC content was observed whenever there was high vehicle frequency. The Airtec instrument which was installed downstream showed overall higher EC values compared to the other one upstream. The Airtec instrument which was installed upstream failed after 4.2 hours. There was a junction in between both measuring points. Air quantity downstream was $84.5\text{m}^3/\text{s}$ and upstream was $79.8\text{m}^3/\text{s}$.

Table 1. Elemental carbon, organic carbon and total carbon measured by SKC NIOSH 5040

Mucking	EC ($\mu\text{g}/\text{m}^3$)	OC ($\mu\text{g}/\text{m}^3$)	TC ($\mu\text{g}/\text{m}^3$)
Downstream	130	42	170
Upstream	97	27	120

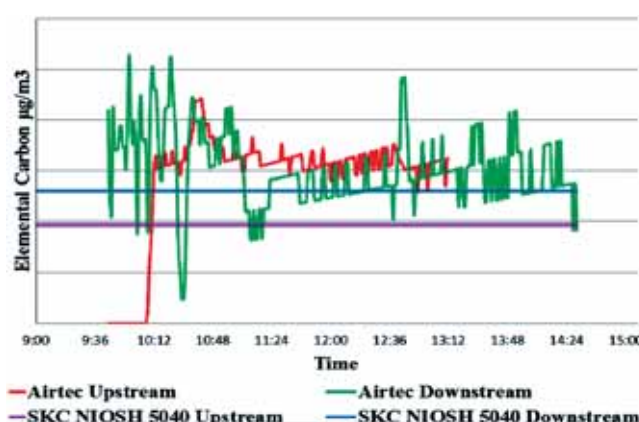


Figure 4. Time versus real time EC by Airtec and SKC

The peak value of EC observed at the upstream Airtec was 0.220mg/m³ after 1.7 hours. The peak value of EC observed at the downstream Airtec was 0.263mg/m³ at 0.95 and 1.2 hours. Table 1 gives the EC, Organic Carbon (OC) and TC by SKC. Figure 4 shows comparisons between each of the Airtec and SKC units for upstream and downstream stations.

Front end loader cab tests

The second set of DPM measurements at Mine E, a metalliferous mine, was based around Front End Loader (FEL) activities during a normal working operation. The aim was to check the effectiveness of the FEL operator's cab for decreasing DPM exposure and this was achieved by measurement at locations both inside and outside of the cab. Monitoring was conducted for 285 minutes including 90 minutes idle time (break time) near the middle. From 9:40 to 13:06 FEL was working inside the mine at two locations. Airtec and SKC DPM monitors were used at both measuring stations and placed inside and outside the cab.

A count was made of the number of 300kW powered haul dumpers being loaded by a FEL. Other equipment passing by the FEL was noted. A total of three different dumpers were involved in mucking operations. During the measurement period, 42 dumpers were loaded by the FEL. Table 2, below gives the Elemental Carbon, Organic Carbon and Total Carbon measured by the SKC.

Table 2. TWA, Elemental carbon, organic carbon and total carbon measured by SKC NIOSH 5040

FEL Cab	EC (µg/m³)	OC (µg/m³)	TC (µg/m³)
Inside	85	5	85
Outside	300	150	450

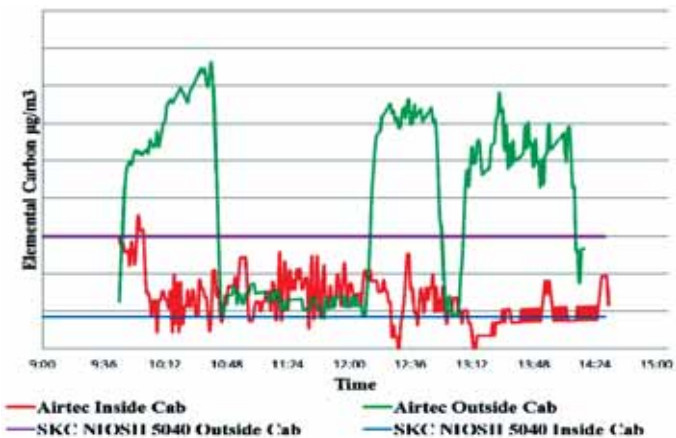


Figure 5. Time versus inside and outside real time Airtec EC and SKC measurements

Airtec Inside the Operator's Cab: The Airtec which was installed inside the operator's cab shows quite high EC values. Before break time the peak EC was 0.352mg/m³. Even during break time the EC content was higher than 0.1mg/m³. The reason for this is the movements of vehicles in the workshop during lunch time, as the dumper was parked near the underground shop during break time. On resuming the loading activity, the EC values increased again. Very high, consistent and alarming EC values were observed. During most of the measuring time the EC values are consistently higher than 0.1 SKC instruments installed inside FEL operator's cab is shown in Figure 5.

Airtec Outside the Operators' Cab: Before break time almost all EC values on the Airtec were above 0.5mg/m³, which is much higher than MSHA prescribed value. During break time (10:50 to

12:20) the EC values range from 0.08 to 0.167mg/m³. This is due to the vehicles movement in the shop during lunch time as the FEL was parked near the underground shop during break time. On resuming the work at about 12:25, the EC values raised again. The value was consistently higher than 0.6mg/m³. At about 13:06 the loader has moved to another location, Section 9 Undercut for loading and the EC values has decreased rapidly to about 0.1mg/m³. Once loading operation started at the new location, EC value has increased rapidly. These values are found to be higher than 0.4mg/m³ all the time and once peak at 0.685mg/m³. EC versus time graph for both Airtec and SKC which were installed outside the FEL operator's cab is shown in Figure 5.

Diesel powered drilling jumbo tests

A third set of DPM measurements were taken at Mine F, also a metalliferous mine. A 224 kW jumbo drill was measured during its normal operation. The aim was to measure DPM output from jumbo drilling operations and also to check the effectiveness of the jumbo drill operator's cab in reducing DPM content by taking atmospheric measurements at locations both inside and outside the cab. Monitoring was conducted for almost 4 hours inside and outside of the operator's cab. As before Airtec real time DPM monitors were placed with SKC time weighted average units.

During the monitoring period any additional activity of diesel vehicles or machinery in the area were observed and noted. One 97kW diesel powered hydraulic mechanical scalar (Getman Model S330) was also working during the whole monitoring time in the nearby entry or at another face.

Table 3. TWA, Elemental carbon, organic carbon and total carbon measured by SKC NIOSH 5040

Jumbo Drill cab	EC (µg/m³)	OC (µg/m³)	TC (µg/m³)
Inside	6	6	10
Outside	470	230	700

Table 3 gives the EC, OC and TC recorded by SKC units. Airtec inside the Operators' Cab gave very much lower EC values (Figure 6). The values were very low and sometimes not even measureable. During most of the measuring time, the EC value was found to be less than 0.06mg/m³ and it is apparent that the cabin reduced DPM content significantly. The consistency of trend needs to be tested by more measurements.

An Airtec monitor was installed outside the drill jumbo. These recorded very high, consistent EC value. The value of EC increases gradually during drilling and reached up to a maximum of 1.06mg/m³ which is high. This particular peak in the plotted graph was due to the movement of one Load Haul Dump machine which was loading and hauling the broken rock material at a nearby face. During most of the measuring period EC values were found to be more than 0.4mg/m³ which indicates the need for serious remedial measures. Figure 6 shows a comparison of readings from Airtec and SKC units for inside and outside stations.

SOUTH AFRICAN DEVELOPMENTS

After the recent reclassification of DPM as a Group 1 Human

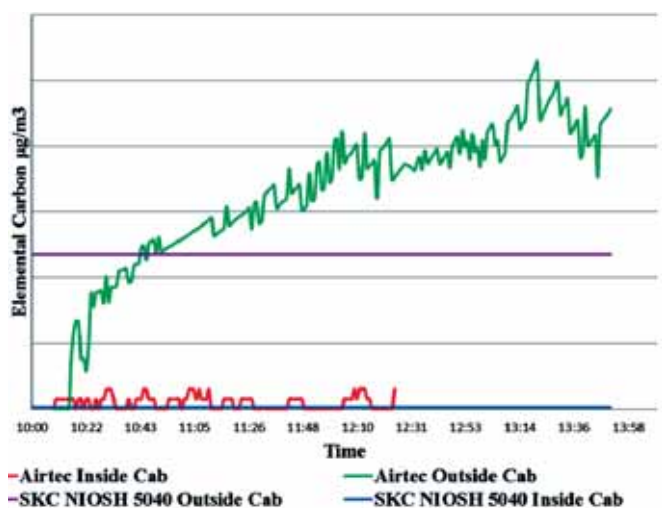


Figure 6. Time versus real time EC by Airtec and TWA EC by SKC

carcinogen by the International Agency for Research on Cancer in June 2012, efforts to establish an Occupational Exposure Limit (OEL) is becoming a necessary reality.

The use of diesel engine locomotives in South African mines can be traced to Van Dyk Consolidated Mines Ltd on the Witwatersrand gold mines in 1928 as a replacement for battery locomotives (Belle, 2008). The advantages and disadvantages were recognised in those days, and surprisingly, there are no significant additions to this list, but only refinements. The recognised advantages were minimum installation costs, high mobility and greater power. The disadvantages were heat input into the air, noxious gases exhausted into the air, danger of explosions (in coal mines) and fires. The mining regulations at the time required that the proportion of CO and CO₂ should not be more than 0.01% and 0.1% respectively. This translated to a dilution factor of 0.017m³/s/kW for the diesel engines used at the time. Unlike gold and platinum mines which are generally at low levels of diesel mechanisation, coal mines use large numbers of diesel vehicles for transportation, materials handling and other support operations like LW and bord and pillar section belt moves. In recent years, small diesel vehicles are commonly used for worker transportation from surface to underground. DPM research in the 1990s was focused on exhaust and control measures in underground workings (Haase, Unsted and Denysschen, 1995; Unsted, 1996). Advances in DPM measurement technology have resulted in OELs being set in some areas such as for metal mines in the US.

Considerable research on DPM has been conducted in Australia, Canada, Germany and the USA. In some Canadian provinces respirable combustible dust (RCD) has been used as a measure of the exposure of miners to DPM. RCD is that portion of a respirable dust sample (collected with a cyclone preseparator) that can be burnt off a silver membrane filter when exposed to a temperature of 400°C for two hours. The filter is weighed before and after heating with the amount of material being lost deemed RCD.

NIOSH researchers in conjunction with Sunset Laboratories developed a thermo-optical technique for the analysis of the EC fraction of DPM, which is now the commonly accepted analytical method - known as NIOSH Method 5040 (first published in

1999). CSIR in South Africa is the only laboratory facility that carries out DPM analyses in accordance with the NIOSH 5040 method.

South Africa occupational exposure limit position

Currently there are no regulatory mechanisms (such as OELs) that specifically address DPM limits. No mention is made of DPM in the regulations of either the Mine Health and Safety Act or the Occupational Health and Safety Act. Employers are obliged to conduct risk assessments on all hazards that may affect the health and safety of employees and initiate appropriate risk mitigation measures. There are references that some metal mine ventilation designs consider 0.12m³/s/kW dilution rate to manage and dilute exhaust and particularly gases produced. A value of 2mg/m³ has been generally accepted as being an appropriate OEL for DPM (Unsted, 1996).

DPM research into exposure to diesel engine emissions has been primarily through the Mine Health and Safety Council, the Chamber of Mines Research Organization or privately funded activities of mining houses into exposure levels (base lining) and control methods. A DPM study (Gen 010) by Haase, Unsted and Denysschen (1995) concluded that the critical components of diesel exhaust emissions were NO₂ (peak exposures) and DPM (time averaged exposures). It was stated that DPM should be considered critical in the light of the stringent Threshold Level Values that had been announced at the time by the American Conference of Governmental Industrial Hygienists. It was found that DPM emissions were sensitive to engine maintenance and it could be conveniently measured with gravimetric dust samplers. Also of note was that operators of diesel-powered vehicles and persons in headings with low or no ventilation experienced the highest exposure levels.

In a DPM study (Gen 208) by Unsted (1996), it was noted that size selective sampling would not be appropriate due to the large fraction of submicrons particulates that would conceal any DPM emissions, which are mainly submicrometer in size. The project concluded that there was no technical justification to advocate the use of low emissions diesel fuel, as emissions levels could be better and more effectively controlled by an engine maintenance programme than by a change in fuel formulation. However, it is important to note that DPM definition was not known at the time (NIOSH 5040) and mines used to measure gaseous component of diesel exhaust, which was stopped for unknown reasons with the promulgation of 1996 Mine Health and Safety Act.

There has been a SIMRAC research study which was attempting to estimate DPM emissions in coal and platinum mines. The study failed to provide clarity on the DPM sampling methodology used, for instance sampling flow rate, personal or area sampling or location of samples or if it indeed sampled DPM. For example, one of the data suggested a personal DPM value of over 4.5mg/m³ which is remotely unlikely. The reason for the disagreement is that if the DPM is of 4.5mg/m³, then the respirable dust would be of the value of 50mg/m³ and total dust would be in the order of 500mg/m³ which is not a possibility in a metal (gold or platinum) mine work area. Therefore, use of the research report or the data at its face value needs to be reviewed prior to perpetuating misleading and alarming levels of exposures.

With the impending legislation surrounding DPM in South Africa and in the absence of any previous work specific to DPM measurement, the above section of the paper discusses the ongoing South African journey of measurement and limitations of exposure data for ventilation planning and regulatory purposes. Unfortunately, discussion on setting up an OEL limit has been going on for over a decade to date.

In South Africa currently the Department of Minerals and Energy (DME) and Department of Labor (DoL) listings of OELs do not include an OEL for DPM but the DME is currently investigating this possibility. In Australia, currently accepted DPM limit is $0.1\text{mg}/\text{m}^3$ measured as EC to avoid the complicated TC/EC ratios associated with the measured DPM values (Belle, 2008, 2013). Figure 9 shows the complexity of these ratios highlighting differences for various mined commodities.

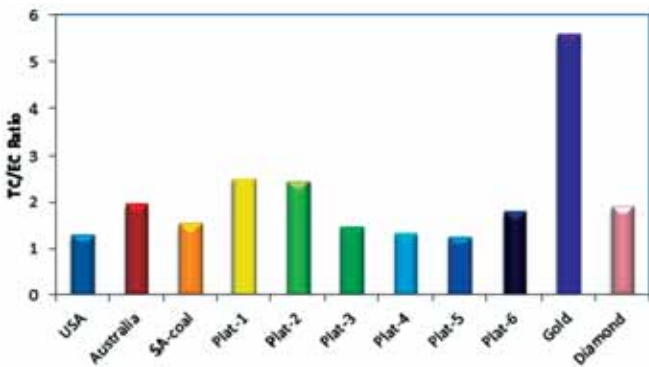


Figure 9. DPM TC/EC ratios in various mining commodities (Belle, 2013)

Monitoring of diesel particulate matter

Belle (2008) revealed that TC versus EC ratios measured in South African mines have shown the limitations associated with the adoption of international standards because the median TC / EC ratio for underground platinum mines in South Africa is 1.8, with a range of 1.2 to 5.8. Measurements in coal mines have shown a median ratio of 1.44, with a range of 1.25 to 2.13. Both of these commodity measurements differ from established international TC / EC ratios. This makes it difficult to duplicate existing international OELs to regulate DPM in South African mines. Further investigation may be necessary at a globe level to address the differences in TC / EC ratios and its use in setting up OELs.

Further to the adoption of an appropriate OEL, the tripartite working parties compiled a guideline for the preparation of a mandatory code of practice or guidance note on the use of diesel engines in all mines to address and manage health aspects (occupational exposure to diesel emissions).

Figure 10 shows a photographic view of the South African gold, diamond, platinum and coal mine DPM samples (soot coloured) prior to carrying out NIOSH 5040 analyses.

Figure 11 shows the typical real time coal dust and DPM level trends recorded during a belt move-shift (data is not adjusted for any correction factors) using PDR real time dust monitor. The DPM measurement data in coal mines indicate that in surface mines or under normal underground mining conditions, the DPM exposure is well below the overseas DPM compliance limits, unlike

underground belt or section or LW moves. The reasons for high DPM exposures can be attributed to the increased number of diesel operating engines, diesel vehicle conditions, engine maintenance and loaded engines during the belt moves and continuous idling of diesel engines underground sections. Also, the disruptions in section ventilation layout during belt moves may also have contributed to the high DPM exposure.

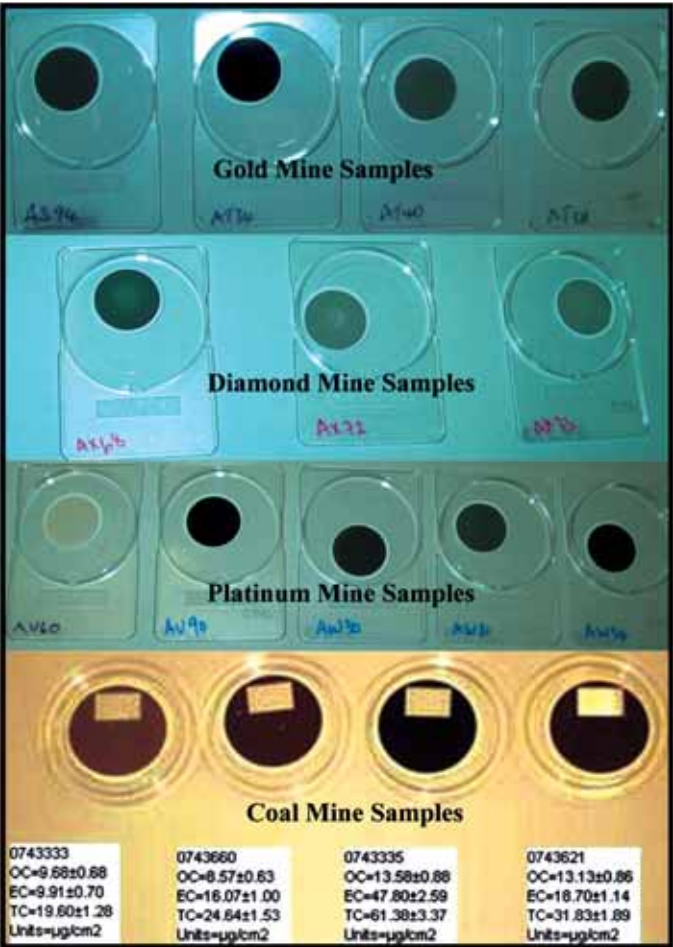


Figure 10. Various mine samples sent for NIOSH 5040 analysis (Belle, 2002)

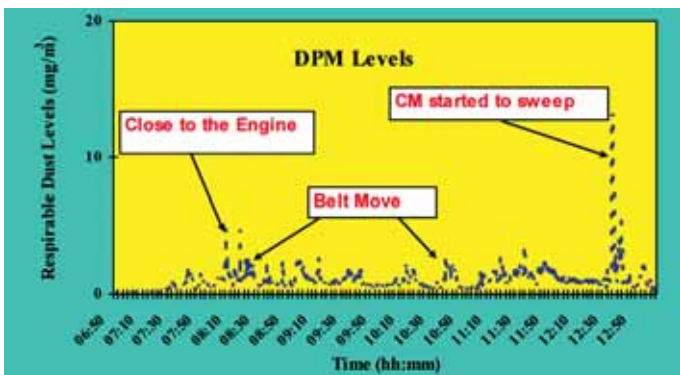


Figure 11. Typical real time DPM/Dust trend levels (unadjusted) during coal mine belt extension shift

It is understood that all of the diesel engines use water-cooled scrubber filtration systems, which do not have any readily available records of their maintenance. It was noted that all the coal mines use low-sulphur diesel fuel supplied by a central depot (up to

500ppm sulphur unlike overseas mines with sulphur levels of 10ppm). However, the validation of the sulphur levels in the supplied diesel to the mines could not be found in all mines. Overall, a comparison of the exposure data indicates that there is a significant difference in the range of DPM values between surface and underground mines. For example it was observed a section still contained measurable DPM levels even though there was no diesel equipment present during the normal coal cutting operations. As an example a Continuous Miner section where there was no LHD present during the shift had measured EC levels of 0.027mg/m³).

Currently in the absence in South Africa of regulated limits some local mining companies have adopted an EC DPM OEL of 0.1mg/m³ or 0.2mg/m³ sampled as the submicron fraction (TC). It is envisaged that adoption of continuous monitoring practiced overseas during diesel intensive and infrequent mining operations in South Africa would assist in effective management of DPM in mines.

SUMMARY AND CONCLUSIONS

Various DPM approaches, regulations and ambient monitoring practices currently used in underground mines in Australia, South Africa and the US have been discussed and compared. Some monitored results undertaken in the three countries have been examined. DPM monitoring approaches have been available for some time based on shift average monitoring. This approach has limitations in gaining a full understanding of DPM levels over short time periods. Real time monitoring produces data that is required for engineering evaluation exercises or to control effectiveness. Real time monitoring often highlights situations where DPM levels are relatively high for substantial time periods.

There are limits to the tools available to improve mine face conditions. One of these is increasing air-flow ventilation in the working area. Another is to carefully control the position of miners to upstream of working diesel machines. A third is to carefully introduce use of exhaust DPM filters designed to capture DPM and a fourth is to make use of electric equipment where practical.

Modern large mines may use hundreds of diesel powered vehicles. Real time DPM monitoring allows the industry to pinpoint high exposure zones such as those encountered where various vehicles work in areas of constrained or difficult ventilation. Identification of high DPM concentration zones allows efficient modification of local mine ventilation, operator positioning, work practices, introduction of exhaust filters and other engineering tools to reduce exposures.

Based on the findings of the study, it is concluded that real time DPM ambient monitoring practices in underground mines are gradually being accepted as an engineering tool to optimise the DPM control strategies in the mining industries of the countries under study. However, statutory monitoring still relies on shift average monitoring for determination of personal exposure levels.

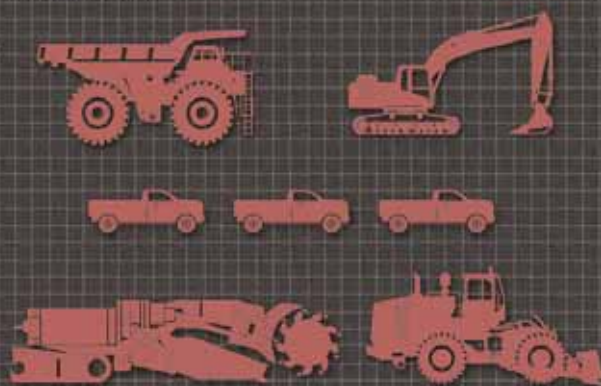
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Underground fires: *Prevention is better than cure*

Mentz Karsten, Mine Ventilation Consultant

INTRODUCTION

In the article by Biffi (2016), the comment is made: Through the years, the mine ventilation officer has been assigned additional “vicarious” responsibilities to provide protection and interventions in the event of fire incidents. The Cambridge English dictionary defines “vicarious” as: *experienced as a result of watching, listening to, or reading about the activities of other people, rather than by doing the activities yourself*. Another definition refers to it as: *taking the place of another person or thing; acting or serving as a substitute*. I fully agree with this comment by the authors in terms of the topic under discussion and will expand on a possible solution. However, what worries me just as much is the scarce knowledge of some mine ventilation professionals about fire management. There are instances where these professionals have “other vicarious” responsibilities assigned to them, (for example; ventilation construction, pure inspections, involvement in underground ablation facilities control, etc.) These odd jobs or simply put “other vicarious” responsibilities are in my view a major stumbling block in their path, preventing them from focussing on their real tasks as mine ventilation professionals.

The regulations referred to in the article require “competent persons” to perform tasks. This competency not only comes from experience and obtaining the Certificate in Mine Environmental Control, but also from further education in fire management.

FIRE INCIDENT STATISTICS

During 2002, I obtained permission from Mines Rescue Services to gather fire statistics from their records for use towards “Intake fire audits” which I was tasked to do at all AngloGold mines at the time.

Figure 1 illustrates the origin of all gold mine fires, which indicates the origin of fires in intake airways at all gold mines for the period

All AU Mines - Origin of Intake Airway Fires (1981 to 2002)

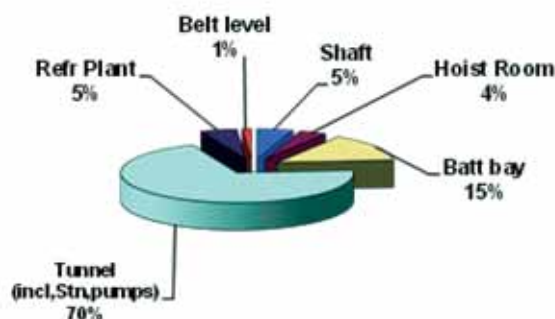


Figure 1. The origin of all gold mine fires

Response to article, Underground Fires: Prevention is better than cure. 2016 Volume 69, No 3.

in question. The same figure was drawn for AngloGold itself for comparative purposes.

Fires in intake airways obviously carry a higher risk, see figures 2 - 4.

With the permission of Mines Recue Services the industry data was updated and presented at the February 2012 MVS Conference - “Fire Detection and Control Rooms”.

Fires/Explosions - All Causes – 1981 to 2011

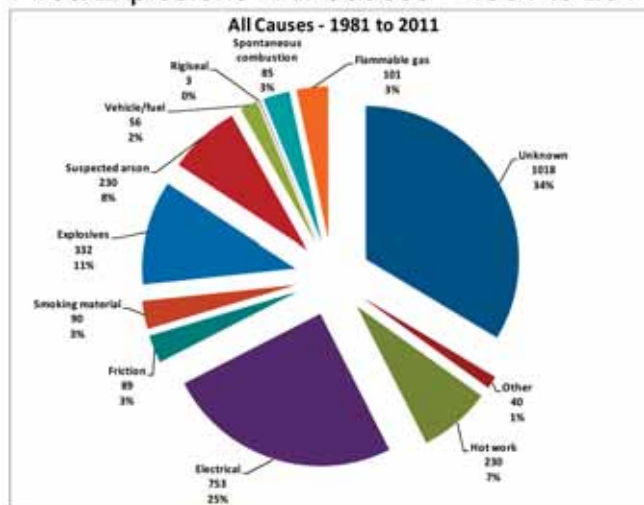


Figure 2. Fires/explosions all mines - all causes

Fire Fatalities – 1982 to 2012

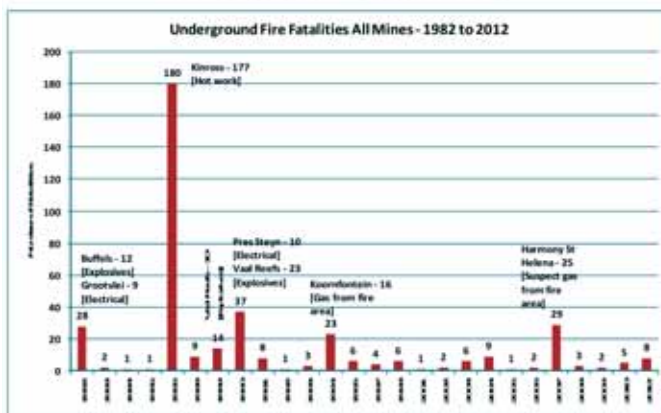


Figure 3. Fire fatalities all mines

The mine fire statistics can be manipulated and expressed in various formats for their best use at the particular mine. The main objective is to determine the risk to underground employees and loss of production and to develop mitigating control measures.

The point here is that mining houses should be aware of what is happening not only in industry, but also within their own company.

Gathering and analysing statistics and making it known, together

Fire Fatality Causes – 1982 to 2012

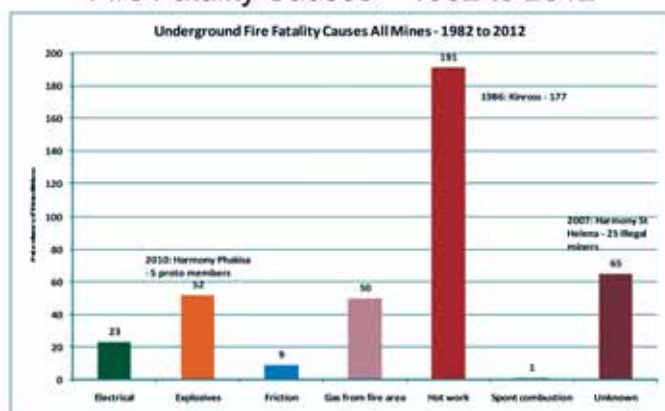


Figure 4. Fire fatality causes for all mines

with a mitigating action plan should be one of the tasks of a “competent person”.

FIRE HAZARD MANAGEMENT

The trick is to establish a system whereby management is made aware of deficiencies in control measures. These deficiencies should be measurable by indicating a degree [percentage] of compliance [to mine procedures, standards and codes of practice], as well as the current risk [high, medium or low]. These two indicators will determine the priority order of corrective action. The corrective action plan includes the person responsible and his / her date of completion. Once a decent audit protocol covering the compliance to the mine's Code of Practice for Fire Prevention has been drawn up, improvement / deterioration to compliance can be measured by repeated audits.

It is essential that line supervisors be made aware of and appointed for fire prevention- and control-related duties in their particular areas of responsibility. This could be built into their daily logbooks and by means of addendums and particularly the “Planned maintenance system” (specific to the engineering department) used by the mine.

SKILLS BASE BASE REQUIREMENTS

I agree with the authors insofar the basic knowledge that is obtained via the syllabus for the various Mine Environmental Control qualifications. However, advanced skills comes with more learned knowledge, experience and studies of incidents / accidents that occurred in industry.

In some cases, ventilation professionals never get sufficient experience on the mines at which they work. This experience is of course best obtained through active involvement during events, grasping the immediate and basic causes and assisting in drawing up the corrective action measures to prevent similar future events.

Advanced skills, such as having a well trained and experienced fire master, is imperative to any mining house in current operations as well as where new areas are being developed. As in any business the aim should be to attempt to engineer out possible risks from the start.

CONCLUSION

I agree with the conclusion by the authors, but would like to add / reiterate the following points:

- A specific person or persons of the particular mining house's ventilation profession should be earmarked for further education, for example;
Mines Recue Services course for “Managing a Control Room”.
Fire Protection Association of South Africa [FPA] course for Fire Prevention & Safety Strategies.
Fire Protection Association of South Africa [FPA] course for Advanced Fire Prevention.
- The same person[s] mentioned above should be exposed to / allowed to study previous fire events in industry and in his own company and come up with presentative control measures.
- Ventilation professionals should be applied only in their technical field of expertise and management should refrain from giving them “other vicarious” responsibilities.

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