Mitigating Ventilation Air Methane Emissions Safely and Cost-Effectively

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ABSTRACT

Methane has been controlled in collieries in the past only for safety and statutory compliance reasons; however, concerns over greenhouse gas emissions mean that this is now changing. About 65% of greenhouse emissions associated with underground coal mining come from Ventilation Air Methane (VAM). The machinery to mitigate these fugitive emissions has cost and safety concerns. An alternative solution would be a method to prevent methane from entering the mine airstream and becoming VAM in the first place. Recently, in a colliery in the Hunter Valley, this mitigation method underwent a 12-month trial. A reduction in fugitive emissions of 80,307 t/CO_2-e was quantified, (1) at an average cost of A$1.28c t/$CO_2$-e. The mitigation method outlined herein represent a first known attempt in an operating mine, to lower a collieries' environmental footprint by preventing CH_4 from entering the mine airstream and becoming VAM gas by the deliberate use of several targeted mitigation measures, that were individually quantified and costed. Mine safety is also improved with each mitigation measure used.

Key words:

Ventilation Air Methane; Greenhouse Gases; Fugitive Emissions; Global Climate Change.

1. Introduction

Coal mining is Australia's second largest export industry, is a large employer and provides the fuel for most of the country's electrical power generation; however, it is also a substantial emitter of greenhouse gases. Also 65% of the greenhouse gas emissions from collieries (2) is in the form of uncontrolled Ventilation Air Methane (VAM) fugitive emissions. Methane (CH_4) is a strong greenhouse gas; its greenhouse effect (technically its Global Warming Potential, or GWP) is 21 times that of Carbon Dioxide (CO_2) (Table.1).

Table 1. GWP of CO_2 and CH_4; 100-year time frame (3)

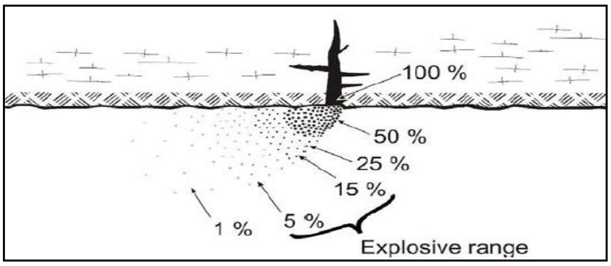
Kyoto Protocol Gases	**Global Warming Potentials**
Carbon Dioxide (CO_2)	1
CH_4 (CH_4)	21

The low concentration of the VAM gas (typically 0.1% – 0.8%) in the high airflow mine air means that it is very difficult to remove (4).

1.1 CH_4 in collieries

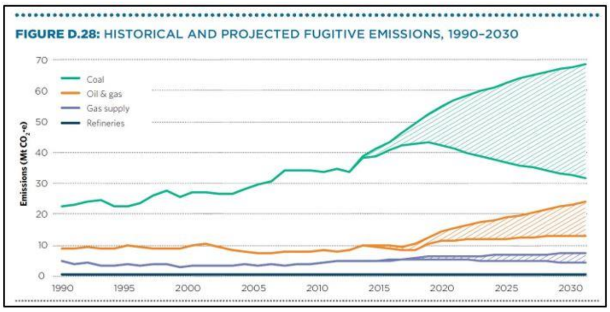
Coal seam gas is usually CH_4 (5). CH_4 is a problem in collieries mainly because it forms an explosive mixture between 5% and 15% when in air (6). CH_4 is stored in coal by a process called adsorption, and the amount of CH_4 contained in a tonne of coal can range from 2m³ to 30m³, the CH_4 is adsorbed into the micro-porous matrix of the coal by intra-particle diffusion (7). When the pressure which is keeping the CH_4 in place reduces, it diffuses into the cleats of the coal. Work by Saghafi et al., (8) has shown that the CH_4 released from a mine is generally four to seven times that which is contained in the coal seam being mined. The act of long-wall mining a seam relaxes strata up to 170m above and up to 60m below the seam, means that most VAM can originate outside the seam being mined, from both above and below (9). CH_4 emissions from the coal seam into accessible roadways are generally made safe by rapid dilution using large volumes of fast-flowing air (Fig. 1). When diluted into the mine air, CH_4 essentially becomes the fugitive greenhouse gas VAM.

Fig. 1. CH_4 emitted from a coal seam into flowing mine air becomes VAM (6)



VAM comprises a significant 1.5% of global anthropogenic greenhouse gas emissions; this is 630Mt CO_2-e. At present, the technology to mitigate VAM cost-effectively, efficiently and safely using a large dedicated plant is a work in progress (10). However, it is prudent to have an alternative method of VAM mitigation which can be immediately rolled out at low cost if needed. Of course, CH_4 has been controlled in collieries for a long time, but historically this has always been for safety and statutory compliance reasons – never for environmental reasons. However, there has been some recent work to indirectly reduce VAM gas by using enhanced gas drainage methods (11). The aim of this work is to show how to prevent some CH_4, in a safe and cost-effective way, from entering the mine airstream and becoming fugitive emissions in the form of VAM. The climate change authority (12) outline the possible range of VAM fugitive emissions from collieries in Australia to 2030 (Fig. 2).

Fig. 2. Fugitive emissions from Australian coal mines



VAM fugitive emissions from collieries currently (2016) represent 38Mt CO_2-e which is a significant 6.3% of Australia's total greenhouse gas emissions. An increase in VAM gas emissions is anticipated in all scenarios, because of projected increases in coal exports and the mining of lower, seams. Around 2022 the use of VAM plants is expected to reduce this source of emissions. However, there are large uncertainties in the projected Figures.

1.2 VAM mitigation difficulties

VAM emissions for the two largest emitters, China and the U.S. have been estimated by the USEPA (13) at 6.7 Billion m³ and 2.6 Billion m³, respectively. Australia's emissions in 2002 were estimated at 0.7 Billion m³ (14). Commercial VAM treatment plants are becoming available, such as the VAM Thermal Oxidiser (VAMTOX). The first successful demonstration of a small scale VAM plant was in 1994 at Thoresby colliery in the UK, then later at Appin colliery and at West Cliff colliery in NSW, Australia. The West Cliff VAM plant successfully generated 6MW of electrical power during operation. More recently a VAM pilot project, which handles 10m³/sec has been undergoing operational testing at Mandalong colliery in NSW (14). Mandalong are planning to eventually fully treat all their VAM gas by using a thermal VAM plant; this would be a world first if it is achieved. To operate efficiently, VAM plants typically require at least a 0.8% CH_4 concentration in the mine airflow; this often means that a supplemental source of CH_4 needs to be added to the mine air flow to reach this level. One alternative to needing this supplement would be to separate the long-wall return from the development return flows, (1) and only treat the long-wall flow which is generally over 0.8%. However, this has not been proposed or put into practice at any mine. Other stumbling blocks to this technology are safety concerns, the very high cost of this form of mitigation and dirty mine air causing problems for the catalysts used in the oxidiser.

1.3 Mitigation and mine safety

The manner and amount by which fugitive greenhouse gases are mitigated depends to a large extent on how much they are taxed or how much is available for mitigation through government schemes like direct action, and what mitigation is specifically covered by those schemes. VAM plants are covered, however there are serious safety concerns in underground coal when attempts are made to attach VAM gas plants to the main fans. Most VAM plants are basically a huge oven, which destroys CH_4 by oxidation at 1,000ºC (15). The idea of connecting a VAM plant to a colliery is meeting stiff resistance from many coal mineworkers and managers. Even flaring drained CH_4 has met resistance to date in the USA; a practice that is arguably safer than a VAM plant is perceived to be (16). It is a fact that coal miners and coal mine managers are rightly cautious when it comes to safety, especially in regards to ignition sources. In Australia, smoking is not permitted anywhere on site, many items are classified as contraband and are not permitted into the mine, items such as cameras and phones, even an ordinary battery operated watch is banned.

VAM can be mitigated in another way, without using a VAM plant. The exhaust mine air can be used as feed air to a genset. This not only destroys the CH_4, but uses all of the feed VAM to generate power in the genset. In a world's first commercial use of VAM, the Appin colliery in NSW, Australia used 20% of its exhaust mine air to feed 54 x 1MW gas gensets which it runs on site from its gas drainage system. This was estimated to add 4 to 8MW to the output of the gensets, resulting in an average power generation of 55.6MW, which was on-sold to a utility (17). Because the feed air is not ignited until it is inside a genset cylinder, this method is seen as safer than some of the alternative methods of VAM gas destruction; however, some serious safety issues still remain. There must be no point at which exhaust mine air (which could contain a plug of flammable CH_4 gas) might contact surfaces of high temperatures and ignite. Strict controls have to be enforced between the point where mine air exits the fans and the point where the high temperatures in the gensets exist. Using the dirty mine air as feedstock air for the gensets was discontinued at Appin due to the frequent cleaning required which caused cost inefficiencies for the operation (17).

The concerns raised by connecting a VAM plant directly to the mine fans centre around the properties of the gas CH_4, which is explosive in air when between 5% and 15% concentrations. If the mine were to expel a plug of CH_4 through the fans at 5%+ concentration, due to the failure of a seal or an outburst event for example, this could prove to be catastrophic. If ignited by the VAM plant, the flame would propagate all the way back through the mine to the source of the leak and could trigger a mine dust explosion. Hence the reticence of some mine managers to even consider the connection of a VAM plant. Occasionally, as here, the two aims of a requirement to not reduce mine safety levels and the desire to mitigate emissions conflict. These conflicts should be identified and dealt with using good, comprehensive risk assessment process. As always, the safety of the mineworkers must take precedence over any thought of mitigation of emissions.

This is where the six measures that were used in the 12-month Hunter Valley trial of the mitigation method, have the advantage over VAM destruction by heat; they are not only completely safe, but by their nature the application of any of them further improves mine safety. Concerns about these mitigation measures therefore will not come from the point of view of safety, but only from a cost perspective. Australian government policy is currently to pay for reductions in emissions which are performed by NGER's regulated collieries through an auction system. It is expected that the cost of implementation in Australia will be wholly met by the clean energy regulator through a major project application to the direct action auction system.

2. Methodology; The mitigation method in the trial involved six measures;

a) Identify and stop seal leaks from seals (1)

Example: A small leak was discovered in an old seal, the leak was measured by surveying the roadway on either side of the seal for airflow and percentage of CH_4, and calculated to emit an average of 700ml per second of CH_4. To simplify matters, no allowance is made in any calculation for pressure or CH_4 density changes due to movement of CH_4 in a vertical direction, which in any case are small because of the shallow depth of the workings in question. Emissions in CO_2-e are given by:

Calculation a);

Ideal CH_4 Law; Density = PM/RT [1]
P = mine pressure = 0.978 atm
M = molar mass = 16.042 g/mol
R = CH_4 constant = 0.82057 L atm mol^-1K^-1
T = mine temp in Kelvin = 298.15 K
Global Warming Potential (GWP) from the IPCC's AR4 = 21
Mine density CH_4 = 0.978 atm x 16.042 g/mol / (0.082057 Latmmol^-1K^-1 x 298.15 K) = 0.641 gm/litre
Litre/700ml = 1.4
Leak is therefore 0.641 grams every 1.4 seconds
There are 31,550,000 seconds in a year/1.4 = 22,500,000
22,500,000 x 0.641 grams = 14.4 tonnes CH_4
CH_4 make x GWP = CO_2-e emissions
14.4 x 21 = 302.4 tonnes CO_2-e emissions/year

A small leak of this size, is difficult to detect without regular and accurate measurements or leak tests taken very near the seal; these are not routinely done. The daily diurnal pressure changes also mean that the leak may often stop or even reverse, making it all but undetectable at those times. Leaks like this are very common in old seals around sealed up panels; and old sealed panels often have 50 or more seals. Finding and plugging this small leak is the equivalent in greenhouse emissions saved, to taking 60 cars off the road. The equivalent of 24 leaks of this size were detected in surveys, (totalling a VAM reduction of 16.8 litre/sec CH_4) and all were quickly and satisfactorily plugged and sealed using portable silent seal products.

Table 2. Costs associated with measure a (1)

Cost calculation	A$
Deputy and Ventilation Officer's time 20 hours	2,000
Mineworker's time applying product 15 hours	1,000
Silent Seal x 2	1,200
Total cost	4,200

b) Seal off unused roadways in the mine (1)

The single-entry back road of the upper PG (Pikes Gully) seam was sealed up on the 29th September, 2012; because this 5km of roadway was already planned to be sealed, the associated emissions savings were not counted as part of this study. However, other roadways were not planned to be sealed off in the normal course of events; main-gate 9 and the back road of LW8 (Fig. 3).

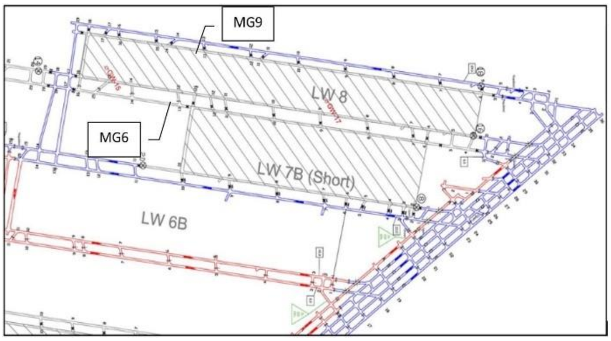


Fig. 3. Hunter Valley colliery detail of LW8 & LW7B

These roadways were known at the mine to be a particular source of CH_4 leaks. This was due to;
* Old-style shotcrete seals, which were unsatisfactory
* Wooden cribs which shrink as they dry out, causing them to fail to support the roof properly
* Geotechnical issues caused by the narrow width of the MG9 pillars
* Spalling ribs in the cut-throughs which caused leaks

Although there had been no plan to seal off these roadways, the cost of the above maintenance issues coupled with the related emissions costs incurred to the mine in terms of the then-existing carbon price, made the decision to seal it off possible. Prior to the seal-up of MG9 and the back road of LW8, the airflow required to ventilate them was 29.5m³/sec; this was more than 10% of the entire mine airflow. Even then, the in-bye end of the roadway could exceed 1% CH_4 during a rapid diurnal reduction in atmospheric pressure, (this represents a CH_4 make of 300 litres/sec) often occurring at 4pm and 4am; thus preventing machinery access at those times (19).

Several spot CH_4 measurements and surveys of the roadway were made by the deputies and the ventilation officer respectively, and the average intake and exit CH_4 levels just prior to seal-up were determined to be;

Intake 5-6 c/t A heading, MG9 = 0.05% CH_4
Backroad LW8 = 0.50% CH_4
MG9 CH_4 CH_4 = 0.45% of 29.5m³/sec
Average make CH_4 = 132.75 litres/sec

Using above calculation, a);

0.641 gm/litre x 132.75 = 85.1 gm/sec
31,550,000sec x 85.1 = 2,684 t CH_4/year
2,684 x 21 GWP = 5.6378×10^4 t CO_2-e/year

The annualised cost of these emissions to the mine at the time was 56,378 x A\$23 = A\$1,296,000. A decision was made to seal these two roadways and to inertise them by using CH_4 from the LW8 sealed goaf.

Calculated time to inertise the 1,800m roadway;

Volume of roadway; 1,800m x 14.5 = 26,100m³
CH_4 required to inertise road; 26,100 x 0.15 = 3,915m³
Estimated CH_4 make of MG9 seals; from 12 noon to 6pm = >300 litres/sec
Estimated time to reach >15% CH_4; 3,915/0.3 = 13,050 sec (3.6 hours)

* Sealing was undertaken on a falling barometer between 11 am and 5pm
* Out-bye end of MG9 will be on –ve 400Pa return pressure causing the seals to breathe out
* LW8 contains 95% CH_4; this will be used to inertise MG9 and the LW8 back road
* The 100mm inertisation pipeline which passes through the seals will be opened on personnel exit
* Up to 20,000 m³ of CH_4 will be stored in this roadway
* Approx. volume of CH_4 in LW8 goaf is 150,000 m³
* This stored CH_4 can be tapped from the surface and used for power generation
* The emissions saved are annualised for one year, even though these roadways in the normal course of events would have been in use for much longer.

Table 3. Costs associated with measure b (1)

Cost calculation	A$
Deputy and Ventilation Officer’s time 100 hours	10,000
Risk assessment	12,000
Seal-up doors x 2	11,000
Pipework and clearing roadway	20,000
Ventilation change and seal-up works	25,000
Monitoring costs	2,000
Total cost	80,000

c) Install 35kPa stoppings in front of 140kPa seals (1)

To use the single-heading of MG6 as a long-wall main-gate intake and as a belt road, called for innovative thinking at the mine (Fig. 3). This situation was brought about because of out-of-sequence panel mining, caused by delays in surface environmental works associated with the re-location of a creek. The installation of the extra 35kPa stoppings was undertaken along the single heading in MG6 to increase seal resistance enough to keep CH_4 ingress from sealed panel LW7B into MG6 to within the statutory limit. Because this roadway was later to be the main-gate (intake) for the extraction of LW6B, these prevention works were important because the statutory limit is just 0.25% CH_4 on long-wall intakes in NSW. Measurements of pressure and CH_4 were done along MG6 and its seals associated with LW7B in order to quantify CH_4 make and any likely problems which may occur due to this leakage into the main-gate during the extraction of LW6B.

The installation of 35kPa mine plaster barrier stoppings in front of the existing 140kPa seals was decided on in at a risk assessment for the mining of LW6B, in spite of the excellent condition of the seals. The decision was based on two factors, prudence; and the modelled and calculated extra leakage expected from these seals as they were put under more differential pressure due to the extra airflows needed for servicing a production main-gate.

A further control would be a 0.25% CH_4 detector in the main-gate, which would trip the power to the long-wall if exceeded. Other contingencies were planned for in the risk assessment, such as an application for an exemption to the 0.25% intake rule; it was hoped permission to allow 0.5% for this panel could be gained from the inspector. Other contingencies which were planned for were a provision to draw off the CH_4 in between the barrier seals; pipework for this was to be pre-installed and excess CH_4 was to be directed into the returns. However, it was hoped that in practice, neither of these contingencies would be required.

Pressure drop down MG6 now is given by:

Calculation b);

Find Atkinson's resistance of the current situation first, using;
$R = KL\ Per/A^3$ [2]
K = Friction Factor = 0.009 kg/m^3 (after McPherson)
L = Roadway Length = 1,000m
Per = Road Perimeter = 16.4m (5.4m wide, 2.8m high)
A = Roadway Area = 15.1m^2
$R = KL\ Per/A^3$
$= 0.009 \times 1{,}000 \times 16.4/15.1^3$
$= 0.0428\ Ns^2/m^8$
Find pressure drop, P;$P = RQ^2$ [3]
$= 0.0428 \times 23.0^2$
$= 22.64$ Pa
Projected pressure drop down MG6 is given by;

Calculation c);

K = Friction Factor = 0.011 kg/m^3 (4)
L = Roadway Length = 1,000m (at start)
Per = Road Perimeter = 16.4m (5.4m wide, 2.8m high)
A = Roadway Area = 15.1m^2
$R = KL\ Per/A^3$
$= 0.011 \times 1{,}000 \times 16.4/15.1^3$
$= 0.0524\ Ns^2/m^8$
Find pressure drop, P;
$P = RQ^2$
$= 0.0524 \times 60.0^2$
$= 188.64$ Pa

Airflows along MG6 were expected to rise from the current 23.0m^3/sec to 60.0m^3/sec during production; a conveyor belt is also to be installed in MG6; because of this, vehicle access is to be largely via B heading in the tailgate. The pressure drop down the length of the roadway was calculated to increase from 22.64 Pa to188.64 Pa. This would increase the pressure differential down the length of sealed panel 7B by the difference between 2/3rds of these numbers (given that panel 7B is two-thirds the length of MG6). This increase in differential pressure along the length of the sealed goaf 7B, can then be quantified as;

2/3rds of 22.6 Pa = 14.9 Pa and
2/3rds of 188.6 Pa = 124.5 Pa respectively.

The pressure differential along the sealed panel length therefore is projected to increase substantially from a negligible 14.9 Pa to 124.5 Pa. This would be expected to cause the in-bye seals to leak more by adding to the already substantial (approx. ±150 Pa) diurnal changes during their median daily lows at around 4pm and 4am. The average diurnal change of ±150 Pa, added to the pressure fall during production along the length of the sealed panel would be 124.4 + 150 Pa = 274.4 Pa.

However, since any passing storm will cause a barometer fall in excess of the average diurnal change, prudence in planning demands that a factor of safety here before any pressure level in our calculations in the determination of the seal resistance which we require to prevent excessive CH_4 leakage into MG6. Because of the very real concern over the potentially high costs associated with any productions shutdown due to CH_4 ingress, a factor of safety of 1.5 was decided on, hence a maximum differential of; 274.4 x 1.5 = 412 Pa was to be used in the CH_4 flow calculations.

Calculation d);

Flow	= 60m^3/sec
Max CH_4 concentration allowed	= 0.25%
CH_4 concentration panel intake	= 0.05%
Leakage allowed is therefore	= 0.20%
	= 60 x 0.002 = 0.12
	= 0.12 m^3/sec
Allow for the CH_4 conc being 90%	= 0.14 m^3/sec

At LW6 start-up, nine seals in MG6 would need to be included in calculations (In B hdg; 3c/t, 4c/t, 5c/t, 6c/t, 7c/t, 8c/t, 9c/t, 10c/t and in A hdg 10-11c/t).

Calculation e);

Since; $Rt = P/Q^2$ [4]
Then; $= 412/0.14^2$
Total resistance; $= 21{,}000\ Ns^2/m^8$
Since;
$1/\sqrt{Rt} = 1/\sqrt{(R3c/t)} + 1/\sqrt{(R4c/t)} ... 1/\sqrt{(R10\text{-}11c/t)}$ (5)
And if; R3c/t = R4c/t = R5c/t = R6c/t = R7c/t = R8c/t = R9c/t = R10c/t = R10 -11c/ t
Then; $1/\sqrt{Rt} = 1/\sqrt{21{,}000} = 0.0069$
And; $0.00077 = 1/\sqrt{(R\text{ any seal})}$
Then; R any seal = $1.68 \times 10^6\ Ns^2/m^8$

The seals in each cut-through therefore need to achieve a resistance of 1.7 million Ns^2/m^8 to satisfy CH_4 leakage restrictions into MG6. This proved to be achievable; each 140kPa seal was sprayed over again, and up to 4m of rib and roof was also sprayed to a depth of 50mm. Then a second barrier seal of 35kPa rating was installed 4m in front of that seal, and again, the roof and ribs sprayed out to 4m and to a 50mm depth. Tube bundle and local monitoring pipes were installed, along with a 200mm CH_4 drainage line through each 35kPa stopping, and running to the main returns. The CH_4 drainage line was to be used in the unlikely event that leakage through the seals into MG6 caused production stoppages.

The installation of the 35kPa stoppings and the over-spraying resulted in some CH_4 leak reductions during the time-frame of this study, however most of the savings would have occurred after this study ended in June, 2013; the reason is because the projected increase in airflow and hence the higher pressure differential did not happen until after that. Therefore, an estimated pro-rata emissions savings and costs have been applied in this case.

Table 4. Costs associated with measure c (1)

Cost calculation	A$
Deputy and Ventilation Officer's time 10 hours	1,000
35kPa stopping installation (part cost)	4,000
Total	5,000

d) Change seal design from shotcrete to mine plaster (1)

Old-style 140kPa shotcrete seals were being installed at the mine. These were found to be unsatisfactory form several points of view;

* Frequently became leaky when roof, ribs or floor moved because of their rigidity and so required constant maintenance
* Many leaks appearing on the roof-line or under through-seal pipework due to slumping of the concrete on installation adding to VAM gas and emissions
* Costly and time-consuming to install
* No new-seal specific documented inspections carried out after installation
* Single-tube roof monitoring arrangement unsatisfactory
* 140kPa seals were being made using different materials to the 35kPa stoppings, requiring separate stocks of materials
* Issues with safety because of the design re; materials handling
* Long-time of installation causing unnecessary risks to workers i.e.; exposure to goaf and its Gases
* Use of outdated wooden cribs, issues are; flammable materials, slow installation, materials handling issues, lack of ability to create rapid and positive roof support when needed, wood shrinkage issues causing failure to support roof over time, lack of continuous support causing rib spalling and leaks

A new seal design was implemented. The new seals are 140kPa and made from High Strength Water Resistant mine plaster (HSWR). The old wooden cribs were replaced by quick to install adjustable 40 tonne roc-props. The single roof-top copper sample pipe was replaced with 3 x pvc sample tubes, set at different heights behind the seal; the "traffic light" standard of red, yellow and green sample tubes. A new seal inspection regimen was implemented to assess and record the installation standard. The HSWR mine plaster seals were superior in almost every respect to the old shotcrete/single sample point/wooden crib arrangement which was in place. However, there is a need to concentrate on their benefits with regard to prevention of CH_4 leakage, both immediately on installation and longer term.

The old seals would commonly have leaks straight after installation, typically of 5-10 millilitres/sec. With time, the cribs would shrink and fail to support the roof both behind the seal and in front of it. The concrete would flake and decay, the bottom became affected by standing water; the roofline would separate from the seal top, movement in the ribs would cause spalling and more leaks. Constant inspections and repair works were necessary. Due to constraints on the deputy's time, often only the leakiest seals were noted and attended to. Detection of lesser problems and action on them was often left to the ventilation officer, working with one of the out-bye undermanager's.

Quantifying the effect on VAM of the switch to HSWR seals is difficult, because due to the expense, they were installed only as required during the normal running of the mine, and were not primarily installed for the specific purpose of controlling emissions. Their superior resilience, rapidity of installation, flexibility, lack of slumping, resistance to mine water and use of positive support like roc-props instead of cribs, all conspired to reduce both their initial production of VAM, and their production of this fugitive emission over time. These new seals are being installed at the rate of approximately 25 per panel, and one panel is mined on average every year. If it is assumed the lowest likely average leak difference of 50 millilitres/sec, then the emissions saved after 1 year of steady replacement of the old leaky seals would be;

Calculation f);

$50 \times 25 \times 0.5 \times 3.15 \times 10^7 = 1.9 \times 10^7$ litres saved
If we take the density of CH_4 is taken from calculation a above, = 0.641 grams/litre, then;
Savings: $0.641 \times 1.97 \times 10^7 = 1.264 \times 10^7$ grams
$= 12.64$ t $CH_4 \times 21$ GWP
$= 265.44$ t CO_2-e/year

Because the switch to a new seal design was not primarily done to reduce emissions, only a small part of any possible change-over cost is allowed here. In fact, a subsequent cost-benefit analysis has shown that the change-over to new seals had no net cost, and was revenue positive for the mine.

Table 5. Costs associated with measure d (1)

Cost calculation	A$
Deputy and Ventilation Officer's time (part cost)	500
Engineer's design drawings (part cost)	500
Total	1,000

e) Reduce leaks from goafs by pressure balancing panels (1)

Leaks from old sealed panels in a long-wall coal mine still happen, even after the above strict regimen has been followed; i.e. fix leaky seals, seal off unused roadways, install barrier seals and switch to new and better seals. These leaks can be because of a combination of; the way the ventilation is arranged, and the diurnal change in atmospheric pressure. Something can be done about the ventilation arrangements. A sealed panel, if not pressure balanced, will leak CH_4 out of one side, and leak mine air into the other side. This is most undesirable in three ways;

I. More VAM gas is created than needs to be, causing more fugitive emissions and also potential access issues due to gas in the returns during storms or common diurnal pressure falls.

II. Mine air leaks into sealed areas are to be avoided if possible, due to spontaneous combustion and explosive atmosphere risks.

III. Efforts to prevent the ingress of mine air into the sealed area, and efforts to prevent gas from leaking out of the sealed area cost time and money.

Even so, sealed panels which are not pressure balanced are common in underground coal mines in Australia. In the example here (Fig. 3) the sealed panels LW8 and LW7B were calculated to have combined due to the strong likelihood of some of the seals between them collapsing during the mining of LW8, in particular 8c/t and 9 c/t, MG8. They can therefore be treated as a single sealed panel for pressure balancing purposes. Given that LW6B was still to be mined, and that it would be undesirable to have the seals in MG6 leaking excessively when put under a potential pressure drop of 412 Pa (as calculated in c above) not only from the point of view of the continuity of production, but from an emissions standpoint, it was decided to induce a negative pressure gradient across from MG6 to MG9.

To enable this, the correct course of action was decided from modelling to put MG9 on full return pressure to pull back the goaf gases in the combined panel away from the MG6 seals as much as possible to prevent CH_4 ingress into the LW6B main-gate. To this end, a ventilation change was made, which removed all regulation in the PG mains returns in A or B heading and introduced regulation in the form of mine doors in A heading, MG9 4-5 c/t. Regulation started across the doors at 475 Pa and in succeeding months varied up to 910 Pa as production moved from the ULD to the mining of LW6B in the PG seam; the MG9 seals were basically kept on the existing full return pressure for months before, and throughout the mining of LW6B.

After mining of LW6B commenced, this had the effect of helping to prevent excessive CH_4 movement across from the combined panel into the new LW6B goaf, as it was expected from a geotechnical study that one or two seals in MG6 would probably collapse after the long-wall passed them. The prevention of sudden movements of high-percentage stored CH_4 gas was always a part of the ventilation planning process. From tube bundle monitoring, the combined panel was known to contain approx. 90% CH_4, therefore it was not possible to increase the CH_4 content in this goaf very much. However, it was possible to do this in other panels, such as the sealed goaf of LW1 in the PG seam; this additional stored CH_4 (by increasing CH_4 concentration in a previously sealed goaf) will be quantified next. However, the initial effect of putting MG9 on full return pressure was to create a pressure balance across the sealed panels and so prevent leakage through seals on all sides of the sealed panel. This reduced the creation of VAM gas. Gas surveys were taken to quantify this saving in emissions due to the pressure balancing across the sealed panel LW8 and LW7B details of this measure is described as follows;

Table 6. Measurements of CH_4 concentrations in MG6 from a gas survey (1)

Place measured	CH_4 % general body; 10m out-bye of the most in-bye seal noted
MG6 A hdg 3-4 c/t	0.05
MG6 A hdg 4-5 c/t	0.08
MG6 A hdg 5-6 c/t	0.12
MG6 A hdg 6-7 c/t	0.15
MG6 A hdg 7-8 c/t	0.20
MG6 A hdg 8-9 c/t	0.22
MG6 A hdg 9-10 c/t	0.25
MG6 A hdg 10-11 c/t	0.30
Total gas make is:	0.25% of general body

As noted in measure c above, (1) it was necessary to achieve a resistance of 1.7 million Ns^2/m^8 in the seals along MG6 in order to satisfy our CH_4 leakage restrictions into MG6. It is now positioned to be able to calculate the resistance of these seals prior to the ventilation change to triple the airflow down MG6. This assists us with planning the fine detail of the ventilation arrangements for the mining of LW6B.

Calculation g);

Measured airflow in MG6, A hdg intake, 4-5 c/t: 28.4 m^3/sec
Average make CH_4 28.4 x 0.25% = 71 litre/sec
Assume all 8 seals involved have the same resistance.
Average seal leakage is approximately; 71/8 = 8.87 litre/sec

Measured seal pressures during the gas survey are; across 4 c/t seal +200 Pa and across the 10c/t seal +240 Pa therefore MG6 seals are all breathing out (gas survey was deliberately carried out during a diurnal fall in the barometer). The average seal pressure is taken to be +220 Pa. Find the resistance of the individual seals;

Since;
$$R = P/Q^2$$
$$= 220 / 0.00887^2$$
$$= 2.8 \times 10^6 \ Ns^2/m^8$$

This gas survey confirms that the seal over-spraying, installation of a second 35kPa barrier seal and roof and rib spraying has worked and the seal target resistance of $1.68 \times 10^6 \ Ns^2/m^8$ is easily reached.

Average gas make MG6 before pressure balancing
= 82.9 litres/sec
Average gas make MG6 after pressure balancing
= 51.1 litres/sec
Measured mitigation from pressure balancing LW8 and LW7B = 31.8 litres/sec

Calculation h);

0.641 gm/litres x 31.8 = 20.38 gm/sec
31,550,000sec x 20.38 = 643 t CH_4/year
643 x 21 GWP = 1.35 x 10^4 t CO_2-e/year

Table 7. Costs associated with measure e (1)

Cost calculation	A$
Deputy and Ventilation Officer's time 50 hours	5,000
Ventilation change	2,000
Total	7,000

f) Use pressure differential to move CH_4 to goaf voids (1)

As noted above in measure b, 20,000 m³ of CH_4 was stored in MG9 and the LW8 back roadway. Another panel where pressure differentials were used to move CH_4 is when the LW101 panel was being mined in the Upper Liddell seam (ULD) which is a lower seam to the PG, being 40m lower. In this case, the CH_4 was moved by putting the sealed panel LW1 of the PG seam on full return pressure, through its accessible seals. This amounted to a pressure differential of 250 Pa when compared to the centre of the long-wall (1) of the LW101 panel at start-up and increasing to 515 Pa at the 2/3rd mined stage.

Another reason this was done was to prevent CO_2 from coming down onto the long-wall (1) from the old LW1 goaf and causing the statutory CO_2 level of 1.25% in working areas from being exceeded. The CO_2 levels were known to be 9% - 22% in the old LW1 goaf from tube bundle monitoring; the CH_4 levels were also known to be 5% - 10% with negligible levels of O_2. The plan was to keep this overlying goaf inert right through the extraction of the LW101 panel by causing much of the CH_4 released during the mining to flow upwards using a sufficient pressure differential. The O_2 was kept low by a strong regimen of surface remediation works, which involved using a dozer over the subsidence-induced surface cracks to rip and then compact the surface wherever cracks were seen.

Excessive O_2 was prevented from flowing upwards from the long-wall by a 'loop' of pressure from the main-gate to the tail-gate; causing the majority of ventilation air to descend down into the tail-gate returns. This was done to reduce inter-seam or goaf spontaneous combustion risk and to reduce the volume of potentially explosive gas mixtures. Other active CH_4 controls were a tight brattice barrier across the main-gate, level with the chocks, a tail-gate brattice barrier and a close back-road bleed to pull CH_4 away from the tail-gate machinery. The mining of LW101 was preceded by extensive modelling, monitoring and calculations to ensure that CH_4 movements were not going to be adverse when the panel was mined. One aim was to ensure that as much CH_4 as possible was left in the combined goafs after LW101 was completed and sealed. This was achieved through buoyancy pressure and differential mine pressure brought about through the ventilation arrangements. CH_4 production from the long-wall which was excessive was drawn off by a surface goaf CH_4 drainage plant, which operated through pre-drilled vertical holes at a spacing of 500m, centred on the panel and ending 17m above the PG seam. The concentration of CH_4 in the LW1 panel was monitored by three pre-existing tube bundle points, and increased from 6% to 30% during the period 20th Sept – 1st Nov 2012 (Fig. 4).

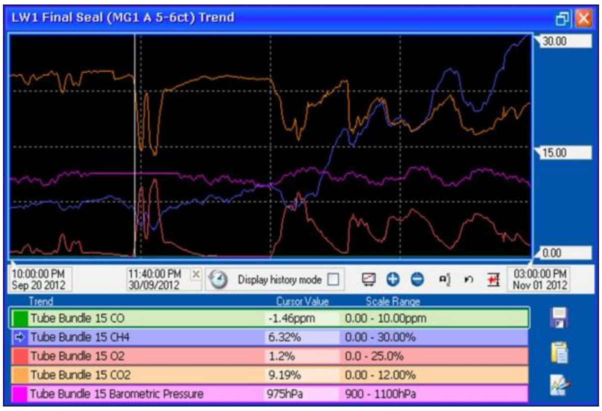


Fig. 4. CH_4 (blue line) increases in sealed LW1 panel (1)

Concentrations of CO_2 and O_2 remained steady. By the completion of the entire LW101 panel, the CH_4 concentration had lifted to 65% in the LW1 PG (the upper seam) goaf. (1)

To ascertain the amount of extra CH_4 being stored, we need to ignore the lower LW101 panel goaf, and count the extra CH_4 stored only in the LW1 PG goaf. The costs involved in this storage were minimal, since the main expense was a limited amount of the ventilation officer's time for ventilation modelling. Surface remediation costs were not included, since they would have happened anyway due to spontaneous combustion and explosive atmosphere concerns.

Calculation i);

Volume of LW1 goaf = ½ volume of removed coal
LW1 = ½ x 1,980 x 2.5 x 210 = 208,162 m³
Total = 208,162 m³

CH_4 increases from an average of 7.5% to 65%.
Pre-existing CH_4 stored = 7.5% of 208,162 m³
= 13,530 m³
New stored volume = 65% of 208,162 m³
= 135,305 m³
Extra amount stored = 135,305 – 13,530
= 121,775 m³

Add the 20,000 m³ which was stored in the seal-up of MG9 and the LW8 back road;
Total stored = 141,775 m³

From calculation a, the density of CH_4 under the specified conditions is 0.641 grams/litre or 0.641 kg/ m³ Therefore, the total extra CH_4 stored in these two voids is equal to a CO_2-e of;
141,775 x 0.641 kg = 90.88 tonnes CH_4
CO_2-e is; 90.88 x 21 = 1,908 tonnes CO_2-e

Table 8. Costs associated with measure f (1)

Cost calculation	A$
Ventilation Officer's modelling time 35 hours	3,500
Ventilation change	2,500
Total	6,000

Abatement calculated to be achieved in the period August 2012 to June 2013 (in tonnes CO_2-e, over a projected 12-month period) using the six different measures were as shown in Table 9;

Table 9. Abatement achieved - each of the six measures (1)

Measure	Tonnes CO_2-e
a) Stop leaking seals	7,256
b) Seal off roadway	56,378
c) Install 35kPa stoppings	1,000
d) Change seal design	265
e) Pressure balancing panel	13,500
f) Increase CH_4 % old goafs	1,908
Total	80,307

Fig. 5 is a direct measure of the PG VAM changes over the trial period (1); VAM abatement is detailed in Table 10.

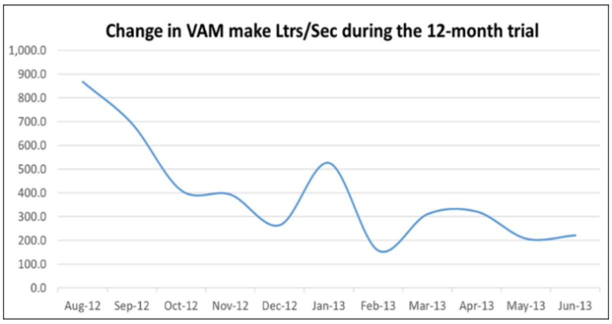


Fig. 5. VAM as measured in the main return of the PG seam during the 12-month trial*

The total mitigation achieved over a projected 12-month period, (80,307t CO_2-e) is equal to taking 17,000 cars off the road (20). In addition, for a total mitigation cost of A$103,200 a total of A$1,847,061 in projected carbon tax costs over the subsequent 12 months alone was averted.

**note; not all the reduction in VAM gas seen here is due to the mitigation trial.*

Table 10. VAM abatement in litres/sec for each measure (1)

Methodology	VAM abatement achieved in litres/sec;
a	16.8
b	132.8
c	2.5
d	0.6
e	31.8
f	4.4
Cumulative	188.9 litres/sec

3. Government incentives to push abatement

3.1 Direct Action

The current Australian government is using a different means of greenhouse gas reduction to the previous government's carbon price. Under 'direct action' the government buys carbon abatement in auctions through the clean energy regulator. The first auction of emission reductions was completed in April 2015, at an average abatement price of $13.95 t/$CO_2$-e (21). Collieries do qualify for abatement under direct action because they are almost all registered as large emitters under the National Greenhouse and Energy Reporting System.

4. Method may be extended

4.1 Other Australian collieries

The mitigation method that consists of six measures (all of which are detailed here) and which underwent a 12-month trial in the Hunter Valley, is applicable to most if not all collieries in Australia; and if it were to be extended to all Australia's 30 collieries, VAM gas emissions reductions amounting to several million tonnes of CO_2-e per year should be possible. The method has four great advantages over other mitigation methods in collieries; these are that this type of mitigation is achievable at a very low cost, complicated process equipment is not needed, roll-out can be very rapid and implementation actually increases mine safety. In fact, all six measures used in the 12-month trial individually enhance mine safety. However, when compared to mitigation or abatement of greenhouse gases in other areas of industry, the greatest advantage of this method is the very low cost of mitigation, which means in effect that far more emissions can be cut for the same dollar investment.

5. Conclusion

Planning for the control of fugitive greenhouse gas emissions such as VAM gas from a coal mine were not even considered until very recently. The new paradigm of a possible or an actual imposed cost (dollar cost or a reputation cost) or a possible financial benefit (for example; direct action mitigation) in relation to fugitive emissions means that greater consideration needs to be given to ventilation planning in certain specific areas. One solution would be a method to prevent more CH_4 from entering the mine airstream and becoming VAM in the first place. The mitigation method outlined herein represent a first known attempt in an operating mine, to lower a collieries' environmental footprint by preventing CH_4 from entering the mine airstream and becoming VAM gas by the deliberate use of several targeted mitigation measures, that were quantified and costed. Recently, in a colliery in the Hunter Valley, this mitigation method underwent a 12-month trial, and involved six different measures. Measurements were taken to assess the emissions mitigation which was achieved and the cost of the works, all the results of the trial are detailed here. A reduction in fugitive emissions of 80,307 t/CO_2-e (1) below that which was projected for the next 12-month period was quantified, at a total cost of A$103,200 which represents an average mitigation cost of A$1.28 t/$CO_2$-e. (1) Note that the measured abatement is specifically related to those emissions which were projected to happen over only the following 12 months, in a business-as-usual case. In reality, the actual abatement total could be expected to be several times greater, simply because the mine life was several times greater than 12 months. Essentially this is due to the VAM emissions reductions achieved with each measure continuing after the initial projected period. The estimated abatement (and abatement costs) therefore represents a very conservative assessment of the likely actual abatement (and abatement costs) here achieved.

Even so, this average mitigation cost of A$1.28 t/$CO_2$-e is both well below either the old carbon pricing mechanism, the prevailing low carbon price in Europe or the current Australian government's emission reduction fund's average price of $13.95 t/$CO_2$-e from its first auctions in April and November of 2015, under its direct action plan through the clean energy regulator. It is also two orders of magnitude lower than the mitigation cost of large scale wind or rooftop solar photo-voltaic (22). The mitigation method in this study is applicable to most collieries in Australia; and if it were to be extended to all Australia's 30 collieries, VAM gas emissions reductions amounting to several million tonnes of CO_2-e per year should be possible. The great advantages of this method is that this would be achievable at a very low cost when compared to mitigation in other areas of the economy, complicated machinery is not required to achieve it, rapid roll-out is possible, and safety is not compromised; in fact, all measures used as part of this method individually enhance mine safety.

6. Acknowledgements

Several Australian collieries assisted greatly by providing data and access to their mines, without which this study would not have been possible.

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