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STRATEGIC REVIEW OF GAS MANAGEMENT OPTIONS FOR REDUCED GHG EMISSIONS

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Glossary

AGO	Australian Greenhouse Office
ANFO	Ammonium nitrate/fuel oil mixture
CFC	Chlorofluorocarbons
CO	Carbon monoxide
CO2	Carbon dioxide
CO_2-e	CO_2 mass equivalent of greenhouse gases; e.g. 1 kg of CH_4 = 23 kg CO_2-e
CH_4	Methane
Hi-vol	High-volume sampler for ambient particulate sampling
IPCC	Intergovernmental Panel on Climate Change
NEPM	National Environment Protection Measure
NGGI	National Greenhouse Gas Inventory
NO	Nitric oxide
NO_2	Nitrogen dioxide
NOx	Oxides of nitrogen
N_2O	Nitrous oxide
NOHSC	National Occupational Health and Safety Commission
NPI	National Pollutant Inventory
PAH	Polycyclic aromatic hydrocarbons
PM1	Particles with an aerodynamic diameter of less than 1 μm
PM2.5	Particles with an aerodynamic diameter of less than 2.5 μm
PM10	Particles with an aerodynamic diameter of less than 10 μm
ppm	Parts per million
ROM	Run of mine
SO_2	Sulphur dioxide

STEL	Short term exposure limit
TWA	Time weighted average
UNFCCC	United Nations Framework Convention on Climate Change
VOC	Volatile organic compounds
GRS	Gas reservoir size, m^3/m^2
TDGC	Total desorpable gas content, m^3/t
SGE	Specific gas emission, m^3/t mined
Er	Fraction of GRS released
Tws	Working section $t/m^2 = \rho.t$
ρ	Strata density, t/m^3
t	Strata thickness, m

EXECUTIVE SUMMARY

The total greenhouse gas (GHG) emissions from Australian coal mining industry were around 26 to 30 Mt CO_2-e in 2007. In order to reduce greenhouse gas emissions from Australian coal mining industry, it has been identified that there is a fundamental need for review and development of novel strategies for gas and ventilation management in coal mines. Gas management has also been ranked as the number one critical parameter to be addressed to achieve high production rates from longwall faces and is arguably the most significant single factor that will constrain mining tonnage from underground coal mines in Australia, particularly with the unavoidable increase in depth of future workings.

To address this critical issue, ACARP has formulated a research programme with the main objective of reducing coal mining fugitive emissions through step-change advancements in gas and ventilation management technologies and strategies in Australia. This project, as a first step towards achieving the objectives of the research programme, has conducted a critical review of gas reservoir characteristics and gas emissions from underground coal mines in Australia; reviewed and analysed the current gas drainage technologies used in Australia; reviewed and evaluated the potential benefits of overseas methods that are not used in Australia; and identified technologies and practices that might have potential to achieve significant improvements in gas management with reduced GHG emissions under Australian conditions.

A critical review of gas reservoir characteristics in Australia has shown that the range of gas contents is from <1.0 m^3/t to between 18 and 30 m^3/t with the limit of higher gas contents at greater depths (currently around 550m) being dependent on CO_2/CH_4 composition ratios. As most of the Australian coal basins contain multiple seams, the gas reservoir sizes are generally high and range from < 5 to 90 m^3/m^2, but in some locations are as high as 200 m^3/m^2 where there is potential ingress from large gas reservoirs. These large gas reservoirs lead to high gas emissions in Australian coal mines, with specific gas emissions (SGE) ranging from <1 to more than 20 m^3/tonne of coal production. In very highly gassy mines, the SGE is as high as 35 m^3/t. However, it is to be noted here that a large proportion of these specific gas emission is being drained in most of the gassy underground coal mines and there are substantial variations in fugitive emissions from different coal mines. It is very critical to identify and characterise all sources of gas in order to develop appropriate technologies and strategies for reducing fugitive emissions from coal mines.

The total greenhouse gas emissions from underground coal mines are around 16 to 17 Mt CO_2-e out of the total coal mining industry emissions of around 26 to 30 Mt CO_2-e. The total ventilation air methane (VAM) emissions are around 80% of the total fugitive emissions from underground mines. In addition, 20 m^3/s of CH_4 is drained from the underground coal mines, which equates to around 40% of the total coal mining methane (CMM) emissions. It is to be noted here that around 75% of the drained gas is utilised for power generation and/or mitigated in flares.

In Australian coal mines, a total gas capture efficiency of 30 to 50% is typical with the highest being about 75%. It is to be noted here that, if pre-drainage is ignored, then the gas capture efficiency reduces to 20 to 30%. Therefore, development of technologies and strategies to increase gas capture efficiency are very important to reduce fugitive emissions.

In a number of mines, longwall airflow is typically around 30% of mine ventilation, but may contain up to 70% of the gas reporting to VAM. Therefore, the application of VAM units to parts of a mine ventilation circuit would be advantageous in many gassy operations where longwall gas emission is largest contributor to fugitive emissions. In addition, alternative ventilation systems such as split or bleeder ventilation systems may be employed in highly gassy mines to mitigate VAM emissions.

A critical review of international gas drainage practices and technologies has identified a number of gas drainage strategies not currently employed in Australia, such as cross measure roof holes adjacent to the face and superjacent drilling galleries. The review has revealed that a number of novel technologies and strategies are being deployed in overseas countries to increase gas capture efficiencies, even under low to moderate gas emission conditions. Essentially most of the gas drainage options used internationally are available to the Australian coal mines, although application is subject to site specific conditions, safety considerations and economic parameters.

Currently, the main objective of gas drainage in Australia is to prevent the risk of outbursts and control gas concentrations in coal mines. There is a need for fundamental shift in the approach for gas management, i.e to move from the current "Gas Control" approach to "Gas Capture Maximisation" to reduce GHG emissions from coal mining. In this context, for example, the opportunities for increasing pre-drainage gas capture ratio includes: pre-drainage of non-working seams, pre-drainage well ahead (5 years) of production operations, use of hydro-fracturing in increase drainage rates and use of inert gas injection to increase pre-drainage rates. The opportunities for improving post-drainage ratio includes: implementation of deep goaf gas drainage strategies, increased drainage in close proximity to the face, use of MRD technologies for post drainage and sealed area goaf drainage. From the analysis guidelines provided in this report on various gas drainage and VAM mitigation options and CO_2-e emissions, decision making needs to be undertaken on a site specific basis.

There is a need for further work on the continuous measurement of air quantity in coal mines in general both for GHG emission and operational reasons. There is a need to improve methodologies and technologies to estimate or directly measure emissions from coal stockpiles on the surface. The current post-mining estimation factors provided may significantly underestimate emissions that are arising in gassier coal mines with short underground residence times. The post-mining emissions also depend on long term residual gas contents and the rate of desorption on surface.

The aim of the coal mining industry should be to develop and operate "near zero emission" mines. It is to be noted here that the specific technologies and strategies to achieve near zero emissions for any individual mine depends on site specific parameters and conditions. In this context, the ideal future GHG friendly underground coal mine is one in which;

- All sources of seam gas emission are properly accounted for, monitored and quantified.

- There is good reconciliation between gas reservoir in place, effect of mining on all sources, including roof and floor seams, and therefore a defined reliable relationship between gas emission and production rates together with prioritisation of pre drainage targets.

- The fugitive/VAM emissions are reduced significantly (to only around 500 l/s flow rates, i.e., to below 10 – 20% of total CMM emissions in highly gassy mines, as indicated in Figure 1)) through development of alternative goaf gas drainage technologies and strategies.

- The volume of gas captured by pre and post drainage systems is maximised with due consideration to the cost of capture compared to discharge.

  That means, the objective of gas drainage should change from the current ”Gas Control” approach to “Gas Capture Maximisation” approach. The efficiency of gas drainage operations in different mines has to be increased significantly from the current levels of 30 – 50% to over 70 - 80% levels.

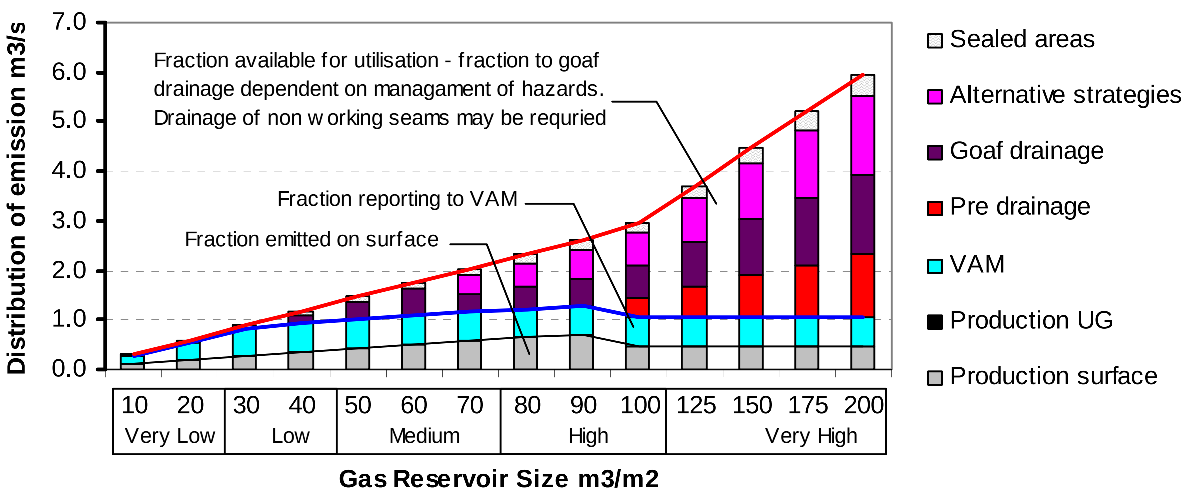


Figure 1 Gas capture maximisation scenario in ideal GHG friendly mine

- All captured gas streams are at least flared with power generation or direct gas sales provided for when economically viable.

- Consequently, the volume of gas reporting to ventilation systems is minimised (Figure 1) allowing less ventilation to be employed (lower fan power consumption), less development required for distribution, improved safety and reduced gas constrained production. Where appropriate VAM oxidation units are applied to some or all VAM streams with increasing use of reject heat (VAM units and IC engines) to reduce mine or local area power consumption.

Currently, fugitive GHG emissions are very high at more than 50% of total coal mine methane and reaching rates up to 3,000 l/s in a number of gassy mines. It is critical to adopt “Gas Capture Maximisation” approach and develop innovative gas drainage technologies and strategies to significantly increase the total gas capture rates and reduce GHG emissions from coal mining.

1. INTRODUCTION

The total greenhouse gas (GHG) emissions from Australian coal mining industry are around 26 to 30 Mt CO_2-e in 2007. In order to reduce greenhouse gas (GHG) emissions from Australian coal mining industry, it was identified that there is a fundamental need for review and development of novel strategies for gas and ventilation management in coal mines. Gas management has also been ranked as the number one critical parameter to be addressed to achieve high production rates from Australian longwall faces and is arguably the most significant single factor that will constrain mining tonnage, particularly with the unavoidable increase in depth of future workings.

It is to be noted here that extensive research work has been carried out in Australia and internationally over the years on coal seam gas characterisation, gas emission estimations, pre-drainage, outburst control, directional drilling and post-drainage technologies. The research led to development of a number of gas drainage technologies and significantly improved the performance of gas management systems in coal mines for operational control and mine safety. However, it was identified that there is a critical need for development of "breakthrough" strategies for gas and ventilation management in gassy mines in order to progress towards high capacity longwalls in the Australian coal industry and significantly reduce GHG emissions from the coal mines.

To achieve the objectives of the research programme, the project conducted a critical review of gas reservoir characteristics and gas emissions from underground coal mines in Australia; reviewed and analysed the current gas drainage technologies used in Australia; reviewed and evaluated the potential benefits of overseas methods that are not used in Australia; and identified technologies and practices that might have potential to achieve significant improvements in gas management with reduced GHG emissions under Australian conditions.

1.1 Objectives and scope of work

The main objective of the research programme is to achieve step-change advancement in gas management technologies and strategies in Australia to reduce GHG emissions and facilitate higher longwall production levels. This project, as a first step towards achieving that objective, has conducted a strategic review of various gas and ventilation management options to reduce gas emissions from underground coalmines. The scope of the project includes:

- analysis of gas reservoir characteristics, gas drainage data and gas emissions from a number of underground coal mines in different Coalfields of Australia;
- review and analyse the limits of current gas drainage technologies used in Australia;
- review and evaluation of overseas gas management strategies for possible applications in Australia;
- brainstorming of opportunities, ideas and concepts that may have potential to achieve significant improvements in gas management technologies and strategies under Australian mine site conditions;
- brainstorming of various alternative ventilation options including segregated ventilation systems to maximise gas capture and reduce GHG emissions; and
- development of guidelines for mine decision making in respect of various strategies for reducing GHG emissions and identify areas for further detailed research.

1.2 Overview of project studies

The project has coordinated several elements and conducted a comprehensive review of various aspects of gas emissions and management in coal mines. The pertinent aspects of Australia's approach to, and legislation concerning accounting of fugitive GHG emissions from coal mines are summarised in Chapter 2.

The gas reservoir characteristics and gas emissions from Australian coal mines are presented in Chapter 3. The range of parameters influencing gas emission from Australian underground coal mines are summarised in this chapter, which provide a means of quantifying the current and potential future range of gas emission rates together with identifying opportunities for improved mitigation of fugitive emissions.

A brief review of global coal mine methane emissions and gas drainage and control technologies practiced in major coal mining countries are presented in Chapter 4, together with a discussion on gas utilisation options and ventilation air methane (VAM) mitigation technologies and issues.

The main objective of the current ventilation and gas management practices in Australian coal mines is to control gas within the safe limits to enable the operation of coal mines, i.e. to carry out minimum gas drainage required to prevent outbursts and operate mines safely. Chapter 5 discusses a number of additional gas drainage strategies and practices that can be adopted to maximise gas capture in underground coal mines in order to reduce fugitive GHG emissions. A detailed discussion on various ventilation circuit configurations to reduce and mitigate VAM and other GHG mitigation strategies are presented in Chapter 6.

With respect to mitigation of GHG emissions, it is recognised that there are three gas streams to consider. Firstly, gas emitted into ventilation then transported to surface at low (typically <2.0% CH_4) concentrations, secondly, gas transported to surface in ROM coal, at a rate dependent on gas content and production, then emitted from surface stockpiles, and thirdly pre and post gas drainage streams reticulated to surface at high (typically >30% CH_4) concentrations where it is available for utilisation or destruction. The key to reducing GHG emission lies in increased capacity of pre and post drainage system thus reducing the emission load through ventilation and production.

Chapter 7 provides an operational and economic rationale for decision making at mine sites on various strategies to be adopted under different gassy conditions to reduce GHG emissions. The main conclusions and recommendations are summarised in Chapter 8.

2. AUSTRALIAN GHG EMISSIONS

The purpose of this section of the report is to summarise pertinent aspects of Australia's approach to, and legislation concerning accounting of fugitive greenhouse gas emissions from underground coal mines.

2.1 GHG Emissions Accounting

The *National Greenhouse and Energy Reporting Act 2007* (the Act) was passed in September 2007 establishing a mandatory corporate reporting system for greenhouse gas emissions, energy consumption and production. As per the Act, from 1 July 2008, corporations were required to register and report for the 2008-2009 financial year if:

- They have operational control of a facility that emits 25 kilotonnes or more of greenhouse gases (CO2-e), or produce or consume 100 terajoules or more of energy; or
- Their corporate group emits 125 kilotonnes or more greenhouse gases (CO_2-e), or produces or consumes 500 terajoules or more of energy.

For the purposes of applying these levels to underground coal mines, 25,000 tones CO_2-e = 56 l/s CH_4. A value that covers most underground coal mines in Australia.

To facilitate GHG accounting, *The National Greenhouse Accounts* (NGA) Factors has been prepared by the Department of Climate Change, Australia. The NGA Factors are designed for use by companies and individuals to estimate greenhouse gas emissions for reporting under various government programs and for their own purposes. As per the NGA factors published in November 2008, the emissions from coal mines are further divided into sub sectors viz., emissions from underground mines, open cut mines, fugitive emissions from flared coal mine waste gases and emissions from decommissioned underground mines. A summary of these NGA emission factors for coal mines are presented below.

2.1.1 Underground Mines Without Gas Utilisation or Destruction

Fugitive emissions from underground mines involve the release of methane and carbon dioxide during the mining process due to the fracturing of coal seams, overburden and under burden strata. Emissions also arise from post mining activities such as transportation and stockpiling of coal due to the release of residual gases not released during the mining process. Emissions will also occur when coal mine waste gas is flared or consumed in power generation engines.

Fugitive emissions from extraction of coal $\mathbf{E_j = Q \times EF_j}$

where:

E_j is the fugitive emissions of methane (j) that result from the extraction of coal CO_2-e tonnes.

Q is the quantity of run-of-mine coal extracted (tonnes).

EF_j is the emission factor for methane (j) (CO_2-e tonnes per tonne of run-of-mine coal extracted), 0.305 tonnes CO_2-e / tonne of run off mine coal for gassy UG mines (>0.1%CH_4 in returns) and 0.008 tonnes CO_2-e / tonne of run off mine coal for non-gassy UG mines.

2.1.2 Direct Measurement of Emissions

The *National Greenhouse and Energy Reporting Act* 2007, and the regulations made there under, specify whether or not any corporation, facility or industry needs to register for reporting the GHG emissions and energy thresholds and maintain the records. The National Greenhouse and Energy reporting guidelines in conjunction with the *National Greenhouse and Energy Reporting (Measurement) Technical* guidelines shall be used for applying the calculation, methods and criteria for greenhouse gas emissions, energy production and consumption from any of the above mentioned categories.

As per the latest NGER technical guidelines 2009, applicable for the reporting year 2009-10, any underground coal mining activity likely to release GHG gases more than the reporting threshold shall account for CH_4 and CO_2 from extraction of coal and CH_4, CO_2 and Nitrous Oxide due to flaring of waste gas. Only the method using direct measurement of CH_4 and CO_2 shall be used for fugitive GHG emissions from underground coal mining. This is called "method 4" under *NGER (Measurement) Guidelines*, 2009 and is as follows;

Method 4 $E_j = CO_2\text{-e}_{jgen}, \text{total} - \gamma_j(Q_{ij,cap} + Q_{ij,flared} + Q_{ijtr})$

where:

E_j is the fugitive emissions of gas type (j) that result from the extraction of coal from the mine during the year, measured in CO_2-e tonnes.

CO_2-e $_{jgen}$, total is the total mass of gas type (j) generated from the mine during the year before capture and flaring is undertaken at the mine, measured in CO_2-e tonnes and estimated using the direct measurement of emissions.

γj is the factor for converting a quantity of gas type (j) from cubic metres at standard conditions of pressure and temperature to CO_2-e tonnes, being:

for methane — $6.784 \times 10^{-4} \times 21$; and

for carbon dioxide — 1.861×10^{-3}.

$Q_{ij,cap}$ is the quantity of gas type (j) in coal mine waste gas type (i) captured for combustion from the mine and used during the year, measured in cubic metres and estimated in accordance with Division 2.3.6.

$Q_{ij,flared}$ is the quantity of gas type (j) in coal mine waste gas type (i) flared from the mine during the year, measured in cubic metres and estimated in accordance with the emission factors furnished for the flaring of gases.

Q_{ijtr} is the quantity of gas type (j) in coal mine waste gas type (i) transferred out of the mining activities during the year measured in cubic metres.

2.1.3 Fugitive Emissions From Post-Mining Activities (Gassy U/G mines only)

Emissions from post-mining activities associated with gassy underground mines can be estimated using the method described below. These are clearly an approximation and will be subject to review.

$E_j = Q \times EF_j$

Where:

Q is the quantity of run-of-mine coal extracted (tonnes).

EF_j is the emission factor for methane (j), measured in CO_2-e tonnes per tonne of run-of-mine coal extracted from the mine and is specified for coal mines in each state, Table 2.1.

Table 2.1 Post Mining Emission Factors

Mine Location	t CO_2-e per tonne	CH_4 m3/t
New South Wales	0.045	3.2
Victoria	0.0007	0.05
Queensland;	0.017	1.2
Western Australia	0.017	1.2
South Australia	0.0007	0.0
Tasmania	0.014	1.0

2.1.4 Open Cut Mines

The method to estimate the emissions from opencut mines is as follows-

$$E_j = Q \times EF_j$$

where:

E_j is the fugitive emissions of methane (j) that result from the extraction of coal from the mine(CO_2-e tonnes).

Q is the quantity of run-of-mine coal in tonnes extracted during the year measured

EF_j is the emission factor for methane (j) measured in CO_2-e tonnes per tonne of run off mine coal extracted during the year of measurement. The emission factors for different opencut mines of Australia are provided in Table 2.2.

Table 2.2 Emission Factors for the Production of Coal (fugitive) - Open cut

Activities related to extraction of coal	Emission Factor (Tonnes CO_2-e/Tonne of raw coal)
Open cut Mines- NSW	0.045
Open cut Mines -QLD	0.017
Open cut Mines – Tasmania	0.014
Open cut Mines -Victoria	0.007
Open cut Mines –South Australia	0.007
Open cut Mines –Western Australia	0.017

2.1.5 Fugitive Emissions from Coal Mines Waste Flared

Green house gas emissions from flaring of coal mine waste gas can be estimated by multiplying the quantity of gas flared by the energy content and emission factor for each gas.

$$E_{(fl)ij} = \frac{Qi,flared \times ECi \times EFij}{1000} \times OFif$$

Where:

$E_{(fl)ij}$ is the emissions of gas type (j) released from the coal mine waste gas (i) flared from the mine CO_2-e Tonnes

$Q_{i,flared}$ is the quantity of coal mine waste gas (i) (Cubic meters)

EC_i is the energy content factor of the coal mine waste gas (i) in GJ/cu.m.

EF_{ij} is the emission factor for the gas type (j) and coal mine waste gas (i) in Kg/CO_2–e per GJ . The standard values of ECi and EFij for different gaseous fuels are listed in the NGA factors book

OF_{if} is 0.98/0.995, which is the correction factor for the oxidation of coal mine waste gas(i) flared.

2.1.6 Decommissioned Underground Mines

Fugitive emissions can be estimated from decommissioned underground mines that have been closed for a continuous period of at least 1 year but less than 20 years. If the coal mine waste gas is not captured after decommissioning the following method is used for calculating the GHG emissions. The method for calculating emissions from decommissioned mines is:

Edm = [Etdm × EFdm × (1 − Fdm)]

where:

Edm is the fugitive emissions of methane from the mine during the year measured in CO_2-e tonnes.

Etdm is the emissions from the mine for the last full year that the mine was in operation measured in CO_2-e tonnes and estimated as described under the emissions from underground mines.

EFdm is the emission factor for the mine derived from the emission decay curves (EDC) prepared for the Australian mines.

Fdm is the fraction of the mine flooded during the year, and is the ratio of the rate of water flow as m^3/year into the mine and the mine void volume in m^3.

To ensure consistent application of methods across the industry, the Government has established a working group to work with the coal industry till mid-2009 to develop detailed guidelines on the application of emissions estimation methodologies. These guidelines aim to standardise the application of estimation methods across the industry to provide certainty to liable entities about their reporting obligations.

2.2 Australian GHG Emissions

In 2006, Australia's reported net greenhouse gas emissions were 549.9 million tonnes (Mt), CO_2-e. The combined energy subsectors (including stationary energy, transport and fugitive emissions) were the largest source of greenhouse gas emissions comprising 72.9% (400.9 Mt CO2-e) of emissions.

Total estimated fugitive emissions for 2006 were 34.5 Mt CO2-e, representing 6.3% of net national emissions. Net solid fuel emissions contributed 69.2% (23.9 Mt) of fugitive emissions. Oil and natural gas production, processing and distribution account for the remaining 30.8% (10.6 Mt) of fugitive emissions.

Overall fugitive emissions increased 18.1% (5.3 Mt) between 1990 and 2006, and increased by 6.0% (2.0 Mt) from 2005 to 2006 (Figure 2.1). From 1990 to 2006 fugitive emissions from solid fuels increased by 47.1% (7.6 Mt) and oil and natural gas emissions decreased by 18.1% (2.4 Mt).

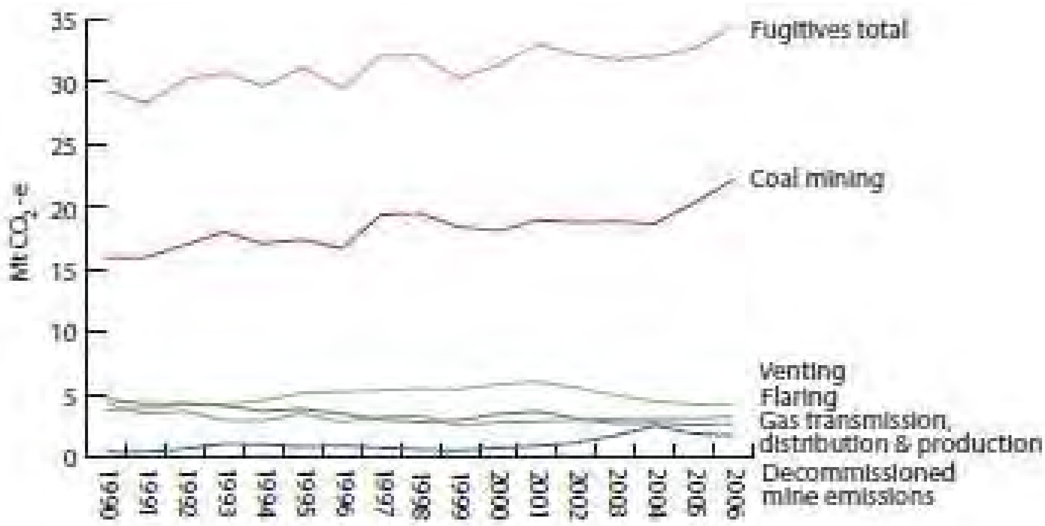


Figure 2.1 CO_2-e Fugitive Emissions by Category, 1990–2006

Solid fuel emissions increased by 8.5% (1.9 Mt) between 2005 and 2006, underpinned by a 3.5% increase in coal production from gassy underground mines. Emissions tend to fluctuate from year to year, depending on the volume of coal mined and the share of gassy underground mines in total production. Mine production of coal has increased from 241 Mt in 1990 to 480 Mt in 2007, an increase of 107%. Since 1990, methane emissions have not grown as fast as activity principally because, since 1998, there has been a decreasing trend in activity from gassy mines while there has been growth in non-gassy mines and surface mines, Figure 2.2 (Australian National Greenhouse Accounts, National Greenhouse Gas Inventory, May 2009). In addition, technologies to recover and utilise or flare CH_4 have been increasingly adopted. CMM emissions reached more than 26 Mt carbon dioxide equivalent (CO_2e) in 2007. These emissions from black coal mining account for 3.1 percent of Australia's total greenhouse gas emissions with ventilation air methane (VAM) being responsible for 64 percent of Australia's coal mine emissions. It is to be noted that this data and trends are an approximation based on estimates rather than individual mine data.

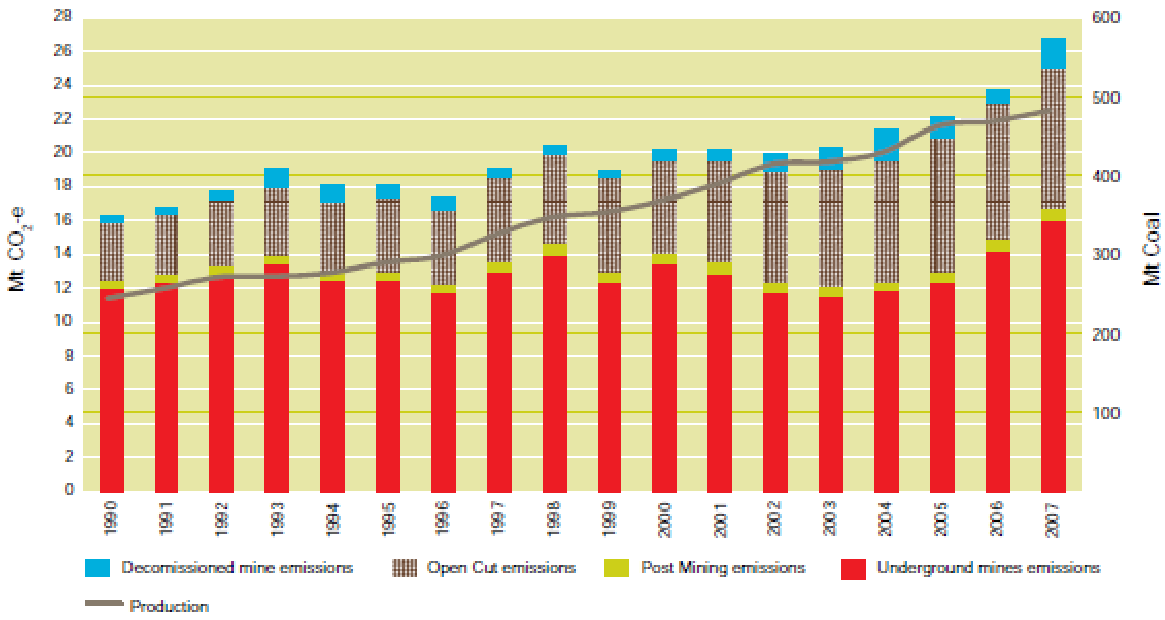


Figure 2.2 Fugitive CO_2-e Emissions from Coal Mining Activities, 1990–2007

The total greenhouse gas emissions from underground coal mines are around 16 to 17 Mt CO_2-e out of the total coal mining emissions of around 26 to 30 Mt CO_2-e in 2007. The total ventilation air methane (VAM) emissions are around 80% of the total fugitive emissions from underground mines. In addition, it is estimated that about 20 m^3/s of CH_4 is drained from the underground coal mines, which equates to around 40% of the total coal mining methane (CMM) emissions.

2.3 Summary of Australian GHG Emissions

The main points arising from Australia's fugitive emissions accounting are as follows:

- Australian Legislation specifies how GHG emissions are to be measured and accounted for. Most importantly, this includes accurate measurement of volumetric flow rates form ventilation and gas drainage systems. The custom and practice of measuring ventilation rates once per month is not adequate.
- Emission accounting includes gas in coal reporting to stockpiles. This may be significant in some mines, particularly those not pre draining gas for outburst control.
- Coal mines can mitigate their emission charges by destroying methane in flares, gas engines or by oxidising methane in ventilation. All these processes incur very significant capital and operational costs that need to be considered against the financial benefit gained. It is therefore very important that mines properly audit their emission streams so that appropriate decisions can be made. In some mines, the do nothing option may be appropriate whilst in others it may be necessary to introduce pre drainage even if not required for outburst control.
- It is important to consider the most efficient methods of increasing gas capture rates and mitigating fugitive emissions from coal mining.

3. AUSTRALIAN COAL MINE GAS CHARACTERISTICS

The purpose of this section of the report is to summarise the range of parameters influencing gas emission from Australian underground coal mines and hence provide a means of quantifying the current and potential future range of gas emission rates together with identifying opportunities for improved mitigation of fugitive emissions. It is also important to be able to place the range of Australian coal mine gas emission rates in perspective with international norms or custom and practice presented in Chapter 4.

Background information contained in this section has been obtained from a number of sources, in particular Hargreaves, 1986, Esterle et al, 2006, Saghafi A, et al, 2008, Thomson et al, 2008 and Williams and Yurakov, 2003. In this respect, it is not the intention to duplicate existing work in this report, rather use it to support conclusions and recommendations for appropriate emission mitigation strategies. It is however an identified objective of this report to explain these issues in basic terms in order to better communicate them to industry stakeholders who may have little or no expertise in the fields of coal mine ventilation and seam gas management.

3.1 Reservoir Characteristics

Coal is a complex, metamorphosed, sedimentary material of wide ranging composition, physical and chemical properties. The range of coal types present today has been formed by burial of a variety of prehistoric vegetable matters, combined with various proportions of soil, minerals and rock particles. The coalification process alters this material under the influence of pressure, temperature and a variety of seam fluids passing though the strata over time. For example, some seams in NSW currently 100 or 200 m below surface may have had a maximum depth of burial of circa 2.0 km and typically 12 m of original vegetable matter is required to form 1 m of coal.

The physical structure of coal seams includes pores, both inter connected and isolated, granular structures, seam plys associated with the differential sequence of deposition and a variable fracture network or cleat. The general orientation of the fracture network, together with joints, will be determined by tectonic stresses applied to the seam at various stages of it's formation. In addition, intrusions of igneous material (dykes) and dislocation of seams through strata movement (faults) also occur.

During the coalification process, water, carbon dioxide and methane are formed from the original composition of hydrocarbons and oxygen present. Carbon dioxide is formed in lower rank coals from excess oxygen but is then normally flushed out, to a greater or lesser extent, by methane formed at a later bituminous or anthracitic stage, Paterson, 1990. Higher rank coals produce about 565m^3 CO_2 and 765m^3 CH_4 per tonne during formation, however, only a relatively small proportion of this gas remains in present day coal seams as free or adsorbed gas, the remainder is lost by migration to atmosphere, seam fluids or adjacent porous strata.

In Australian coal seams, the range of gas contents is from <1.0m^3/t to between 18m^3/t and 30m^3/t with the limit of higher gas contents at depth (currently 400 to 550m below surface) being dependent on CO_2/CH_4 composition ratios.

Various geological structures, such as dykes, may form barriers to secondary introduction of carbon dioxide by seam fluids if they pre date the wash. Consequently, seam gas composition may alter significantly in plan or vertical position at mine site scale which may be a general trend from one gas composition to another or a sharp transition across a single structure, for example the situation found in the Sydney basin, Thompson et al, 2008.

Seam gas exists in two distinct forms, namely free gas and adsorbed gas. Free gas describes those molecules existing in the pores and fracture network of the coal or porous interburden. Porosity being the ratio of total voids to bulk volume and normally ranges 1 to 10 % although some coals have a porosity of up to 20 %, McPherson (1993). As pores also contain water, the degree of saturation effects the available void space and hence the volume of free gas present. It is important to note that free gas also exists in porous non coaly strata adjacent to seams and can form a significant part of the total gas reservoir impacting on a mine, for example that contained in the Bulgo sandstones above the Bulli seam.

The consequence of these various seam gas formation processes and significant variations in geometrical, geophysical and geochemical properties of gas reservoirs present in Australian coal mines, is that a wide range of factors influencing gas emission are present, even on a mine site scale. Consequently, a variety of emission mitigation strategies will be required to provide optimum solutions at individual mine sites and site-specific gas reservoir characterisation is important for decision making.

Further more, it is the experience of authors of this report that many coal mine exploration programmes tend to focus on properties of the working seam with typically only limited pre mining data being available to adequately describe properties of coal seams or porous interburden in the roof and floor of the working section. There are some notable exceptions, for example the pre mining assessment of the Blakefield South project for surface pre drainage, Hennings et al, 2008. Post mining monitoring of full field gas reservoir response to coal extraction, other than subsidence and overall impact on water tables, is also limited or non existent at many mine sites.

3.1.1 Stratigraphic Column and Depth of Mining

Properties of the stratigraphic column determines the volume of coal and porous interburden present within the zone influenced by longwall extraction together with modes of caving and interaction of floor and roof strata. This zone of influence depends on the geophysical properties of interburden and longwall dimensions but is typically taken to be from 50m - 75m below the working section to 100m - 200m above the working section. The zone of influence of pillar panels or narrow (< 150m) longwall blocks will be significantly less and also, with pillar panels, dependent on the degree of extraction.

Whilst the total volume or mass of coal and porous interburden determines the potential volume of seam gas present, the vertical spacing determines the degree to which gas emission will occur due to depressurisation and increased connectivity, firstly during development and then during longwall production.

In Australian conditions, thick and multi seam environments are common in which the vast majority of gas emitted during longwall extraction (and that from the mine) originates from sources other than the working section. For example, a generic section for the Woodlands Hill seam is provided in Figure 3.1 in which there is about 20m of coal within the zone of influence of the 3.0m working seam.

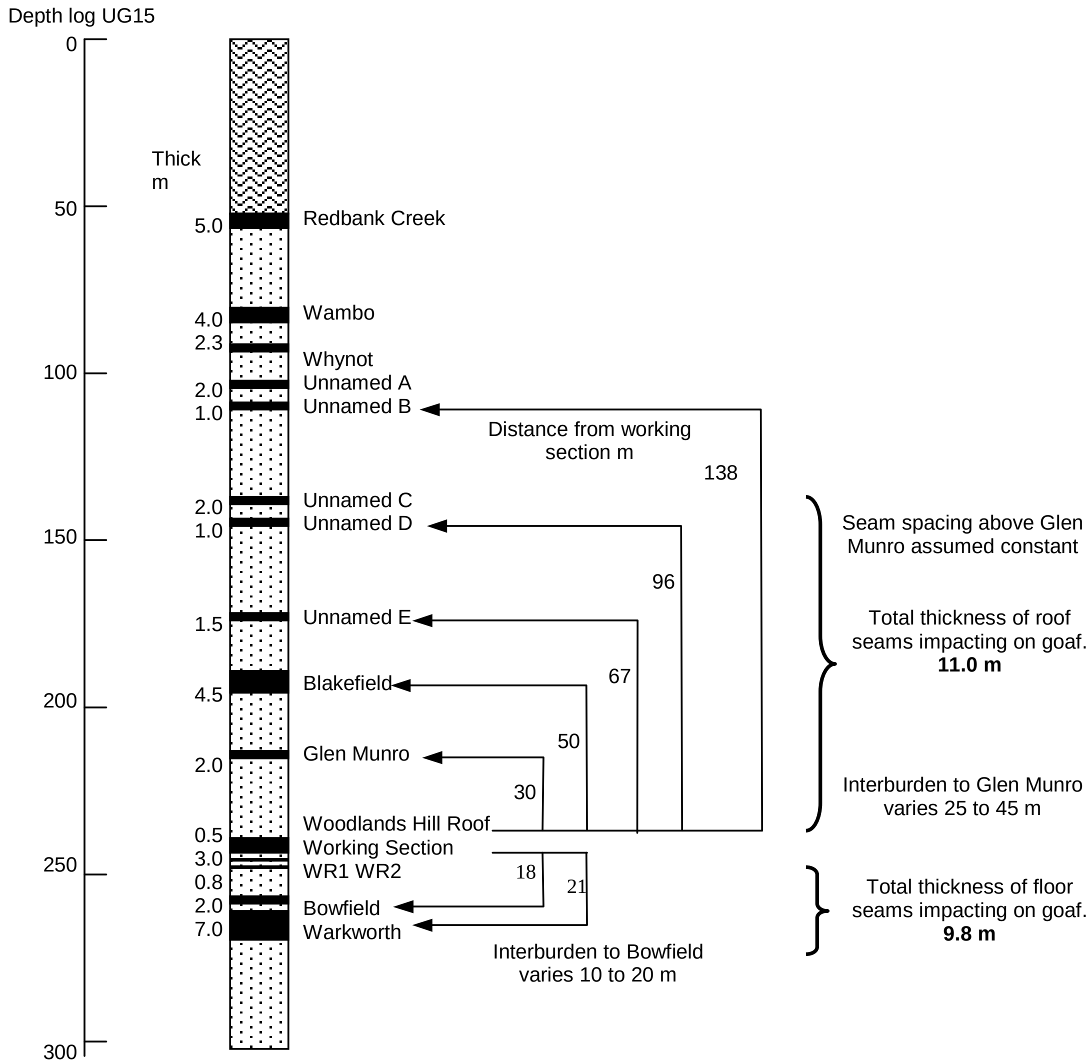


Figure 3.1 Example Stratigraphic Section – Hunter Valley

The range of depths for all Australian longwall mines provides a weighted mean depth of about 300m with deepest mines currently at about 550m, Figure 3.2. Future plans for projects in the Bowen Basin and Hunter Valley will extend the depth of longwall mining to between 700 and 800m below surface sometime from 2015 to 2020. The consequence being that an increasing amount of coal will be produced in regimes of increasing gas content inevitably leading to a potential for increased future fugitive gas emissions i.e. increased m^3 gas emitted per tonne mined.

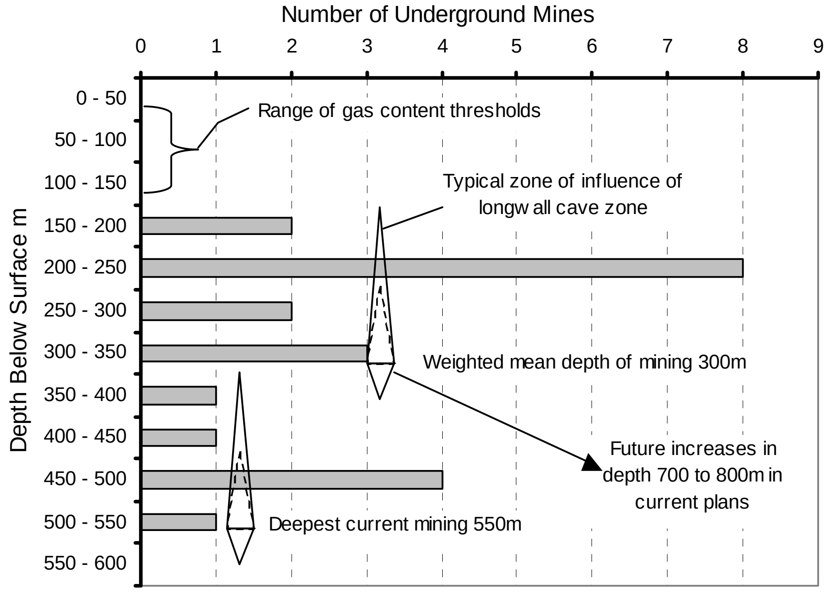


Figure 3.2 Depth of Australian Underground Longwall Mines (2008)

The situation is also exacerbated by the trend in increasing longwall face widths, historically from 150m to 250m, but now to up to 400m. Face width and extraction height effects the degree of interaction with adjacent strata and hence the potential for gas emission. This is a further reason for site specific rather than regional assessment of gas emissions.

3.1.2 Sorption Capacity of Coal

The volume of gas that can be adsorbed onto coal is described by Langmuir isotherm constants (Lama, 1988, Williams and Yukarov, 2006) which vary with gas composition, carbon content of coal (rank), ash content, moisture and temperature. For example, isotherm characteristics for carbon dioxide and methane in a Bowen Basin seam are shown in Figure 3.3.

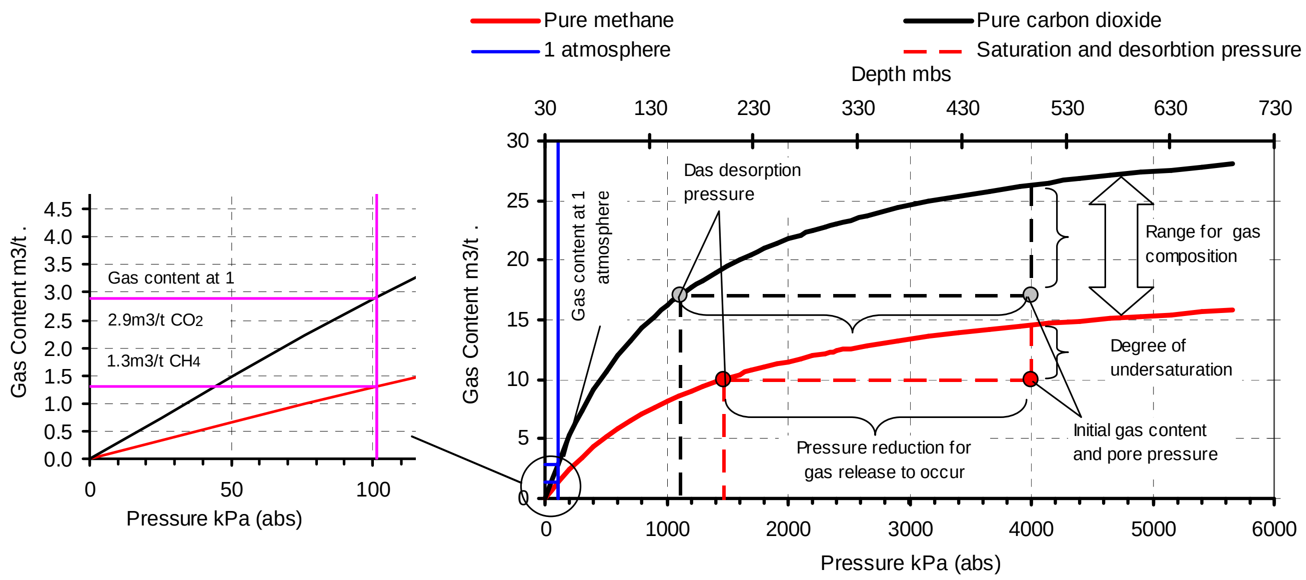


Figure 3.3 Isotherm Characteristics

Pertinent issues to consider are as follows;

- Isotherms are usually determined in laboratory conditions using finely crushed samples which have gas re introduced under pressure. Due to differential adsorbtion rates and curve fitting used to determine isotherm values, this is not necessarily the same as a native isotherm for coal under virgin conditions, particularly where both methane and carbon dioxide are present or diffusion rates are low.
- When virgin state pore pressures exceed the desorption pressure for the gas content present in a coal seam, as is usually the case in Australian conditions, the pressure must be reduced to the saturated gas pressure before gas release occurs. This is an important factor determining rib emission, pre drainage design characteristics together with the degree of gas emission from roof and floor seams adjacent to longwalls.
- The pressure reduction required for desorption to occur increases with carbon dioxide content of seam gas. This makes it progressively more difficult to pre drain seam gas as carbon dioxide content increases.
- At a partial pressure of one atmosphere (i.e. coal surrounded by gas with the same composition at atmospheric pressure), the residual gas content of methane and carbon dioxide are of the order $1.3m^3/t$ and $2.9m^3/t$ respectively. If the partial pressure is reduced, for example by passing fresh air over the coal surface, then the residual reduces, eventually to zero. This is however, the residual gas content only for finely ground coal fragments used to determine the isotherm.

  The consequence of this is observation is that coal reporting to surface will eventually release most gas contained within it and coal within a sealed goaf will release less gas if surrounded by an atmosphere of similar composition to that in the seam.

In an investigation into the distribution of gas contents using 1,500 samples from 19 states in the US, Diamond et al, 1986, notes that the residual gas content of cores using slow desorption methods can be up to 40 of 50% of the total gas content present, in particular for relatively low-rank (High volatile-A bituminous) blocky coalbeds. In contrast, friable, higher rank (Medium to Low volatile) bituminous coalbeds typically had less than 10% residual gas.

This would be a significant issue for decision making if larger residual gas contents reduced surface emissions and hence the need for pre drainage in mines that would other wise not require it. It is also significant in the calculation of desorption time constants of coal and the overall gas emission potential of gas reservoirs. With most Australian mines now using the quick crush method, this is a factor that is not now usually evaluated. Saghafi, 2001 does however provide data indicating that this true residual gas content is only of the order $0.85m^3/t$ for Bowen basin coal seams. This is in fact consistent with the value indicated by the isotherm for the same coal.

3.1.3 Seam Gas Contents and Composition

Seam gas content describes the volume of gas contained within a unit mass of coal, m^3/t which, for comparison, is normally standardised to NTP $20^{O}C$ and 101.325 kPa although Australian GHG legislation uses NT = $15^{O}C$.

The potential volume of gas present in an absorped state is described by Langmuir isotherm constants for the coal and seam gas composition in question (dependent on coal properties such as ash content) and the potential, albeit much smaller, volume of gas in a free state determined by the unsaturated porosity and seam or strata pressure. The actual volume of gas present in coal seams today depends on the range of pressures that have been applied to the seam during it’s burial history. This may include periods of low pressure during which time much of the gas present would have been lost.

Seam fluid pressures depend, in the main, on the connected hydrostatic head which, in most but not all Australian conditions, can be estimated from depth below the RL of the standing water table. The increase in fluid pressure will therefore be approximately 1.0MPa per 100m below the water table, which, in areas of relatively flat surface topography, is typically 20 to 50m below surface. This means, for example, that Bulli seam workings at 500m would be expected to have a virgin fluid pressure of 4.0 to 5.0Mpa. This generally linear relationship may be altered by aquifers with elevated recharge points (e.g. Hunter Valley) or where seams outcrop or are under significantly reduced overburden depths at or below the RL of the working section (e.g. South Coast escarpments).

Seams exposed to periods of low fluid pressure during their burial history may have desorbed some or nearly all gas present during that time, leaving a seam with a low gas content for depth of burial, regardless of present day fluid pressures. This gives rise to under saturated conditions. Conversely, seams that have not been subject to such reductions in fluid pressure, possibly due to being overlain by low permeability strata, may have present day gas contents at or in excess of the saturated gas content provided by the Langmuir isotherm

The main issue arising from these relationships, coal properties and hydro-geological processes is that the gas content and composition of coal seams being mined today may be very different at similar depths even over relatively short distances. For example, coal seams in the Cessnock area of NSW are reported to have gas contents below 1.5m^3/t at 700m depth where as the gas content of seams in the Singleton area (some 40km distant) range up to 10m^3/t at 350m depth. It is also the case that the feasibility of mitigation strategies such as pre drainage will be dependent on the location and characteristics of gas bearing coal or non coaly strata.

Data provided by, amongst others, Esterle et al, 2006 and Williams and Yurakov 2003, demonstrates that a wide range of gas contents and composition exist in Australian conditions, broadly < 1.0 m^3/t to about 18 m^3/t (CH_4 rich) and 30m^3/t (CO_2 rich), Figure 3.4. Maximum gas contents are typically limited by the isotherm for the gas composition present and most Australian gas reservoirs are under saturated to various degrees at current depths of mining.

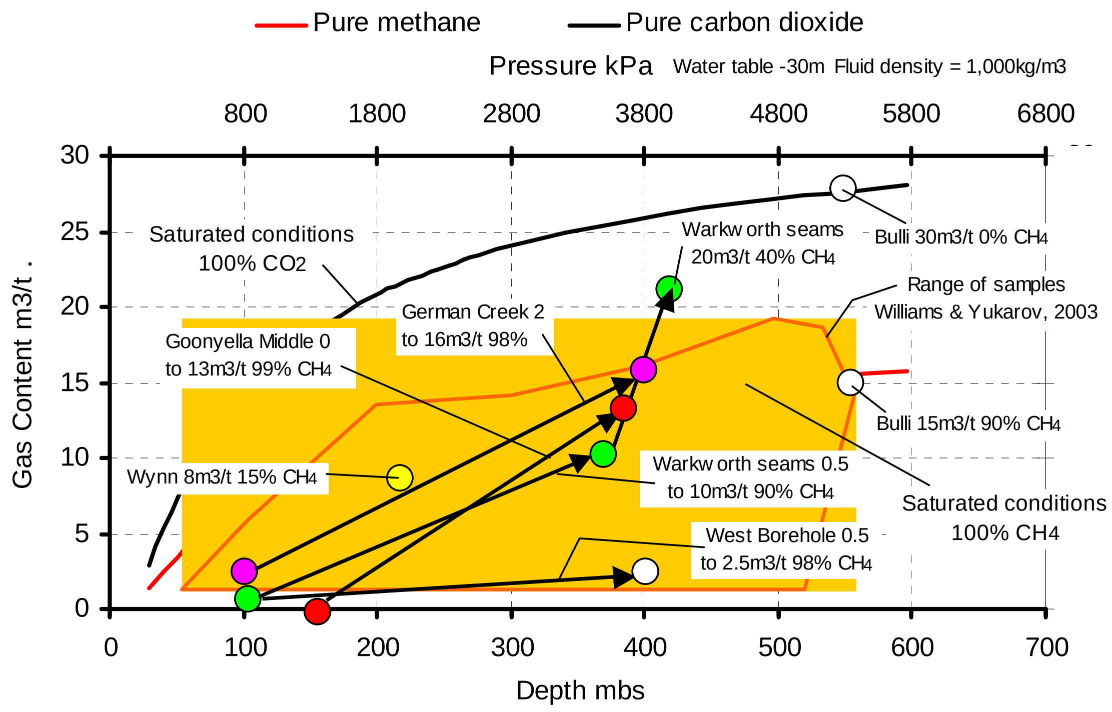


Figure 3.4 Gas Contents of Australian Coal Seams

It is important to note that, in seams such as the Goonyella Middle and German Creek, increasing gas content with depth relationships are also associated with an increase in the degree of gas saturation. This suggests that not only will the total volume of gas increase with depth but so too will the rate of emission, including that from coal and porous interburden adjacent to working sections. It is also the case in some seams, that extending mining depths beyond about 450m will take gas contents in to supersaturated conditions or regimes of saturated gas present at increased pressure, as is currently the case in some deeper areas of the Bulli seam.

With respect to GHG emissions and management of seam gas in general, the nature of gas reservoir characteristics at increasing depth, increased longwall production rates, extraction heights (including top coal caving) and block widths will inevitably increase the rate and total volume of future emissions. That is, future underground coal mine production in Australia will produce more seam gas per tonne, on average, than at current depth of mining.

3.1.4 Gas Content of Other Strata

The density of coal with zero ash content is about 1.29t/m^3 and the density of ash depends on it's actual composition but is typically about 2.3t/m^3. There is therefore a reasonably linear relationship between ash content (of similar composition) and density. For example, Figure 3.5 shows the relationship between density and ash content for the German Creek seam.

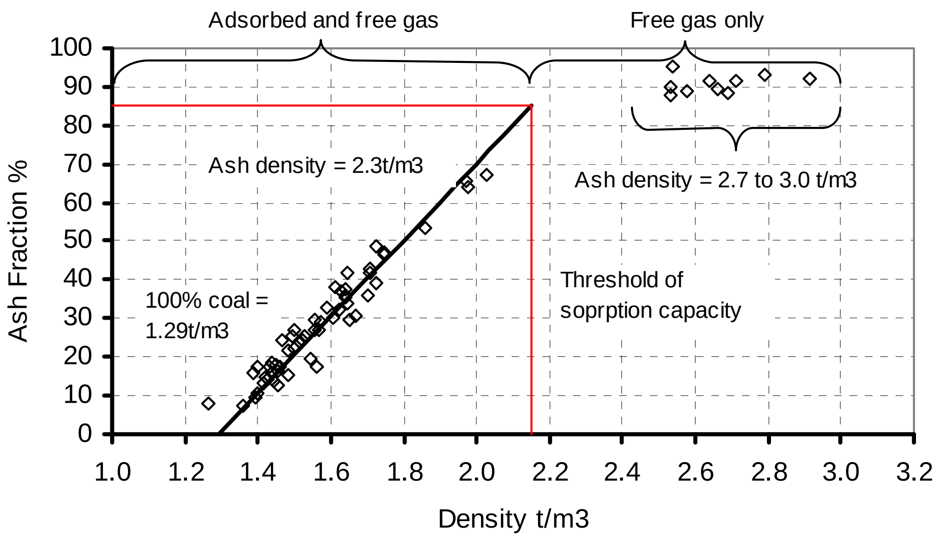


Figure 3.5 Ash Fraction – Density Relationships

The sorption capacity of coaly strata reduces with increased ash content, as reflected in a reduced Langmuir volume and pressure with a threshold of zero sorption capacity at about 85% ash proposed by Williams and Yurakov, 2003. For the data provided in Figure 3.5, this suggests a cut off of 2.1 to 2.2t/m^3 beyond which only free gas will be present. For ash with a higher density, this cut off point will increase proportionally.

Based on the assumption that all ash has a zero sorption capacity, the theoretical and observed reduction in gas content with increased ash content is shown for samples at similar depths in Figure 3.6 (Hunter Valley data obtained from Saghafi A, et al, 2008).

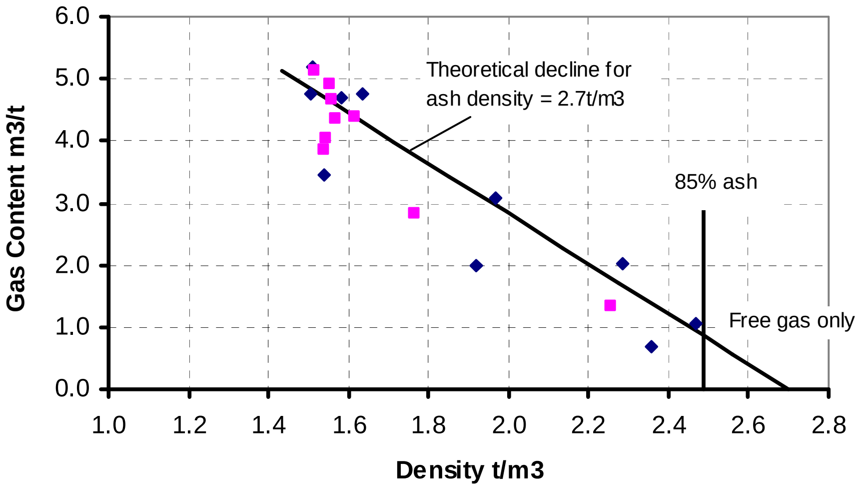
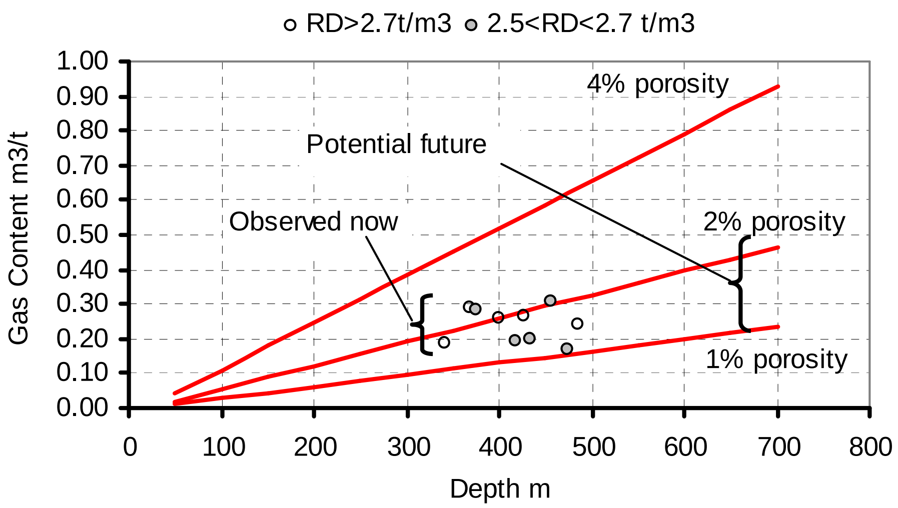


Figure 3.6 Observed and Theoretical Change in Gas Content with Strata Density

From a coal mine gas emission assessment point of view, the main issue arising from these relationships is the potentially significant volume of adsorbed gas being present in coal seams from which few if any exploration phase core samples are taken. These relationships do however provide a means of estimating the gas content of this strata with a reasonable degree of confidence.

The theoretical and observed gas content of non or low carbonaceous interburden with densities greater than 2.5t/m^3 adjacent to the Moranbah coal measures are shown in Figure 3.7. This analysis suggests an unsaturated porosity of 1 to 2% and a gas content of 0.2 to 0.3m^3/t. With respect to future deep workings, it is important to note that the potential contribution of free gas in interburden will approximately double from 350m to 700m depth.



Water table = 30m, strata density = 2.7t/m^3, gas composition 100% CH_4

Figure 3.7 Free Gas Content of Interburden

Whilst calculation of potential free gas contents at mine sites should consider site specific conditions, for example the Bulgo sandstones (Armstrong, 2006), it is important to recognise the potential contribution to longwall gas emission. As an approximation, if the ratio of coal to interburden mass in the cave zone were 1:10 and the ratio of gas content of coal to gas content of interburden were (typically at 300m) about 40:1, then the potential contribution of porous interburden to longwall gas emission would be of the order 25%, i.e. significant. As this free gas is released immediately depressurisation occurs, this could also explain some of the variability or gas emission peaks observed in emission rates and higher than expected emission rates in some mines.

3.1.5 Gas Reservoir Size and Potential Emission

The size of the gas reservoir is most conveniently calculated in terms of m^3 gas per m^2 plan area from the sum of gas contained in individual members of the stratigraphic section within some defined distance from the working seam, typically 100m and 200m above and 50m to 70m below. This provides a measure of the potential gas release per m^2 of longwall extraction. For example, if the sum of coal thickness was 10m with a density of 1.45t/m^3 and the average total gas content was 5m^3/t then the gas reservoir size would be 10 x 1.45 x 5 = 72.5m^3/m^2, including the residual or Q3 content.

The gas reservoir size in some Australian mines are shown in Figure 3.8, ranging typically from <5 to 90m^3/m^2 but with some locations as high as 150 to 200m^3/m^2 where there is potential ingress from large gas reservoirs, for example Bulgo sandstones above and Wongawilli seam below the Bulli seam. The large variation reflects the range of gas contents at depth of mining combined with the variation in coal seam thickness present in the roof and floor of working sections.

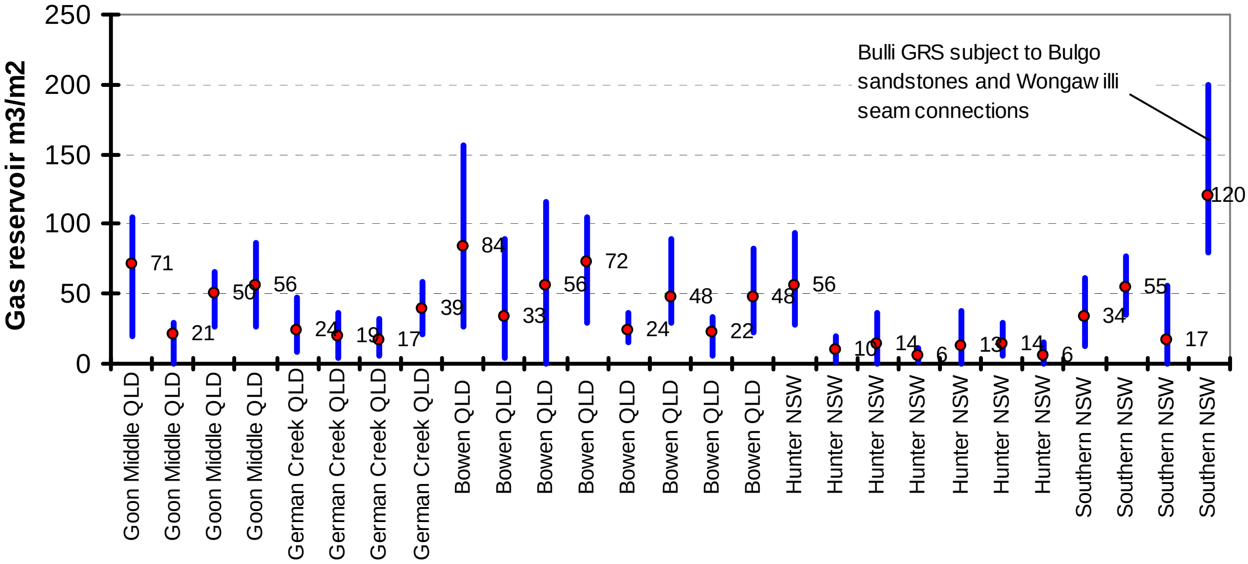
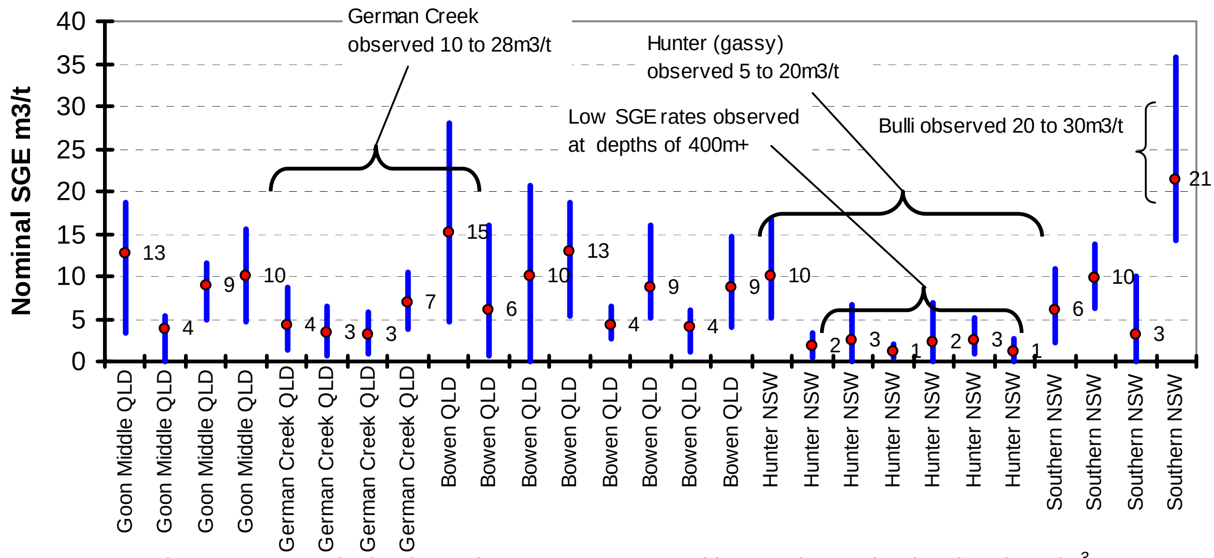


Figure 3.8 Range of Australian Coal Mine Gas Reservoir Sizes (Courtesy Geogas)

The term specific gas emission (SGE m^3/t) refers to the quantity of gas released from the gas reservoir per tonne mined which is a useful method of relating gas emission to production rate. The potential specific gas emission rates arising from these gas reservoir sizes can be obtained by relating them to the mean extraction height of 3.2m for Australian longwall mines. For example, at a working seam extraction height of 3.2m and a density of 1.45t/m^3 the potential specific gas emission with 100% gas release from a reservoir containing 72.5m^3/m^2 would be 72.5/(3.2 x 1.4) = 16.2m^3/t.

In reality, not all of the gas reservoir will be released, and of the gas released the majority will be released during production with a smaller but significant fraction released after production has ceased. These issues are discussed further below, however, it is informative to compare these nominal maximum specific gas emission rates (m^3/t) with the range of observed values, Figure 3.9, and the potential longwall gas emission rates (m^3/s) that would occur at various production rates, Table 3.1.



Based on gas reservoir size from Figure 3.8, a 3.2m working section and a density of 1.4t/m³

Figure 3.9 Nominal SGE Rates For Range of Australian Gas Reservoir Sizes

Table 3.1 Nominal Longwall Gas Emission Rates

Annual Production Mtpy	Gas Emission Rate (m3/s) for Average SGE of 1 m3/t	5 m3/t	10 m3/t	15 m3/t	20 m3/t	25 m3/t	30 m3/t
3	0.1	0.6	1.3	1.9	2.5	3.1	3.8
5	0.2	1.0	2.1	3.1	4.2	5.2	6.3
7	0.3	1.5	2.9	4.4	5.9	7.3	8.8
9	0.4	1.9	3.8	5.7	7.5	9.4	11.3
	Typical			High		Highest	

With respect to Australian GHG determinations, it is important to understand the relationship between these calculations and the CO_2-e emission factors employed for estimating emissions from underground coal mines, section 3.1.3 above. The rates are 0.302 tCO_2-e per tonne mined for gassy mines and 0.008 tCO2-e per tonne mined for non gassy mines. For example, the relationship for gassy mines is as follows (Figure 3.10).

0.305 tCO_2-e per t = 0.305/(21 x 1000) = 14.38 $kgCH_4$ per t = 14.38 / 0.681 = 21.1 m^3 CH_4 /t

It is understood that these values are intended to represent total mine emissions (development plus longwall plus sealed areas) including gas reporting to surface in production coal. Quite clearly, values obtained from this method may not be appropriate for many of the mines identified in Figure 3.8.

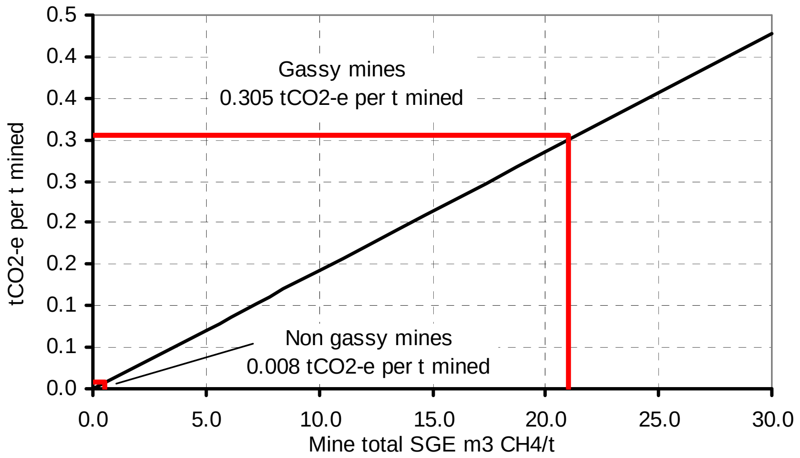


Figure 3.10 Relationship Between SGE ($m^3/t\ CH_4$) and tCO_2-e per t Mined

There are two distinct issues arising from this broad analysis, firstly the rate (m^3/s or l/s) at which gas is emitted into the workings during longwall production determines ventilation and gas drainage system requirements, and secondly, the total volume of gas emitted from the disturbed reservoir, both during and after completion of production, determines potential fugitive gas emission rates. In addition to this will be gas emitted from production coal on surface stockpiles or during surface transportation.

Prior to concerns being raised about fugitive gas emissions and due to the relatively low cost of power in eastern Australia, the design focus of most coal mine ventilation and gas drainage systems has historically been to only manage the rate of gas emission during production. Provided gas emission from sealed areas could be managed by main return ventilation capacity, as is normally the case, total emissions from sealed areas or even abandoned mines has not been a significant management concern. This is, of course, no longer the case and a different monitoring and management approach will be required, in particular, improved assessment of total life of mine gas emission from the gas reservoir in place prior to mining.

3.1.6 Desorption Rates

In order to assess the rate at which gas will or may be released from both working and non working seams it is necessary to quantify desorption rates which are dependent on;

- Size of intact coal “fragments” within the connected coal matrix.
- Diffusion constants for the coal, which are in turn dependent on coal properties, composition and moisture content.
- Gas content and gas composition.

and are important for the following reasons;

- Gas emitted from production coal underground will normally report to intake airways (unless a homotropal maingate belt is employed) and therefore add to the ventilation gas load in any event.
- Gas from production coal not emitted underground will be released on surface and therefore not be directly included in the mine’s overall emission balance by direct measurement, as it would be if released into the ventilation or gas drainage systems. It does however have to be included in the mine’s emission balance under method 4 – extraction coal (NGER, 2008).

- Coal in the immediate floor or roof of the working section will degas at a similar rate to production coal, however, more remote seams will degas at progressively slower rates dependent on their distance from the working section.

Typical results for core sized samples provided in Table 3.2. The most significant parameter in this respect is the desorption time constant Tau (time taken for 63% of gas present to desorb) which should be compared with the typical residence time of production coal in mines of 30 to 60mins depending on conveyor belt lengths and speeds, Table 3.3.

Table 3.2 Rate of Gas Desorption from Production Coal

(1) TDGC m3/t	(2) IDR30 m3/t	(3) Tau Days	(4) Constant K	(4) Ratio D/d2	Fraction Gas Desorbed In Time (mins)					Fraction Gas Desorbed In Time (days)				
					5	10	20	60	120	1	7	14	21	35
1	0.03	30.0	0.0007	9.64E-09	1	2	2	4	5	18	45	61	71	83
2	0.07	16.1	0.0009	1.69E-08	1	2	3	5	7	24	58	74	84	93
3	0.15	10.3	0.0012	3.05E-08	2	3	4	7	9	31	72	88	94	99
4	0.26	7.3	0.0015	4.96E-08	2	3	5	8	12	39	83	95	99	100
5	0.39	5.6	0.0018	7.40E-08	3	4	6	10	14	47	91	99	100	100
6	0.55	4.5	0.0022	1.04E-07	4	5	7	12	17	55	96	100	100	100
7	0.75	3.7	0.0025	1.38E-07	4	6	8	14	20	61	98	100	100	100
8	0.97	3.1	0.0029	1.79E-07	5	6	9	16	22	68	99	100	100	100
9	1.22	2.7	0.0032	2.24E-07	5	7	10	18	25	73	100	100	100	100
10	1.51	2.4	0.0035	2.75E-07	6	8	11	20	27	78	100	100	100	100
11	1.82	2.1	0.0039	3.30E-07	6	9	12	21	30	82	100	100	100	100
12	2.16	1.9	0.0042	3.92E-07	7	10	14	23	32	86	100	100	100	100

Notes

1/ TDGC is Q1+Q2+Q3

2/ IDR30 is from Geogas relationship for Bowen Basin coal

3/ Tau is an approximation from standard core size

4/ K and D/d^2 assumes square root time relationsip applies

Shaded area not normally mined due to outburst precautions

Table 3.3 Approximate Underground Residence Time of Coal (mins)

Maingate + Mains Belt Length m	Belt Speed 3m/s	Belt Speed 4m/s
2,000	11.1	8.3
4,000	22.2	16.7
6,000	33.3	25.0
8,000	44.4	33.3
10,000	55.6	41.7

With respect to gas reservoir assessment, most gas content analysis techniques employed in Australia (quick crush or long term) allows the time constants for coal to be determined with confidence. It is also reported (Airey, 1968) that although the desorption rate should theoretically be proportional to the square of the size of a coal lump, the relationship does not hold for larger (>>6mm) lumps where the size of the coal lump is significantly larger than the size of the matrix components. That is, it is a reasonable approximation to apply core sample time constants to all production coal and coal contained within the immediate goaf at atmospheric pressure.

The most significant operational issues arising from this analysis are;

- For all gas contents at time of mining (highest is currently circa 7.5m^3/t in Hunter Valley) only 5% to 15% of gas will be emitted underground for typical coal residence times.
- The effective rate of gas emission is obtained from the product of the mass flow rate of coal and gas content, for example a main conveyor carrying 1,000 tph coal with a gas content of 3.6m^3/t CH_4 is transferring gas to surface at a rate of 3,600m^3/h or 1.0m^3/s but the fraction of this gas released depends on residual gas content and time. The only way to avoid this situation is pre drainage to very low gas contents or to somehow capture this gas on surface.
- These values obtained for Australian coal seams are consistent with those in the US where studies were undertaken to determine gas management hazards in silos and bunkers, Matta et al, 1978, Figure 3.11. This is therefore a global issue for all underground coal mines.

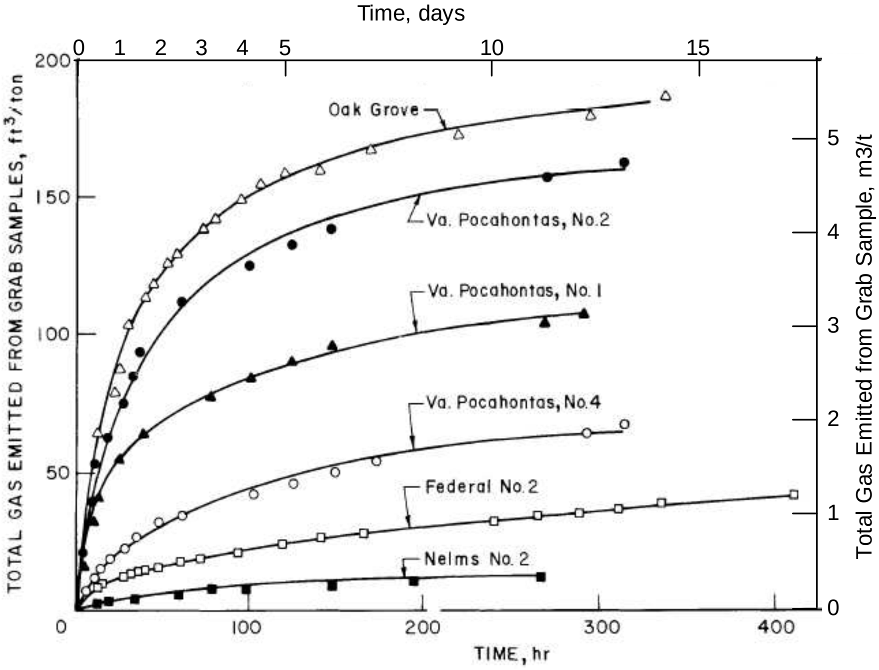


Figure 3.11 Gas Emission from US Longwall Coal in Silos

- For coal remaining in the immediate goaf, or seams in close proximity to the working section, most gas will be emitted within 2 to 3 weeks or 2 to 3 pillars at typical production rates. This has an impact on the performance of goaf drainage holes and management of face environments i.e. a very significant proportion of gas emission occurs in relatively close proximity to the face line and is therefore in close proximity to essentially fresh air. This makes it difficult to capture and reticulate at acceptably high methane concentrations.
- In thick seam environments, removing additional coal from the immediate roof by top coal caving would significantly reduce underground emissions.

For more remote seams, the effective time constant will increase due to increased stress and reduced permeability. This determines both the rate and duration of gas emission from more remote seams together with that occurring from sealed areas and abandoned mines.

3.2 Development Seam Gas Emission

For a given gas content and permeability, the quantity of seam gas impacting on development increases with seam thickness, particularly when the development extraction height is significantly less than that of the seam. In Australian conditions this ranges from the development height being the total seam height in, for example, the Bulli and German Creek seams, to the development height being 3.0m to 4.0m in seams of 6m to 20m, for example, the Goonyella and Warkworth (Wynn) seams.

With consideration to typical outburst threshold limits, development normally takes place in gas contents less than 7 to 8m^3/t with the highest know gas content currently being developed between 7 and 8m^3/t in the Hunter Valley. The range of rib emission rates encountered in gassy Australian conditions is shown in Figure 3.12.

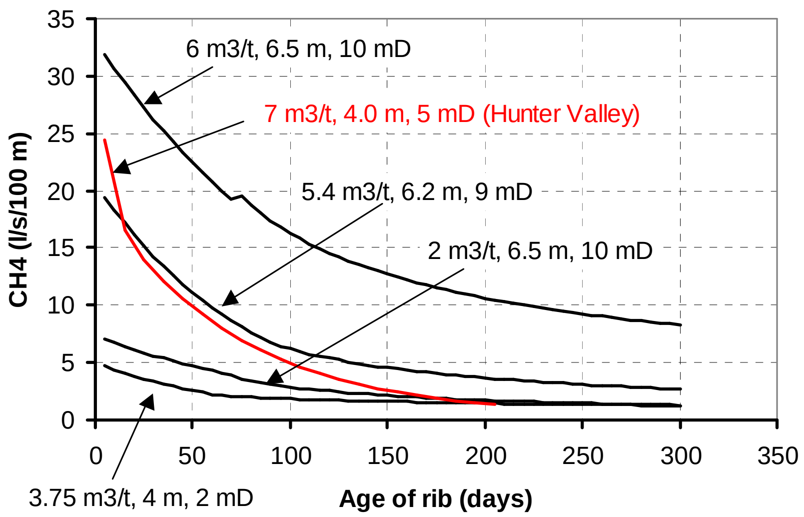


Figure 3.12 Example Rib Emission Rates (Geogas and Moreby)

The characteristic rib emission decay curves determine the gas emission profile of development headings with respect to rate of advance and length. The most significant feature is that the characteristic decay periods, from peak initial to residual gas emission rates, are of the order 150 days or about 5 months. This compares with a typical two heading gate road panel advance rates of 100m to 150m per week or 2 to 3km in 5 months i.e. rib emission decays to residual levels within the life of most gate roads making management of gas emission issues significant during, rather than after, the development phase. It is also for this reason that it is normally acceptable to convert a recently completed gate road to a double intake maingate without, in NSW, problematic gas concentrations (>0.25% CH_4) arising at the longwall hazardous zone.

On a two heading basis the rib emission rate that can be managed by ventilation is dependent on the limits that apply. In Queensland this is normally 0.5%CH_4 at the NERZ/ERZ boundary at some point in the intake and 1.0%CH_4 for use of diesel equipment in returns. In NSW the 0.25%CH_4 limit still applies at commencement of the hazardous zone defined as a location 100m outbye the last completed cut through.

With respect to carbon dioxide, the current general body limit is 0.5% CO_2 in Queensland and 1.25%CO_2 in NSW. With the exception of development in the Wynn seam, it is understood that most coal mines do not encounter problematic carbon dioxide emissions during development.

Table 3.4 provides the range of two heading ventilation rates employed in Australian mines together with limiting methane dilution capacities. The issue to note is that, with acceptable air velocity, it is possible to manage a total 600 to 700l/s CH_4 in most development profiles employed in Australian coal mines. Minimum ventilation rates are determined by requirements for auxiliary fans (typically 20 to 30m^3/s per fan at the last cut through) and operation of diesel equipment (0.06m^3/s per rated kW).

Table 3.4 Two Heading Gateroad Ventilation and Gas Management Capacity

	Total Ventilation m3/s	Profile 2.5 x 5.2 Velocity m/s	Profile 3 x 5.2 Velocity m/s	Profile 3.5 x 5.2 Velocity m/s	Methane dilution capacity at Limit (1) 0.25 CH4%	Limit (1) 0.5 CH4%	Limit 1.0 CH4%
Low	30	2.3	1.9	1.6	75	150	300
	40	3.1	2.6	2.2	100	200	400
Typical	50	3.8	3.2	2.7	125	250	500
	60	4.6	3.8	3.3	150	300	600
High	70	5.4	4.5	3.8	175	350	700
	80	6.2	5.1	4.4	200	400	800
Extreme	90	6.9	5.8	4.9	225	450	900

The operating envelope of actual rib emission rates is shown in Figure 3.13 using, by way of example, a 5km two heading gate road developed at 45m linear per day with high and low rib emission decay curves.

This analysis demonstrates that the current rib emission operating envelope, in most Australian conditions, is such that ventilation can manage rib emission rates in blocks of 2 to 3km in length without having to pre drain other than for the mitigation of the risk of outbursts. Blocks of 5km in length can also be accommodated at lower rib emission rates or where the ventilation capacity is increased in larger development profiles. The exception to this rule has occurred in a few mines with very high permeability (>>50mD) and gas contents of 5 to 7m^3/t.

Problems are likely to occur first in NSW gate road development where higher rib emission rates in the most inbye one or two pillars results in methane concentration exceeding 0.25% on entry to the hazardous zone. In these circumstances exemption from this part of the regulations has been sought and pre drainage may be avoided by moving the hazardous zone boundary further outbye. In Queensland, problems associated with gas concentrations on cutter heads are likely to occur if the provision to raise NERZ/ERZ boundary methane concentrations to 0.5% are employed, particularly if the boundary is moved further outbye.

To minimise GHG emissions during development, the options are pre drainage and or VAM oxidation to reduce gas emission to atmosphere and pre drainage to reduce the gas content of coal reporting to surface.

Increasing block lengths combined with increased development rates necessary to support increased longwall production will inevitably increase rib emission rates and total development emissions in un drained conditions. In these circumstances, it will be necessary to introduce working section pre drainage in situations where it is currently not required. Otherwise, custom and practice in Australian coal mines is to use ventilation to avoid pre drainage of the working section unless required for mitigation of the risk of outbursts.

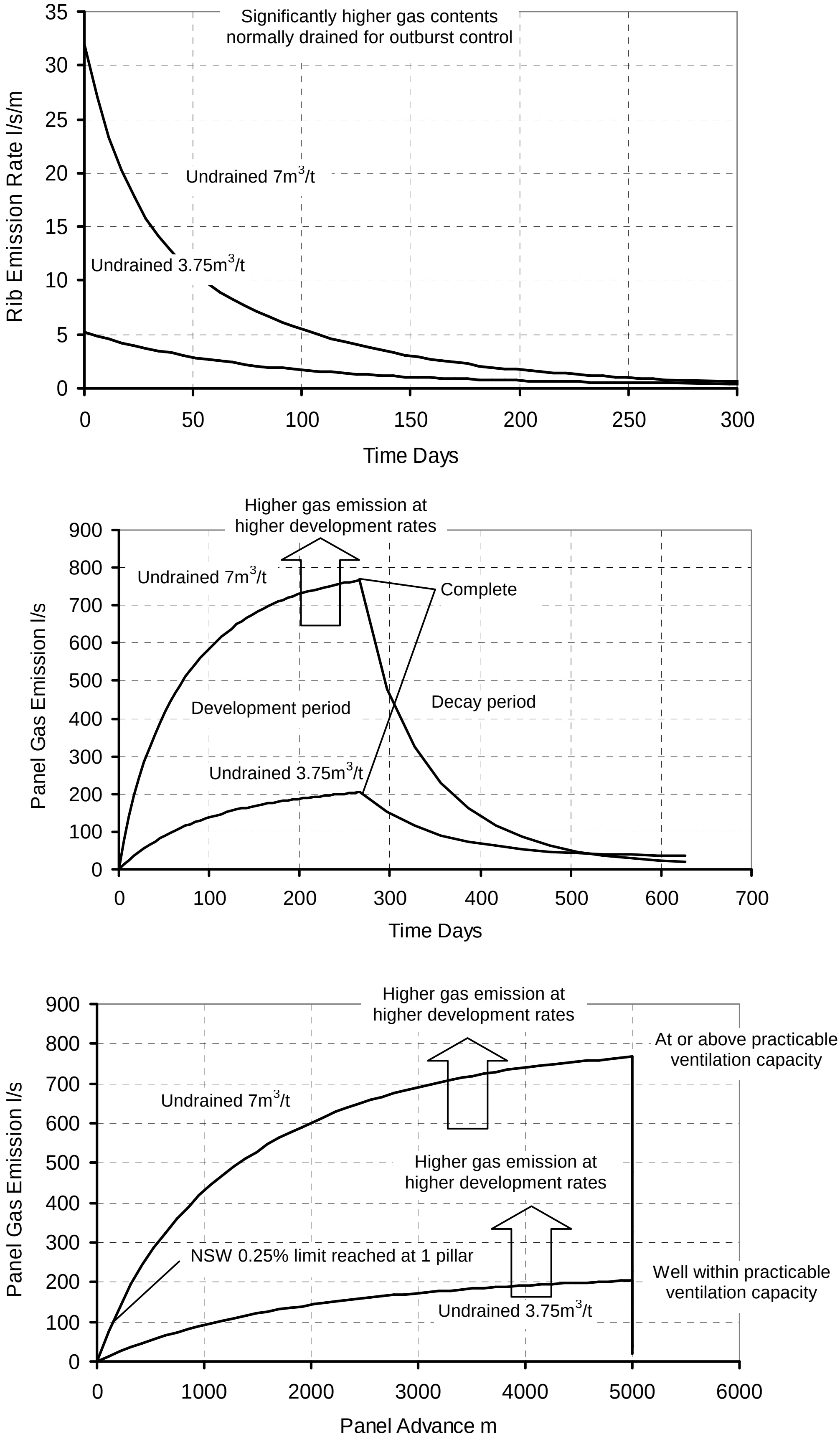


Figure 3.13 Australian Operating Rib Emission Envelope

(5km 2 hdg gateroad at 45m linear advance per day)

3.3 Longwall Seam Gas Emissions

The volume and rate of gas emission into longwalls is dependent on;

- Volume of gas in the reservoir affected by longwall extraction.
- Gas content and degree of saturation of coal seams.
- Desorption time constants for coal seams.
- Distance of coal seams in the roof and floor of the working section.
- Geo mechanical properties of interburden influencing permeability and degree of depressurisation.
- Gas content of porous interburden.

The key planning values to determine are the rate (m^3/s) and total (m^3) of gas emission during production and after production from sealed areas. As described above, it has been normal practice to only focus on production phase gas emission in order to avoid or mitigate gas constrained production.

There are a large number of techniques available for predicting the rates of longwall gas emission (Curl, 1978, Creedy et al, 1997) ranging from simple geometrical models describing the degree of gas emission from surrounding strata (Figure 3.14) to modern finite element models (Ashelford D, 2003, Guo et al, 2006).

For the purposes of this report it is important to note that, although somewhat simplistic, early geometrical models are generally consistent with more recent techniques and were, although specific to the conditions for which they were developed, largely based on field observations giving some credence to their validity. Modern modelling techniques will not significantly alter the net result but can be more readily tailored for site specific conditions.

With consideration to the concept of gas reservoir size described in section 3.1.4 above, the issue is to determine what fraction of the gas reservoir present will actually be emitted. Gas will desorb from it's virgin state to a point on the isotherm determined by the degree of depressurisation that occurs due to extraction of the working section. This fact is not included in the geometrical models where a percent released value is obtained from the geometry and would result in errors where gas contents are significantly greater or less than those for which the model was created For example, if a seam contains 1.5m^3/t it will emit a significantly lower fraction of gas present than a seam containing 6m^3/t if the residual saturated gas content were 1.0m^3/t in both cases. It is however valid to determine the amount of gas that could be released by the difference in virgin state and post mining fluid pressures.

Post mining fluid pressures were, and still are, employed in the European coal industry for this purpose (Noack, 1997), Figure 3.15. These results are specific to deeper European coal mines but are generally consistent with other work in this area (Gale, 2001, Liu and Elsworth, 1999, Lunarzewski, 1998). That is, the working seam pressure must reduce to atmospheric which sets the absolute pressure in the open cave zone above which the fluid pressure will revert to, or close to, pre mining fluid pressure unless the longwall is close enough to disturb the surface water table, as is the case in shallow Australian coal mines. In the floor, Noack's model indicates pre mining fluid pressures will be reached within about 60m that is again consistent with most geometrical models.

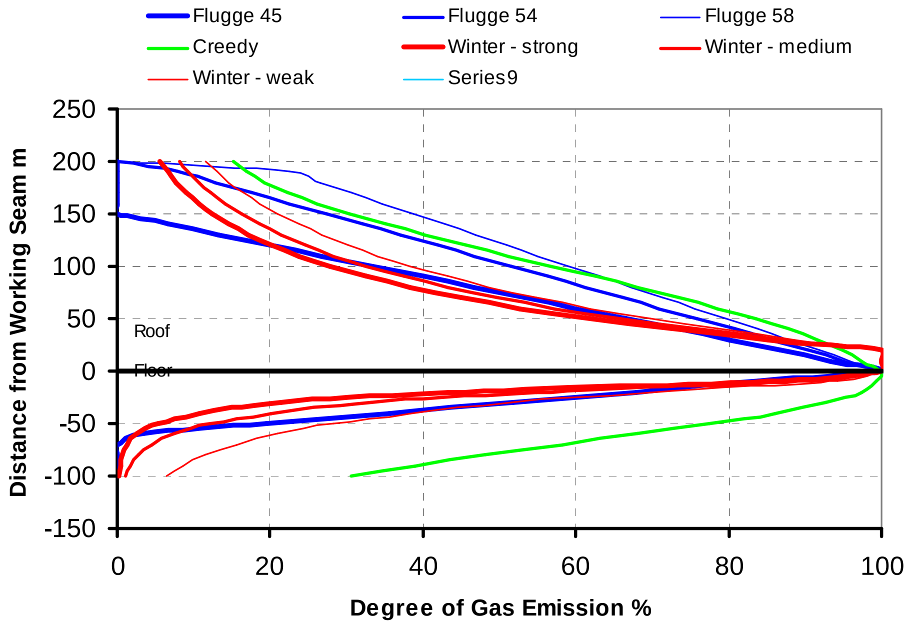


Figure 3.14 Degree of Gas Emission – Empirical and Geometrical Models

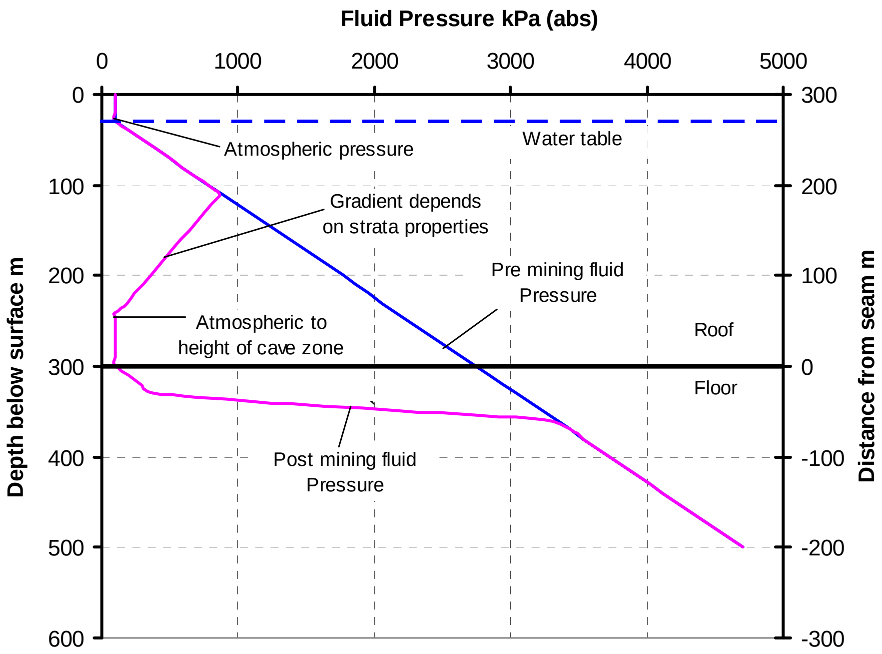


Figure 3.15 Pre and Post Mining Fluid Pressure

The main issue arising from these models is the inevitability of gas being emitted from roof and floor strata within the zone of influence of the longwall. There will be variations due to various geo mechanical properties of strata but the depressurisation occurring within the cave zone will result in gas desorbing to residual values (circa 1.0m^3/t for methane and circa 2.0m^3/t for CO_2) and some higher value in more remote seams. This provides a means of calculating volumetric emission, the next issue is to consider the time frame in which emission occurs.

Time was initially introduced to longwall gas emission models by Airey, 1968, 1979 by considering the effect of stress on effective time constants. This lead to development of the FPPROG software which continues to be employed for the prediction of emission decay rates in abandoned UK mines, Kershaw, 2005A. The time constant values for UK coal seams at various depths and distances from the working section are shown in Figure 3.16 with the consequential rate of emission with time in Figure 3.17.

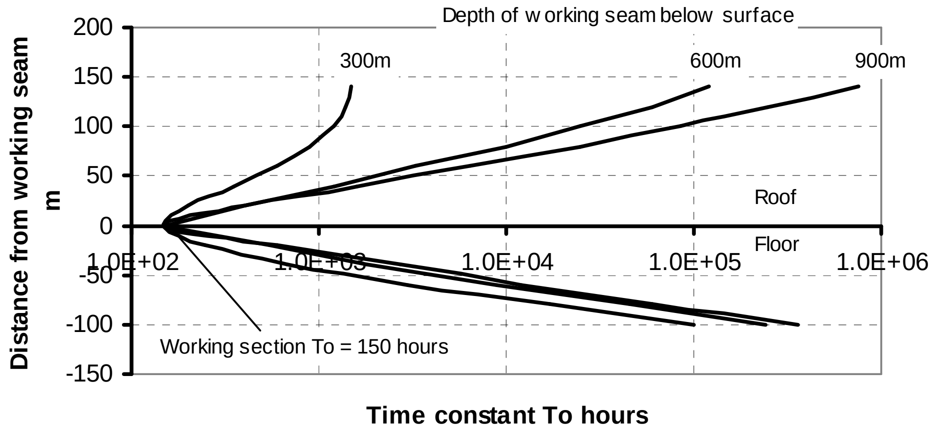


Figure 3.16 Desorption Time Constants (Airey, 1979)

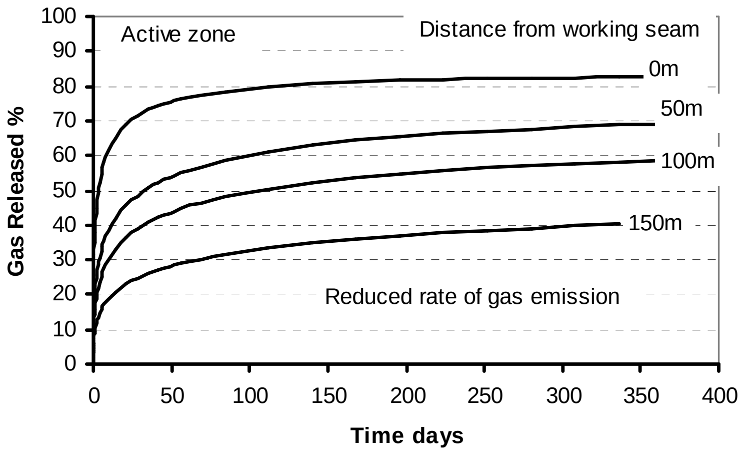
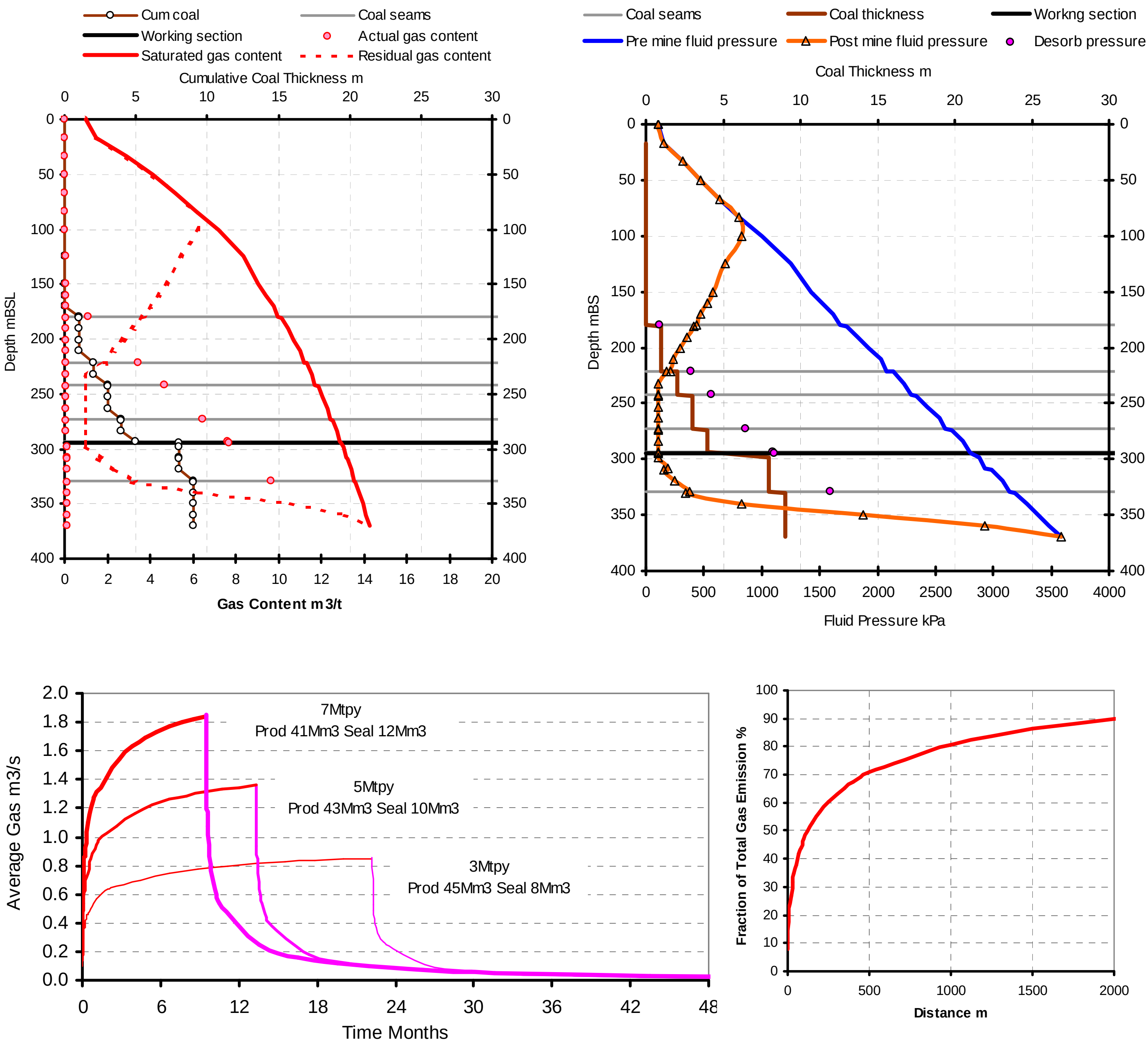


Figure 3.17 Gas Left With Time in Roof Seams

Airey used an observed relationship between time and stress field above and below the working section to determine these curves. In essence, modern finite element programs calculate the same parameter using stress and permeability relationships.

The consequences of these various parameters are best demonstrated by a worked example using typical Australian multi seam conditions and gas content profile for the Goonyella Middle seam, Figure 3.18.



Gas content profile	Moranbah	Block length	5000 m
Interburden porosity	1 %	Block width	300 m
Coal thickness	6 m	Extraction height	3 m

Gas reservoir	51 m3/m2 total
Gas reservoir	75 Mm3 total
Gas reservoir	36 m3/m2 available to residual pressure
Gas reservoir	53 Mm3 available to residual pressure

Figure 3.18 Worked Example Longwall Gas Emission

In this example, about 70% of the total (Q1+Q2+Q3) gas present in the gas reservoir would be emitted based on post mining residual pressures. Of this, of the order 80% would be emitted during the production phase and 20% after production has ceased. More gas would report to sealed areas at faster longwall extraction rates due to the rate of desorption from remote seams leading to a reduced specific gas emission rate during production. It is the production phase gas emission that will contribute most significantly to total mine emissions.

Of the gas emitted during the production phase, 50% would be emitted within 200m of the face line and 70% within 500m. This creates the problem of capturing gas in close proximity to the face line by open circuit goaf holes, particularly if the longwall were retreating up dip in a methane rich environment or down dip in a carbon dioxide rich environment.

In a multi seam environments encountered in many Australian mines, there are multiple gas sources, including the contribution of porous interburden. This makes targeted pre drainage problematic in terms of hole location and cost to achieve sufficient intensity for significant pre drainage effect.

With respect to identifying sources of gas, the fundamental problem currently faced in Australian coal mines is the lack of research, or other sources of data, available to prove that these various models of gas release from roof and floor seams are accurate. It is not adequate to only history match observed gas emission with modelled results due to the large variations in emission rates that occur on a day to day basis. For example, regardless of the sophistication of models employed, a peak factor of 1.5 is often applied to predicted emission rates in order to ensure that ventilation and gas drainage systems are adequate. It is possible that this effect is in part due to gas desorbing from more remote seams, the gas content of porous interburden is higher than anticipated or a combination of both. To improve understanding in this area a comprehensive monitoring of post mining fluid pressure profiles and post mining gas contents of target non working seams is required.

For the purposes of mitigating fugitive coal mine gas emissions, the main risk of proceeding without this information is that incorrect assumptions concerning the source of gas will be made possibly leading to inappropriate, and very expensive, pre drainage trials in seams that may not infact be a significant source of gas. Post mining fluid pressures and gas contents also determine the final degree of gas reservoir emission into sealed areas and abandoned mines.

The operating envelopes for all Australian conditions are provided in Table 3.5 with emission rate calculations based on 5Mtpy.

Table 3.5 Operating Envelope for Australian Mines

Gas Reservoir Size

$GRS = \Sigma\ \rho.t.TDGC\ m^3/m^2$

In this example all coal is at the same density and average desorpable gas content. In practice it is the sum of individual seam thickness, density and gas content.

	Cave Zone Non Working Section Coal Thickness	Gas Reservoir Size (m3/m2) for Average Desorbable Gas Content of Cave Zone Coal								
		1	2	4	6	8	10	12	14	16
	m	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t
Coal	1	1	3	6	9	12	15	17	20	23
Density	2	3	6	12	17	23	29	35	41	46
1.45 t/m3	4	6	12	23	35	46	58	70	81	93
	6	9	17	35	52	70	87	104	122	139
	8	12	23	46	70	93	116	139	162	186
	10	15	29	58	87	116	145	174	203	232
	15	22	44	87	131	174	218	261	305	348
	20	29	58	116	174	232	290	348	406	464

Specific Gas Emission

$SGE = GRS\ .\ frr\ /\ (tw.\ \rho)\ m^3/t$

In this case the average fraction of gas emitted from non working coal is 70% and the working section is 3m thick. In practice the fraction of gas emitted is calculated for each member of the GRS separately and applied to that fraction of the gas reservoir.

	Cave Zone Non Working Section Coal Thickness	Specific Gas Emission (m3/t) for Average Desorbable Gas Content of Cave Zone Coal								
		1	2	4	6	8	10	12	14	16
	m	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t
Working	1	0.2	0.5	0.9	1.4	1.9	2.3	2.8	3.3	3.7
Section	2	0.5	0.9	1.9	2.8	3.7	4.7	5.6	6.5	7.5
3 m	4	0.9	1.9	3.7	5.6	7.5	9.3	11.2	13.1	14.9
	6	1.4	2.8	5.6	8.4	11.2	14.0	16.8	19.6	22.4
Fraction	8	1.9	3.7	7.5	11.2	14.9	18.7	22.4	26.1	29.9
Released	10	2.3	4.7	9.3	14.0	18.7	23.3	28.0	32.7	37.3
70 %	15	3.5	7.0	14.0	21.0	28.0	35.0	42.0	49.0	56.0
	20	4.7	9.3	18.7	28.0	37.3	46.7	56.0	65.3	74.7

Annual Longwall Gas Emission

$AGE = P.SGE\ Mm^3$

Simply production rate Mtpy x gas emission per tonne. Additional to this is gas emission from sealed areas where some fraction of the remaining GRS will be emitted over time.

	Cave Zone Non Working Section Coal Thickness	Annual Gas Emission (Mm3) for Average Desorbable Gas Content of Cave Zone Coal								
		1	2	4	6	8	10	12	14	16
	m	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t	m3/t
Annual	1	1.2	2.3	4.7	7.0	9.3	11.7	14	16	19
Production	2	2.3	4.7	9.3	14	19	23	28	33	37
5 Mt	4	4.7	9.3	19	28	37	47	56	65	75
	6	7.0	14	28	42	56	70	84	98	112
	8	9.3	19	37	56	75	93	112	131	149
	10	11.7	23	47	70	93	117	140	163	187
	15	18	35	70	105	140	175	210	245	280
	20	23	47	93	140	187	233	280	327	373

Average Gas Emission Rate

$GEav = AGE/(prod\ time\ s)\ m^3/s$

This calculation of rate is for the purposes of determining ventilation requirements. The alternative is to use total time in a year or 31,536,000 s.

	Cave Zone Non Working Section Coal Thickness	Average Gas Emission Rate (m3/s) for Average Desorbable Gas Content of Cave Zone Coal								
		1	2	4	6	8	10	12	14	16
	m	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s
Annual	1	0.0	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8
Prod weeks	2	0.1	0.2	0.4	0.6	0.8	1.0	1.2	1.4	1.6
46	4	0.2	0.4	0.8	1.2	1.6	2.0	2.3	2.7	3.1
	6	0.3	0.6	1.2	1.8	2.3	2.9	3.5	4.1	4.7
Prod days	8	0.4	0.8	1.6	2.3	3.1	3.9	4.7	5.5	6.3
per week	10	0.5	1.0	2.0	2.9	3.9	4.9	5.9	6.8	7.8
6	15	0.7	1.5	2.9	4.4	5.9	7.3	8.8	10.3	11.7
	20	1.0	2.0	3.9	5.9	7.8	9.8	11.7	13.7	15.7

Peak Gas Emission Rate

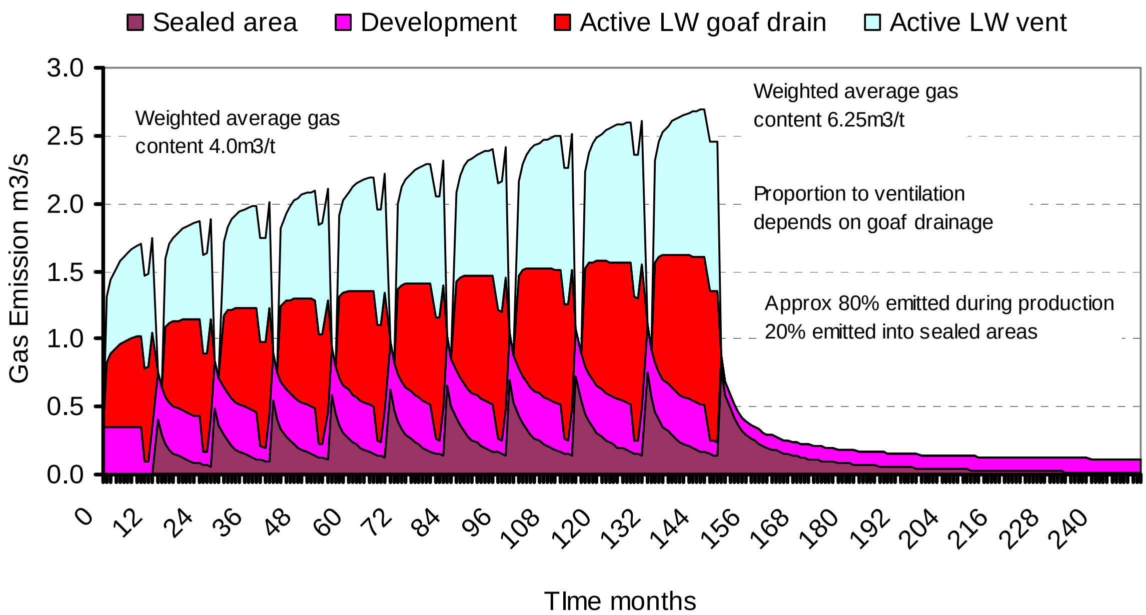
$Gepk = GEav\ .\ PF$

Where PF is the site specific peak factor describing the ratio of peak flow rates to the average due to uneven caving or periodic floor/roof breaks.

	Cave Zone Non Working Section Coal Thickness	Peak Gas Emission Rate (m3/s) for Average Desorbable Gas Content of Cave Zone Coal								
		1	2	4	6	8	10	12	14	16
	m	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s	m3/s
Peak factor	1	0.1	0.1	0.3	0.4	0.6	0.7	0.9	1.0	1.2
1.5	2	0.1	0.3	0.6	0.9	1.2	1.5	1.8	2.1	2.3
	4	0.3	0.6	1.2	1.8	2.3	2.9	3.5	4.1	4.7
	6	0.4	0.9	1.8	2.6	3.5	4.4	5.3	6.2	7.0
	8	0.6	1.2	2.3	3.5	4.7	5.9	7.0	8.2	9.4
	10	0.7	1.5	2.9	4.4	5.9	7.3	8.8	10.3	11.7
	15	1.1	2.2	4.4	6.6	8.8	11.0	13.2	15.4	17.6
	20	1.5	2.9	5.9	8.8	11.7	14.7	17.6	20.5	23.5

3.4 Life of Mine Gas Emission Profiles

Life of mine gas emission profiles are determined by those arising from development and longwall production superimposed on the progressive increase in emission from sealed areas. Fugitive gas emission is this total gas emission (Mm^3) net of that captured by pre and post drainage systems at suitable purity for gas utilisation or destroyed in VAM oxidation units. For example, using the calculation techniques described above, the profile for 10 longwalls at increasing depth is shown in Figure 3.19.



Production rate 5.0Mtpy, longwall duration 12 months (4.5km), face width 300m wide
Gas reservoir $55m^3/m^2$ to $85m^3/m^2$, goaf drainage 50% capture

Figure 3.19 Example Life of Mine Gas Emission Profile – No Pre drainage

Gas emission from abandoned mines and sealed areas have been described for Australian mines (Lunarzewski and Creedy, 2006) and UK mines (Kershaw, 2005A, 2005B), both providing similar characteristic exponential decay curves although with actual values and rates of decay being determined by site specific factors.

An important issue arising from life of mine gas emission analysis carried out by these and other studies is the reconciliation between total gas emitted during the life of a mine and total gas in place prior to commencement of mining. Due to the fact that it is normally only the rate of gas emission into active development and longwall panels that is of concern from an operational point of view, there is limited data to identify to what extent the overall gas reservoir is emitted and the possible influence of gas sources other than that contained in coal seams. For example, analysis of a closed coal mine in the Bowen Basin, suggests that the total volume of gas emitted over the mine's life was two to three times that estimated to be contained in coal seams affected by mining activities.

3.5 Summary of Australian Coal Mine Gas Characteristics

The main pertinent points arising from Australian coal mine gas characteristics are as follows:

- There is a wide range of gas emission rates in the Australian coal industry that must be assessed with site specific data for decision making.

- In addition to VAM, pre drainage and post drainage gas streams, it is important for mines to consider gas contained in production coal which is released on surface.

  The only reliable strategy to reduce VAM and stockpile emissions is to increase pre and post drainage effectiveness, even if this means introducing gas drainage to mines that could otherwise operate in compliance with general body gas concentration limits using ventilation alone. For VAM emissions, thermal or catalytic oxidation technology is available but at considerable capital cost.

- There is clearly a need for mines to improve pre and post mining reservoir characterisation in order to more accurately identify sources of gas and hence possible pre and post drainage solutions. In particular, the degree of depressurisation and hence degassing of non working seams.

- For those mines that do drain gas, a capture efficiency of 30 to 50% is typical with the highest being about 75%. However, it is important to note that this higher value is total drained (including SIS) / (total drained plus total to VAM). If pre drainage is ignored, then the capture efficiency reduces to between 20 and 30% i.e. that amount of gas emitted from active workings that would otherwise report to atmosphere but has been captured by post drainage techniques. Both these values are clearly important for decision making. In effect, it is the life of mine capture efficiency that is most important.

- With respect to pre drainage, the most significant issue in many mines is the large fraction of gas originating from roof and floor seams that will inevitably report to the active goaf environment during longwall extraction. Due to the custom of employing two heading gate roads, concern over spontaneous combustion and Australian management of explosive atmospheres in goaves, there is a limit to post (goaf) drainage capture efficiency. This is already causing some mines to pre drain non working section using SIS pre drainage.

- There is significant room for improvement in the reduction of VAM emissions by thermal or catalytic oxidation in some mines. However, it is to be noted here that VAM emissions are high in mines with high gas emissions and therefore would benefit from increased pre and post drainage. This may provide an overall lower cost option than attempting to destroy methane in very large quantities of air, or in the case of mines that do or can use bleeder shafts, split ventilation systems may be more appropriate.

4. INTERNATIONAL COAL MINE GAS MANAGEMENT

The purpose of this section is to review current world wide gas management practices in order to identify opportunities for improvement in Australian underground coal mines.

4.1 Global Methane Emissions

According to the IPCC report, global average methane concentrations have increased by 150% from 700 to 1,750 parts per billion by volume (ppb) in 1998. The growth in atmospheric methane concentration has slowed down over the past decade according to the global data monitored by National Oceanographic & Atmospheric Administration NOAA with about a 0.5% rise between 2006 and 2007, Figure 4.1.

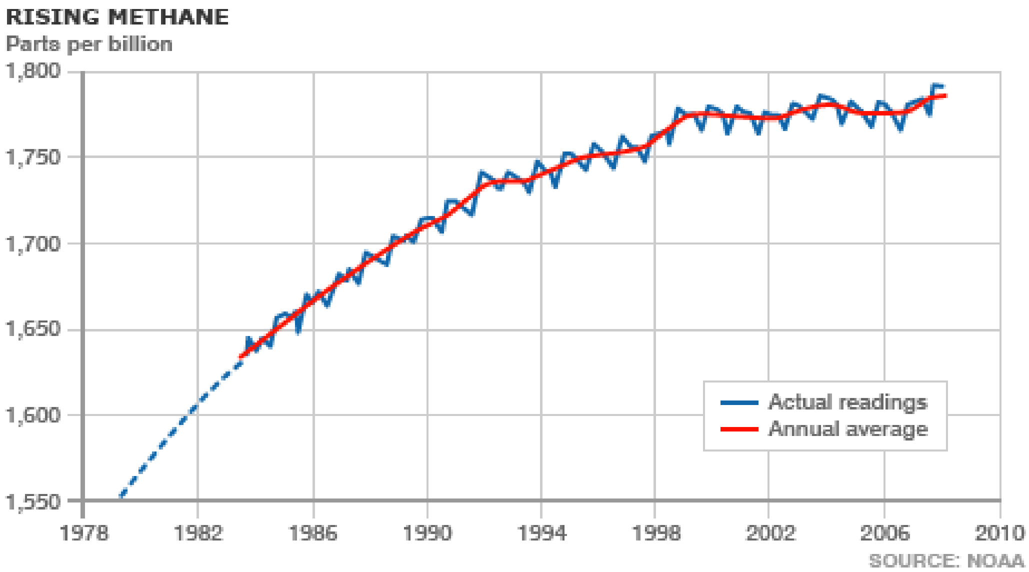


Figure 4.1 Global Atmospheric Methane Concentrations

Methane accounts for 16% of the global GHG emissions with about 60% of this coming from the anthropogenic sources. It is estimated that about 6% of global methane emissions are from coal mines, Figure 4.2.

In 2005 it was estimated that world wide coal mine methane emissions totalled nearly 400 MMT CO_2-e or about 30 billion cubic meters (BCUM). It is further estimated that coal mine methane emissions have increased by 20% from 1990-2000 and are projected to increase further by 25% above 2000 levels by 2020. By 2020 the Worlds CMM emissions are expected to reach 450 MMTCO_2-e (40 BCUM). (Source: Global Anthropogenic Emissions of Non-CO_2 Greenhouse Gases 1990-2020 (EPA Report 430-R-06-003)

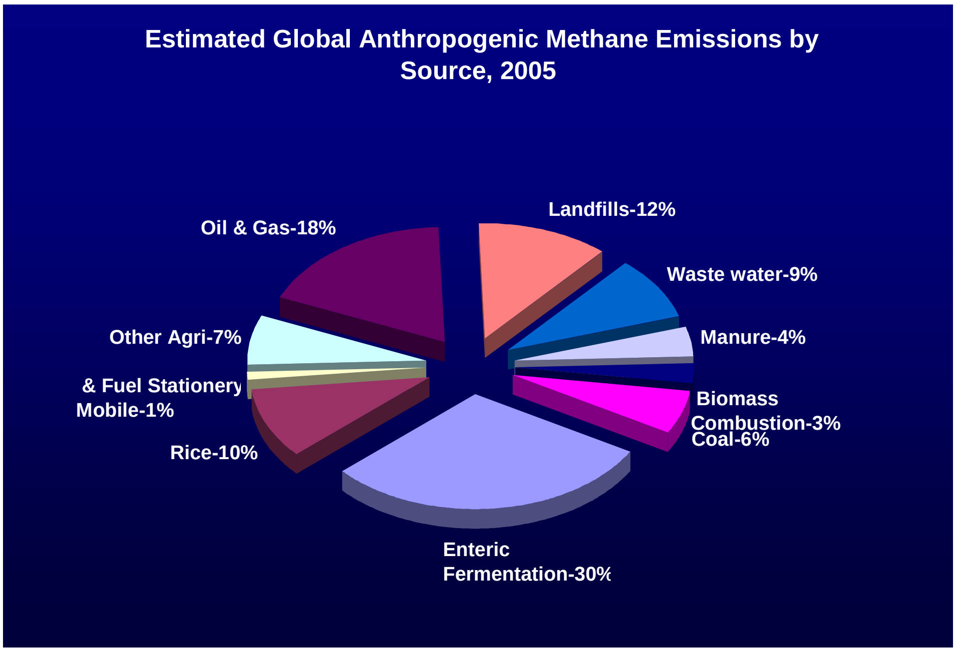


Figure 4.2 Estimated Global Anthropogenic Methane Emissions by Source, 2005

China and the United States, are the world's largest producers of hard coal and are also the leading emitters of CMM. Other countries with significant CMM emissions include Australia, Eastern Europe, Germany, India, Russia and other Eurasian countries (e.g. Kazakhstan and Ukraine), South Africa, and the United Kingdom, Figure 4.3.

Figure 4.3 Estimated Global CMM Emissions, 2005

Global coal mine methane emissions have declined mainly in China between 1990 and 2000 because some of the deeper coal mines have closed and CMM power generation has been introduced. Figure 4.4 depicts the CMM emissions in terms of $MtCO_2$-e world wide since 1990 and the expected trend up to 2020.

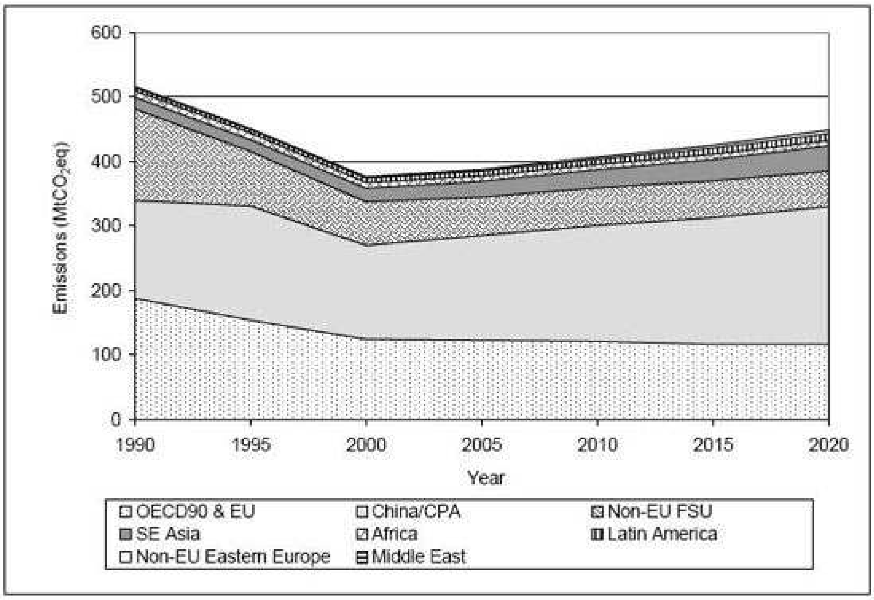


Image courtesy of: http://www.epa.gov

Figure 4.4 Methane Emission from Coal Mines (Source USEPA)

Coal reserves are available in over 70 countries world wide. The current reserve base will serve for 133 years at current production levels. The top five coal producing countries in the world are China, USA, India, Australia and South Africa. The world top ten hard coal producing countries are as shown in Table 4.1 (Source: World Coal, 2007e). Of these China, USA, Ukraine, Russia, Australia, India and Poland contribute 75% of the world CMM emissions.

Table 4.1 International Coal Mine Production

PR China	2549Mt	Russia	241Mt
USA	981Mt	Indonesia	231Mt
India	452Mt	Poland	90Mt
Australia	323Mt	Kazakhstan	83Mt
South Africa	244Mt	Colombia	72Mt

4.2 USA and Canada

Coal accounts for 33.3 percent of energy production of the United States (U.S.) (EIA, 2007a). The U.S. exports only 4.4 percent of its coal production, while its imports equal 2.7 percent of its total domestic production (EIA, 2007b). Table 4.2 quantifies recoverable reserves and recent coal production in the United States.

Table 4.2 U.S. Coal Reserves and Production

Indicator	Anthracite & Bituminous (million tonnes)	Sub-bituminous & Lignite (million tonnes)	Total (million tonnes)	Global Rank (# and %)
Estimated proved reserves (2005)*	112,261	130,461	243,723	1 (28.0%)
Annual coal production(2005**)	519.1	506.8	1025.8	2 (18.66%)

Source: *EIA (2007c); **IEA (2007);

In the U.S., coal mines contribute 10% of all man-made methane emissions (USEPA, 2008). Table 4.3 summarizes coal mining in US by mine type. In 2005, there were 8,000 abandoned underground mines, 440 of which are considered gassy (USEPA, 2004). The US is the second largest country in coal production and coalmine methane emissions. Figure 4.5 depicts the trend of the CMM emissions against that of the underground coal production in US till 2005. The share of the different sources of CMM at different stages is provided in Figure 4.6.

Table 4.3 Recent U.S. Coal Statistics

Type of Mine	Production (million tonnes)	Number of Mines
Underground (active) mines - total	368.61*	612**
Surface (active) mines - total	762.89*	812**

Source: *EIA (2007e); ** EIA (2007f)

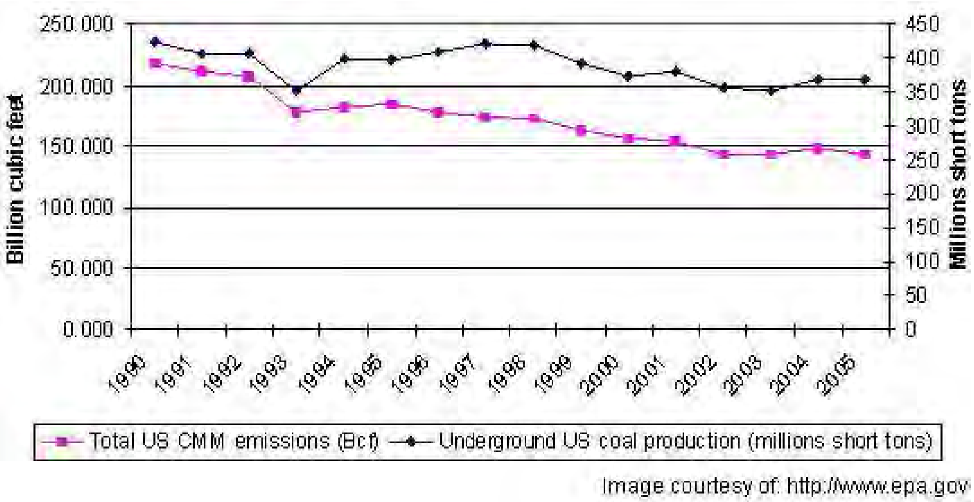


Figure 4.5 Underground Coal production vs CMM Emissions in US

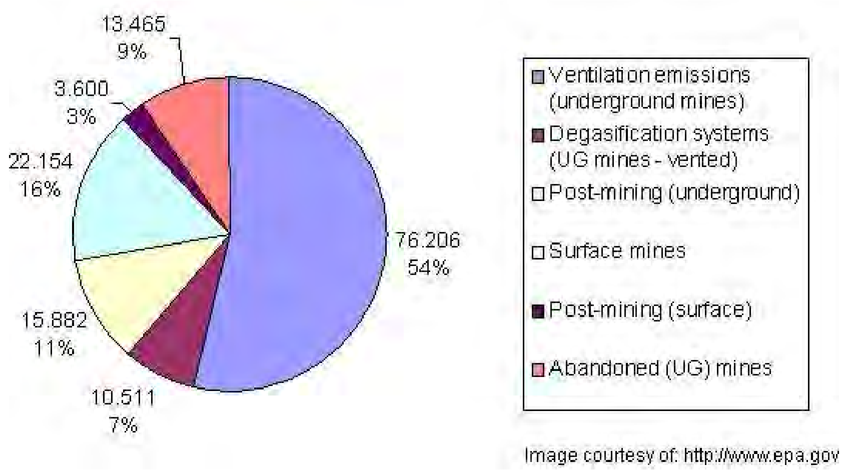


Figure 4.6 2005 US Coal Mine Methane Emissions

In 2006, while only 31% of U.S. coal was produced in underground mines, these mines accounted for over 60% of estimated methane emissions from coal mining (USEPA, 2008). Table 4.4 quantifies methane emissions from the U.S. mining industry from 1990 to 2006. The amount of CMM emissions from US accounts to 38.30 $MMTCO_2$-e at 60% recovery efficiency.

Table 4.4 US CMM emissions (In Million Cu.M)

Emission category	1990	1991	1992	1993	1994	1995	1996	1997
Underground mining	4,363	4,238	4,105	3,388	3,322	3,271	3,204	3,232
Surface Mining	843	797	791	782	824	806	832	851
Post-Mining(UG)	541	511	511	443	488	485	503	523
Post-Mining (Surface)	137	129	129	127	134	131	135	138
Total	5884	5675	5535	4740	4768	4693	4674	4776

Emission category	1998	1999	2000	2001	2002	2003	2004	2005	2006
Underground mining	3145	2913	2756	2675	2487	2520	2665	2466	2512
Surface Mining	884	871	861	922	897	871	905	931	982
Post-Mining(UG)	519	480	468	477	448	450	465	450	439
Post-Mining (Surface)	144	142	140	150	146	141	147	151	160
Total	4692	4407	4225	4224	3978	3982	4183	3997	4092

Source: USEPA (2008a)

4.2.1 Methane Drainage Techniques

Vertical wells (pre-mine wells), gob wells (vertical gob), long horizontal holes & cross measure holes in longwall panels are some of the techniques presently employed in US coal mines for drainage of CMM. But situations at each mine will vary about which method of the above or even a combination of one or all of the above methods will be used for CMM drainage. The selection of the post drainage usage of the CMM gas will also influence which drainage method should be adapted. A summary of the methane drainage techniques used in US is provided in Table 4.5.

Table 4.5 Summary of Drainage Methods

Method	Description	Gas Quality	Drainage Efficiency [a]	Current Use in U.S. Coal Mine zs [b]
Vertical Pre-Mine Wells	Drilled from surface to coal seam months or years in advance of mining.	Produces nearly pure methane.	up to 70%	Used by 6 mines.
Gob Wells	Drilled from surface to a few feet above coal seam just prior to mining.	Produces methane that is sometimes contaminated with mine air.	up to 50%	Used by 23 mines.
Horizontal Boreholes	Drilled from inside the mine to degasify the coal seam shortly prior to mining.	Produces nearly pure methane.	up to 20%	Used by 9 mines.
Longhole Horizontal Boreholes	Drilled from inside the mine to degasify the coal seam shortly prior to mining.	Produces nearly pure methane.	up to 50%	Previously used by at least 2 mines.
Cross-measure Boreholes	Drilled from inside the mine to degasify surrounding rock strata shortly prior to mining.	Produces methane that is sometimes contaminated with mine air.	Up to 20%	Not widely used in the U.S.[c]
Source: USEPA (1993); MSHA (2007);EPA-430-K-04-003,USEPA, SEP 2008. [a] Percent of total methane liberated that is drained. [b] Accurate only at the time of publication of this report; may vary often as mining progresses. [c] Used at West Elk and San Juan mines at one time				

Vertical pre-Mining wells

Vertical pre-mining wells are the optimal method for recovering high quality gas from the coal seam and the surrounding strata several years before mining operations begin. These wells are best drilled into the coal seam several years in advance of mining and may require hydraulic or nitrogen fracturing of the coal seam to activate the flow of methane. These wells typically produce gas of over 90% purity. Recovery from 50 to over 70% of the methane that would otherwise be emitted during mining operations is likely to come out from the vertical degasification wells.

Six of the underground U.S. coal mines currently employing methane drainage systems use vertical pre-mining wells. Figure 4.7, illustrates a vertical pre-mine well.

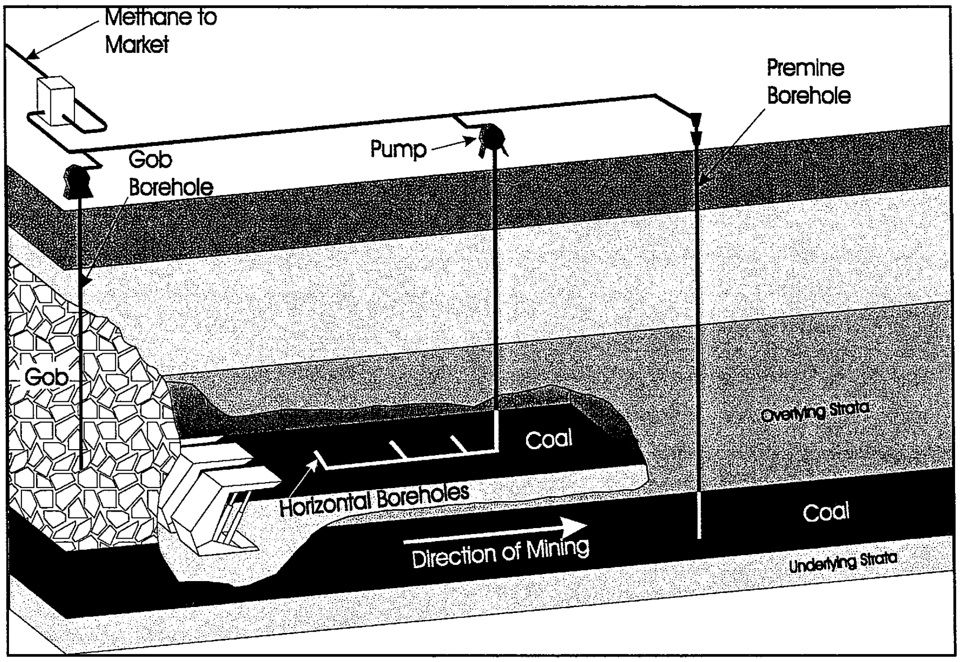


Figure 4.7 Vertical Pre-Mining Gob, and Horizontal Boreholes Gob Wells

(Source: EPA-430-K-04-003, USEPA, SEP 2008)

Gob Wells

Gob wells are drilled from the surface to a point 3-10m above the seam before actual extraction of the coal at regular intervals. During the mining the strata above the seam collapses and creates a fractured zone which is called as “gob “area and is a good source of methane. A pump is generally used above these wells to suck the gas which also prevents methane from entering mine working areas.

During the initial period the gob wells produce good quality and high quantity of methane but over time these may deteriorate due to dilution of mine air into the gob. In some of the mines in US, nearly pure methane production is maintained through careful monitoring & management of the gob wells. In some cases the pumped out gas is upgraded by removing the contaminants.

Depending on the number and spacing of the wells, gob wells can recover an estimated 30% to over 50% of methane emissions associated with coal mining (USEPA, 1990).

Twenty-three underground U.S. coal mines currently employing methane drainage systems use surface gob wells to reduce methane levels in mine working areas, Figure 4.8. Historically, most mines release methane drained from gob wells into the atmosphere.

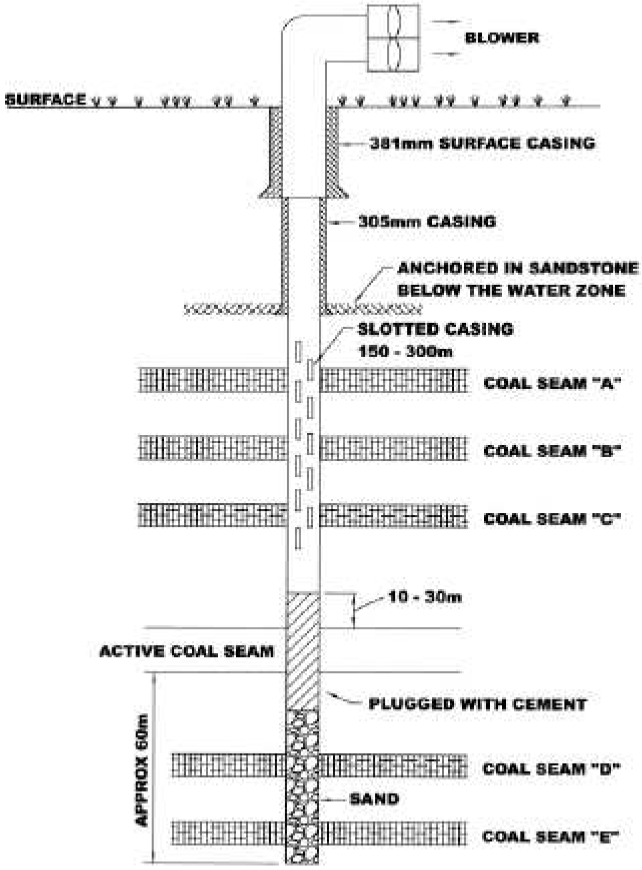


Figure 4.8 Typical Gob Well USA (Thakur, 2008)

Horizontal bore holes

Horizontal boreholes help to drain out methane present in the mine just before the actual mining of the formed longwall panels or the unmined coal blocks of the mine. These holes are drilled from inside the mine and are typically 100 to 250 meters in length. All the bore holes drilled will be networked and will be connected to an in-mine vacuum piping system, which transports the methane out of the mine and to the surface. Horizontal boreholes are generally considered as a temporary relief from methane emissions during mining.

As per the USEPA 1990, estimates the recovery efficiency of this technique is low – approximately 10 to 18% of methane that would otherwise be emitted because methane drainage only occurs from the mined coal seam and not from the surrounding strata.

Approximately nine of the underground U.S. coal mines currently employing methane drainage systems use this technique to reduce the quantity of methane in mine working areas.

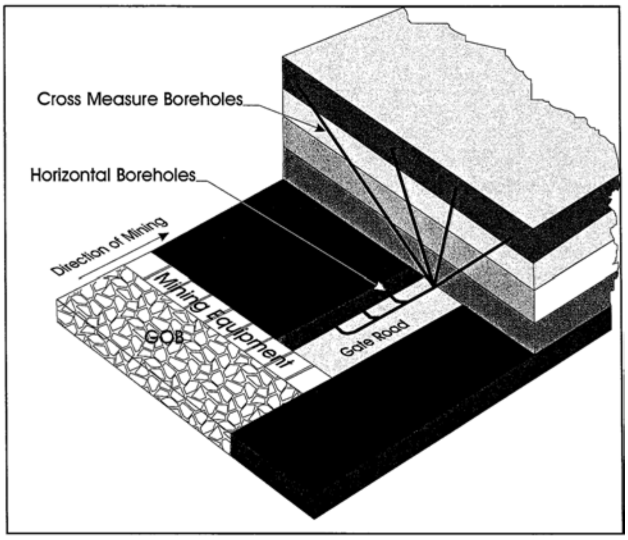


Figure 4.9 Horizontal & Cross Measure bore holes

(Source: EPA-430-K-04-003, USEPA, SEP 2008)

Long hole horizontal bore holes

These are similar to the horizontal boreholes, which are drilled up to a distance greater than 300 metres into the unmined seams using directional drilling techniques. This longhole horizontal boreholes technique is best suited for low permeability and highly gassy seams which require long diffusion time for the gas to drain out of the seam. The holes produce nearly pure methane with a recovery efficiency of about 50% and therefore can be used when high quality gas is desired.

The West Elk mine in Colorado and San Juan South mine in New Mexico have employed longhole horizontal boreholes in their drainage programs.

Cross measure bore holes

Cross-measure boreholes degasify the overlying and underlying rock strata surrounding the target coal seam. These boreholes are drilled inside the mine and they drain methane with a heating value similar to that of gob wells. The method is not so popular in US coal mines as the results are not effective. West Elk mine in Colorado has employed cross-measured boreholes in the past. Figure 4.9 illustrates cross-measure boreholes.

The various strategies employed in underground drilling are described as follows (Brunner, 2005), Figure 4.10.

Modern Directional Drilling Technology

- High Capacity Drills
- High Performance Water Pumps
- High Torque Downhole Motors
- High Penetration Bits
- Precision Survey Tools / Integration with ACAD

Impact on Methane Drainage Applications

- Precision Placement
- Ultra Long Boreholes
- High Capacity Horizontal Gob Boreholes

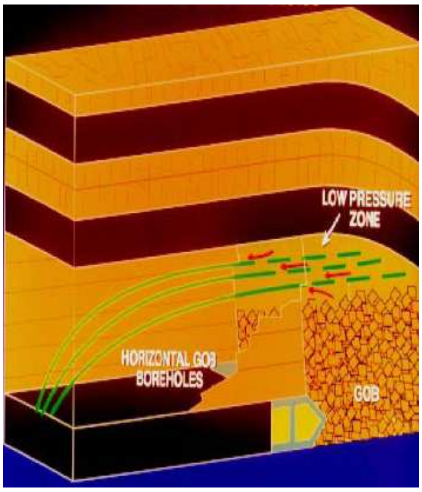


Figure 4.10 Modern US Drainage Strategies – Hydrofracture and Long Gob Holes

4.2.2 Coal Mine Methane Emissions in Canada

Coal is the most abundant fossil fuel in Canada, comprising 66.5 percent of all its fossil fuel reserves. Table 4.6 quantifies recoverable reserves and recent coal production in Canada. Practically all coal mined in Canada (97 percent) is extracted by surface mining methods. Vancouver Island in British Columbia has the only operational underground mine in Canada. Table 4.7 summarizes the country's CMM emissions. (Source: www.Coal.ca, Coal Kit 2003)

Table 4.6 Canada's Coal Reserves and Production

Indicator	Anthracite & Bituminous (million tonnes)	Subbituminous & Lignite (million tonnes)	Total (million tonnes)	Global Rank (# and %)
Estimated proved reserves	3471	3107	6578	14(0.8%)
	28.6	36.7	65.3	11(1.19%)

Source: *EIA (2007); **IEA (2007)

Table 4.7 Canada's CMM Emissions (million cubic meters)

Emission category	1990	1995	2000	2005(projected)
Total emitted (Total liberated)	133	119	70	62

Source: USEPA (2006)

4.3 Europe

4.3.1 United Kingdom

Underground coal mining in the United Kingdom is predominantly from retreat longwall operations. Methane contents range from less than $1m^3/t$ to $15m^3/t$. UK coal seams are generally of low permeability which precludes in-seam pre-drainage as an effective gas control option.

Methane drainage, or firedamp drainage as commonly referred to in the UK, using cross-measures post drainage, is practised in most UK mines. On a retreating coalface, drainage boreholes wherever possible, are drilled behind the face line. In order to achieve this, specific roof support and ventilation arrangements using pre-fabricated curtain back-return construction systems are needed to enable the boreholes to be drilled safely, Figure 4.11. These roof boreholes, depending on the specific target horizons, are typically 30m to 50m in length at angles of 60 to 65 degrees. Standpipe lengths from 13.5m to 15m were being used at different sites. Roof holes were spaced at 4m to 12m depending on the site and floor holes from 30m to 200m. To avoid the access problem and ensure a safe drilling environment, boreholes are also pre-drilled from the return roadway ahead of the face but capture efficiencies are usually less than for holes drilled behind the face due to damage caused by high stresses when the face passes.

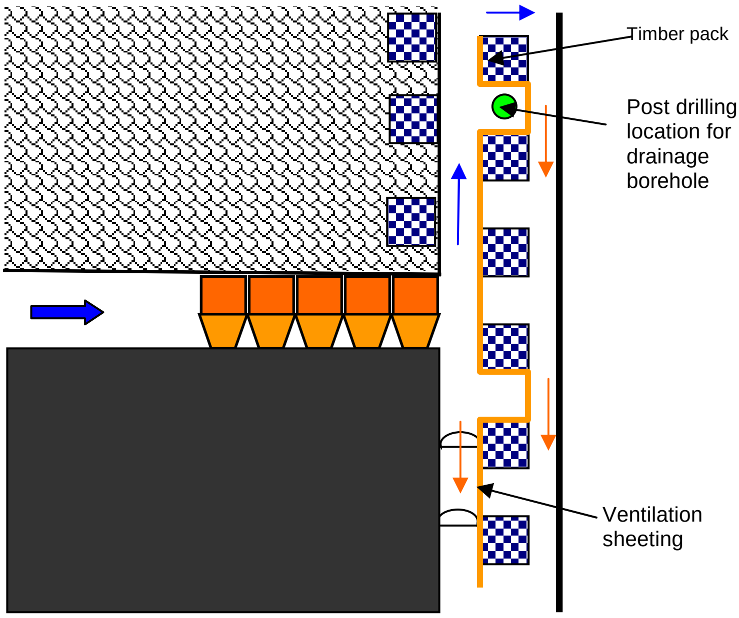


Figure 4.11 Ventilation For Post Drainage Using Prefabricated Curtain

Gas captures from 20% to 45% occasionally attaining 50% were obtained but with poor consistency. Flows of 200 l/s to 300 l/s (pure methane) were typically recorded in the district drainage system. Purity control is achieved using a “leapfrog” system which involves connecting batches of holes to each of two gas collection ranges in turn.

At Tower Colliery the methane was extracted from the workface by means of 30m-deep boreholes, of which the first 15m was encased. These were drilled every 2-3m into the roof of the worked seam at 60º to the workface, as shown in Figure 4.12. The boreholes are connected to an array of Nash pumps at the surface by a series of steel pipes. The drainage procedures produced about 1,000l/s ($13m^3$ of methane released per 1t of coal mined). The gas purity ranged from <35% to 60-70%.

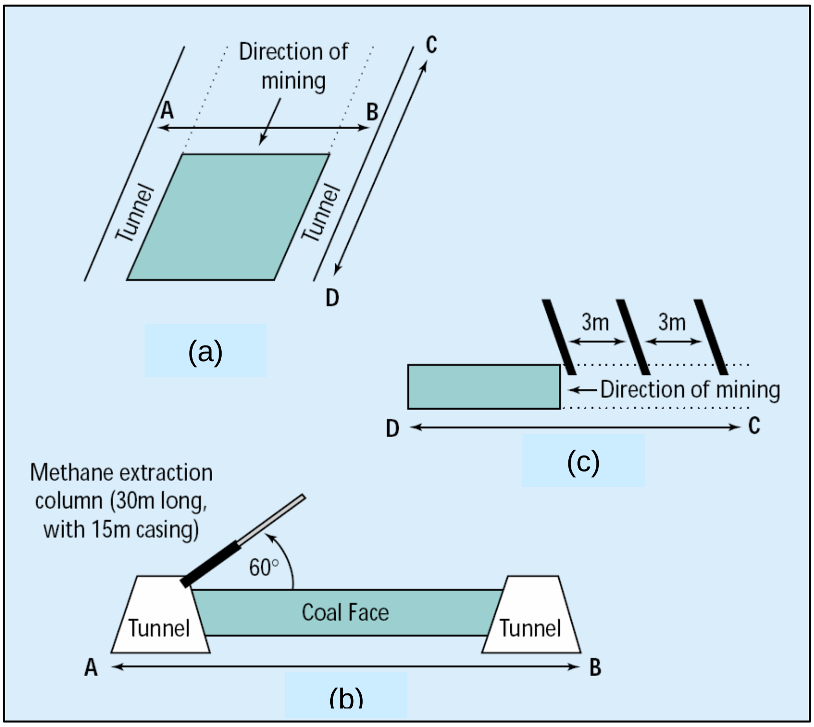


Figure 4.12 Gas Drainage Methods in Tower Colliery, UK

Cross measure drainage in retreat districts is sometimes supplemented by goaf drainage where gas is extracted from behind a seal near the face start line and fed into the firedamp drainage collection pipe work. However, the capture efficiency is generally low due to the low purity of the gas captured.

Another technique is to vent the gas from the rear of the goaf, via a specially driven or pre-existing roadway, into the main air stream where it is diluted to below statutory limits. As a result, firedamp drainage may no longer be necessary, or the required capture efficiency can be reduced.

Creedy (2001) provides a summary of UK gas drainage practice. Key points include:

- On retreat coalfaces, boreholes drilled over the goaf are, on average, more successful than boreholes drilled in advance.
- Boreholes drilled normal to the gate road produce gas for longer periods of time than those angled towards the coalface which tend to become prematurely truncated as a result of differential strains in the disturbed roof strata.
- Gas from uncut roof coal or from seams up to 10 m to 15 m above the worked seam is difficult to capture by firedamp drainage and any that is, will usually be of low quality due to difficulties in achieving air tight seals due to breaks in the roof.
- Failure to consistently drill and complete boreholes near to the coalface may result in a poor and variable firedamp capture performance.
- A specifically designed face-end support system is an essential element of an effective firedamp drainage system.
- Inadequate strata control can lead to fracture of the range and loss of drainage due to ground and pack movement.
- Firedamp drainage performance is improved through good installation, maintenance, regular monitoring and systematic drilling.

Based upon operational observations, previous experience and results reported in the literature, Creedy (2001) identified that the following principal technical factors in ensuring safe and effective firedamp drainage:

- An engineered support system at the face-end to ensure borehole longevity;
- Safe drilling location (stable ground, statutorily acceptable gas concentrations, cool air);
- Good standpipe sealing;
- Standpipes and boreholes of correctly designed and installed length and geometry;
- Optimum borehole spacing;
- Safe access to major gas producing boreholes behind the face;
- Daily flow and purity monitoring on all accessible boreholes;
- Multiple ranges (2 or 3) to assist purity and suction control on batches of boreholes;
- Regular monitoring of outbye district ranges for flow and purity (or remote monitoring relayed to Control Room).

Methane drainage experience in the UK indicates that that the following should be considered when designing and operating a firedamp drainage system:

- Adequate standpipe lengths are likely to be 15m or greater to ensure contact with reasonable purity gas with low air contamination.
- There should be at least 5kPa of suction available inbye.
- Firedamp drainage details should be included with ventilation requirements on mine plans.
- Prefabricated curtains in back-return airways should extend as far back as practicable to ensure the gas “fringe” is kept well back in the waste.
- Firedamp drilling behind the face should, wherever practicable, be undertaken on the waste side to ensure ventilation with relatively fresh and cool face air.
- Standpipes are customarily sealed by allowing drill cuttings to build-up behind a densotape collar. Where consistently low gas purities can be attributed to poor standpipe sealing, alternative sealing methods or the possible need to extend the standpipe length should be examined.
- All orifice plates should be clearly marked showing orientation and orifice diameter Firedamp drainage valves and regulators should be regularly inspected and maintained. Valves should be clearly marked with a number corresponding to a reference on a drainage drainage plan.
- Ranges should be graded to allow water drainage. Water traps should be installed at low points. Manual drains should be checked regularly.
- Staff with responsibilities for firedamp drainage system design should be familiar with the available firedamp prediction techniques and pipeflow calculation methods.
- Ventilation, gas and firedamp drainage data should be presented on a pro forma which clearly shows the current gas control performance and highlights any potential problems which require attention.

4.3.2 Poland

During 2006 Poland produced 94.3 MT coal. Currently, there are 33 active coal mines including 29 gassy coalmine mines in Poland. Twenty of the mines are equipped with drainage systems, and 14 of them are using CMM. Geologically, coalmine methane resource is about 250 billion m^3 with the exploitable gas resource of 95 billion m^3. Total methane released by the mines in Poland was about 870 million m3 in 2006; about 30% of above gas was captured by drainage systems and 70% was emitted into the atmosphere via the ventilation air.

The number of coalmines in Poland has been decreasing over the past two decades. Despite of drop of gassy coal mines number by 48% from 1989 – 2005, drop of absolute gassiness for above period was only 19%. It means that the share of gassy coalmines in Poland has been increasing, and that their gas potential is growing, which will provide significant opportunities and challenges in CMM. (Source: IEA, Workshop report on New trends in coal mine Methane recovery and utilisation April 2008).

4.3.3 Ukraine

Ukraine produced approximately 1 percent of total world coal production in 2005 making it the thirteenth largest producer of coal in the world .Coal production in Ukraine has been declining significantly, falling by almost 50 percent from 116.5 Mt in 1992 to 60.5 Mt in 2004 (IEA, 2005).

Most of the mines in Ukraine are underground producing bituminous coal. There are about 165 active UG mines in 2002 out of which 77% were considered gassy. At some mines, the natural gas content can exceed 35 cubic meters per tonne of dry ash-free coal. (Partnership for Energy and Environmental Reform PEER, Ukraine report 2002).

Ukraine is considered to be the world’s third largest emitter of methane emissions from coal mining activities (USEPA, 2006), even though emissions have been significantly reduced by mine closures and reduced coal production. The Ukraine’s CMM emissions are shown in Table 4.8.

Table 4.8 Ukraine’s CMM Emissions (million cubic meters)

Emission Category	1990	1991	1992	1993	1994	1995	1996	1997
Underground mining - active	3557.51	3276.00	3161.00	2615.28	2414.07	1945.49	1881.85	1839.08
Underground - post-mining	306.41	253.07	251.81	219.87	180.82	160.20	138.05	146.72
Surface mining - active	12.79	9.91	7.97	5.72	3.68	3.16	2.19	1.97
Surface - post-mining	1.83	1.41	1.14	0.82	0.52	0.45	0.31	0.28
Total Emissions	3878.53	3540.40	3421.92	2841.69	2599.10	2109.30	2022.41	1988.05
Additional Recovered and Flared	144.77	137.97	88.22	69.38	94.89	89.03	48.06	56.74

Emission Category	1998	1999	2000	2001	2002	2003	2004	2005	2006
Underground Mining active	1852.92	1819.07	2039.44	1684.68	1911.38	1864.53	1890.53	1837.13	1871.18
Underground –Post-mining	146.72	157.36	156.06	163.33	160.77	156.83	159.02	154.53	157.39
Surface mining - active	1.93	1.63	1.47	1.43	1.23	0.88	0.77	0.43	0.43
Surface - post-mining	0.27	0.23	0.21	0.20	0.18	0.13	0.11	0.06	0.06
Total Emissions	2001.84	1978.29	2197.18	1849.65	2073.55	2022.37	2050.42	1992.14	2029.06
Additional recovered and Flared	83.26	78.93	72.91	134.28	152.35	148.62	150.69	146.43	149.18

Source: UNFCC (2007)

4.3.4 Germany

The substantial coal reserves of Germany are of brown coal (Lignite) category. In 1991 there are only 26 hard coal underground mines which were considerably reduced to 8 in 2006. There are 47 CMM projects in abandoned mines and active mines in Germany. The methane in 33 projects is being used for power generation, while the remaining 14 projects use the methane for combined heat and power. (Source: Statistik (2007) Coal Mining in the Energy Industry in Germany in 2006).

4.4 Russia

Russia is ranked fifth in the global coal production and has a huge repository of coal reserves to the tune of about 157 Billion tonnes only next to US. The details of the coal reserves and the current production levels are shown in Table 4.9.

Table 4.9 Russia's coal reserves and production

Indicator	Anthracite and Bituminus Coal (MT)	Sub-Bituminuos & Lignite(MT)	Total(MT)	Global Rank# (%)
Estimated Proven reserves	49,088	107,923	157,011	2 (18.1%)
Annual Coal Production (2005)	202.9	73.7	276.6 MT	5 (5.03%)

Source: EIA (2007), IEA (2007)

Coal mining in Russia has been privatised since 1996. At present about 77% of domestic coal production comes from independent producers (EIA, 2007a). Table 4.10 presents technology wise production statistics for Russian coal mining.

Table 4.10 Russia's Recent Coal Mining Statistics (2003)

Type of Mine	Production (million tonnes)	Number of Mines
Underground (active) mines – total	93.1	92
Surface (active) mines – total	182.9	119
Total mines	276.0	201

Source: Coal information Bulletin, Tailakov (2005) & Coal (2008)

In Russia, 78 out of 92 underground mines are considered gassy (Tailakov, 2005a) out of which 50 mines are abundant in methane, and 22 of those are using degassing technology (Source Methane to markets-M2M Symposium, 2006). The details of the year wise CMM emissions from Russian Coal mines since 1990 are presented in Table 4.11.

Mines in Russia use traditional ventilation systems as well as bleeder shafts. Traditional drainage is conducted to a very limited extent mainly using gob wells. Collected CMM most likely is being used by the coal mines themselves as well as by manufacturing companies that use large amounts of natural gas. As per the Russian regulations, drained gas must have a minimum methane concentration of 30 percent to ensure that it is not within the explosive range. (Coal Information Bulletin, Tailakov, 2005c).

Table 4.11 Russia's CMM Emissions (million cubic meters)

Emission Category	1990	1991	1992	1993	1994	1995	1996	1997	1998
Underground coal mines	1913.51	1654.72	1657.10	1632.57	1469.25	1508.81	1446.00	1447.10	1390.43
Surface mines	932.76	932.76	860.59	749.05	706.44	658.37	674.07	675.61	686.36
Total	2846.27	2515.30	2406.15	2339.01	2127.62	2182.88	2121.61	2133.46	2131.86

Emission Category	1999	2000	2001	2002	2003	2004	2005
Underground coal mines	1654.85	1449.32	1756.40	2050.90	1781.12	1286.18	1625.28
Surface mines	941.84	944.87	1071.76	1135.13	1197.61	814.29	853.01
Total	2596.70	2394.19	2828.16	3186.03	2978.73	2100.47	2478.29

Source: (Coal Information Bulletin, Tailakov, 2005c).

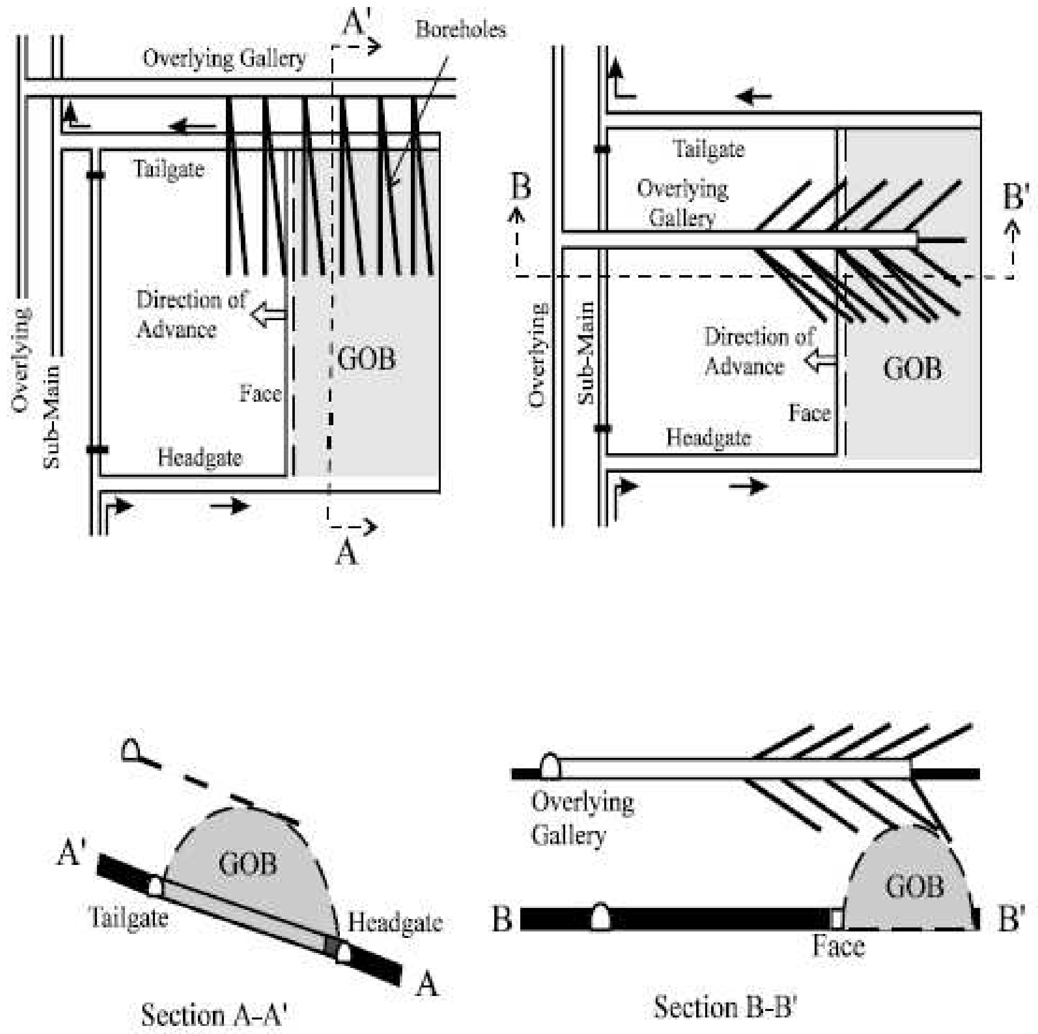


Figure 4.13 Eastern European Goaf Drainage Methods

4.5 Republic of South Africa

The Republic of South Africa (RSA) is the sixth largest Coal producer and the fourth largest exporter of the coal in the world. About 312.5 Million tones (MT) of coal is produced from the RSA mines during 2006. The details of the coal reserves and production are shown in Table 4.12 (EIA, 2007).

Table 4.12 South Africa's Coal Reserves and Production

Indicator	Anthracite & Bituminous (million tonnes)	Sub-bituminous & Lignite (million tonnes)	Total (million tonnes)	Global Rank (# and %)
Estimated Proved Coal Reserves (2005)*	48,750	0	48,750	6 (5.6%)
Annual Coal Production (2005)**	312.5	0	312.5	6 (4.46%)

Sources: *EIA (2007); **IEA (2007)

As per the recent statistics there are currently 69 operating coal mines in South Africa (see Table 4.13). Twenty-five operations use only surface mines, 14 combine surface and underground mining operations, and 30 are solely underground mining operations. About 47 percent of South Africa's coal production is from underground mines and about 63 percent is from surface mines (GCIS, 2007).

Table 4.13 South Africa's Recent Production and Mine Statistics

Type of mine	Production (million tonnes)	Number of mines
Underground (active)	NA	30 (2004)
Opencast / Surface (active)	NA	25 (2004)
Combined OC and Underground	NA	14 (2004)
Total production	239.3 (2003)	69 (2004)

Source:" Aggregate Energy Balances" South Africa Department of Minerals and Energy, 2005

During mid 1990's RSA was considered to be one of the world's top five CMM emitters due to its high coal production but during 2000 it was found that its CMM emissions rank was dropped down to eleventh. A recent study conducted by CSIR measured most of the VAM concentrations from most of the mines in RSA. As mentioned from the excerpts of the above study by Lloyd & Cook (2004) the release of methane from South African coalmines is 106 Million cubic meters with 60 Million cubic meters share from ventilation air methane, 42 Million cubic meters share of the coal after it has left the mine and 4 Million cubic meters share from surface mining operations. The potential end uses for CMM in South Africa include electric power generation, boiler fuel, transportation fuel and petrochemical feed stocks. Methane emissions for South Africa are summarized in Table 4.14.

Table 4.14 South Africa's CMM Emissions (million cubic meters)

Emission Category	1990	1994	1995	2000	2005 (projected)
Underground coal mines – drained emissions	418	423			
Surface mine emission (total)	57	58			
Total emitted (= Total liberated – recovered & used)	471*	-	466*	495*	519*

Source: UNFCCC (2000); *USEPA (2006)

4.6 India

India now ranks third among the top Coal producing countries in the world. Coal is the most important source of energy for electric power generation in India, which consumes more than 70 percent of India's coal production (EIA, 2004a). Table 4.15 provides statistics on India's coal reserves. The coal reserves of India up to the depth of 1200 metres have been estimated by the Geological Survey of India at 257.38 billion tonnes as on 01.04.2007 (Source: Annual report 2007-08, Ministry of coal, GOI).

From the year 2000 the % of coal production in India from underground mines has decreased rapidly from 27% to a current rate of 15%. Though quite a few Deep mines continue to be developed, but more surface mines are also being developed due to the country's vast resource of shallow, low-rank coal deposits. The high-rank coal seams in deeper coalfields represent a significant target for future coal mine methane (CMM) and coal bed methane (CBM) development. Table 4.16 summarizes recent data on coal production.

Table 4.15 India's Coal Reserves

Type of Coal	Proved	Indicated	Inferred	Total
(A) Coking:				
-Prime coking	4614	699		5313
-Medium coking	11853	11061	1880	25334
-Semi-Coking	482	1003	222	1707
Sub-Total Coking	16949	13303	2102	32354
(B) Non-Coking-	81644	106768	35673	224085
(C) Teritiary Coal	467	106	369	942
Total	99060	120177	38144	257381

(Source: Annual report 2007-08, Ministry of coal, GOI)

Table 4.16 Coal Production in India

Year	Production in million tonnes		
Year (April to March)	Coking	Non-Coking	Total
2000-01	30.95	278.68	309.63
2001-02	28.67	293.97	322.64
2002-03	30.49	306.38	336.87
2003-04	29.40	326.32	355.72
2004-05	30.22	347.05	377.27
2005-06	31.51	375.53	407.02
2006-07	-	-	430.83
2007-08 (Dec-07)	-	-	309.51

In India the Coal seams are classified into three categories on the basis of gas emission rates termed as "Degree of gassiness of the Coal seams" .The classification system is furnished in the Table 4.17.

Table 4.17 India's Classification System and Estimates of Mine Gassiness

Class	Rate of Emission (volume of flammable gas / tonne of coal produced)	Number of Mines (in 1994)
Degree I	< 1 m3	288
Degree II	> 1 and < 10 m3	13
Degree III	> 10 m3	24

Source: The Coal Mines regulations, 1957, Issued under Mines Act, GOI.

India's carbon emissions increased by 61 percent between 1990 and 2001, a rate surpassed only by China. In 2004, annual emissions were 1228.54 Mmt CO2e (UNFCCC, 2004). India's carbon emissions are expected to continue to increase throughout the rest of the decade. The rise in India's carbon emissions is in part due to the low energy efficiency of coal-fired power plants in the country. At present there are no mines in India which deals with CMM, which is released into the atmosphere in the form of ventilation Air methane. Table 4.18 summarizes India's CMM emissions.

Table 4.18 India's CMM Emissions (million cubic meters)

Source	1990	1994	1995	2000	2005(Projected)
CMM Emissions (No Utilisation)	*763	957.3	*959	*1,106	*1363

Source: UNFCCC (2004); *USEPA (2006)

Currently one project is under way with the United Nations Development Programme (UNDP), the Global Environmental Facility (GEF), and the Indian Ministry of Coal called "Coalbed Methane Recovery & Commercial Utilisation" seeks to demonstrate the commercial feasibility of utilizing methane gas recovered before, during, and after coal extraction (UNDP, 2007). Recovered CMM will be used for power generation and compressed natural gas fuel for mine vehicles. The Central Mine Planning and Design Institute (CMPDI) is India's lead implementing agency (USEPA, 2004).

There are several projects which on the initial stages to explore the possibility if recovering CBM in two several coal fields Viz., Jharia and Ranigunj Coal fields, etc at greater depths. This exercise is not done as a part of the Coal mining activity, but dealt under separate licensing scheme under the Director General of Hydrocarbons which deals with the Oil & Natural gas resources as well. Estimates of India's CBM potential vary. One source estimates up to 2 trillion m3 of CBM in 56 coal basins covering 64,000 km2. Coal in these basins ranges from high-volatile to low-volatile bituminous with high ash content (10 to 40 percent), and its gas content is between 3-16 m3/tonne. The Directorate General of Hydrocarbons estimates that deposits in 44 major coal and lignite fields in 12 states of India covering an area of 35,330 km2 contain 3.4 trillion m3 of CBM depending on the rank of the coal, depth of burial, and geotectonic settings of the basins as estimated by the CMPDI (Source: Methane to Markets, Indian overview report document).

4.7 China

China is the biggest coal producer in the world and coal output reached 2,716 Mt in 2008. Coal production from underground mines contributes 95% of the total output and over 50% of underground coal mines are classified as gassy and/or outburst prone. Chinese coal mines have a high rate of accidents, among the incidents, gas-related disasters account for over 40%, and 82% of major incidents (over 10 fatalities in a single incident) are caused by gas explosion.

A wide range of gas drainage methods have been developed to suit the varying geological and mining conditions encountered in China. These include both pre and post drainage using surface and underground methods. Advanced underground inseam guided drilling techniques have been demonstrated by foreign contractors and are being applied with some success in China.

4.7.1 Pre-drainage in-seam drilling

This technique is used where coal seams are outburst prone and/or permeability of the coal is sufficient to allow pre-drainage of the coal in advance of mining operations. In-seam gas drainage is used for both mine development and coal production. Some mines, particularly those prone to outbursts require boreholes to be drilled in advance of the development drivage, as shown in Figure 4.14. While this creates a restriction in development rates it allows the roadway to be constructed in a safe manner.

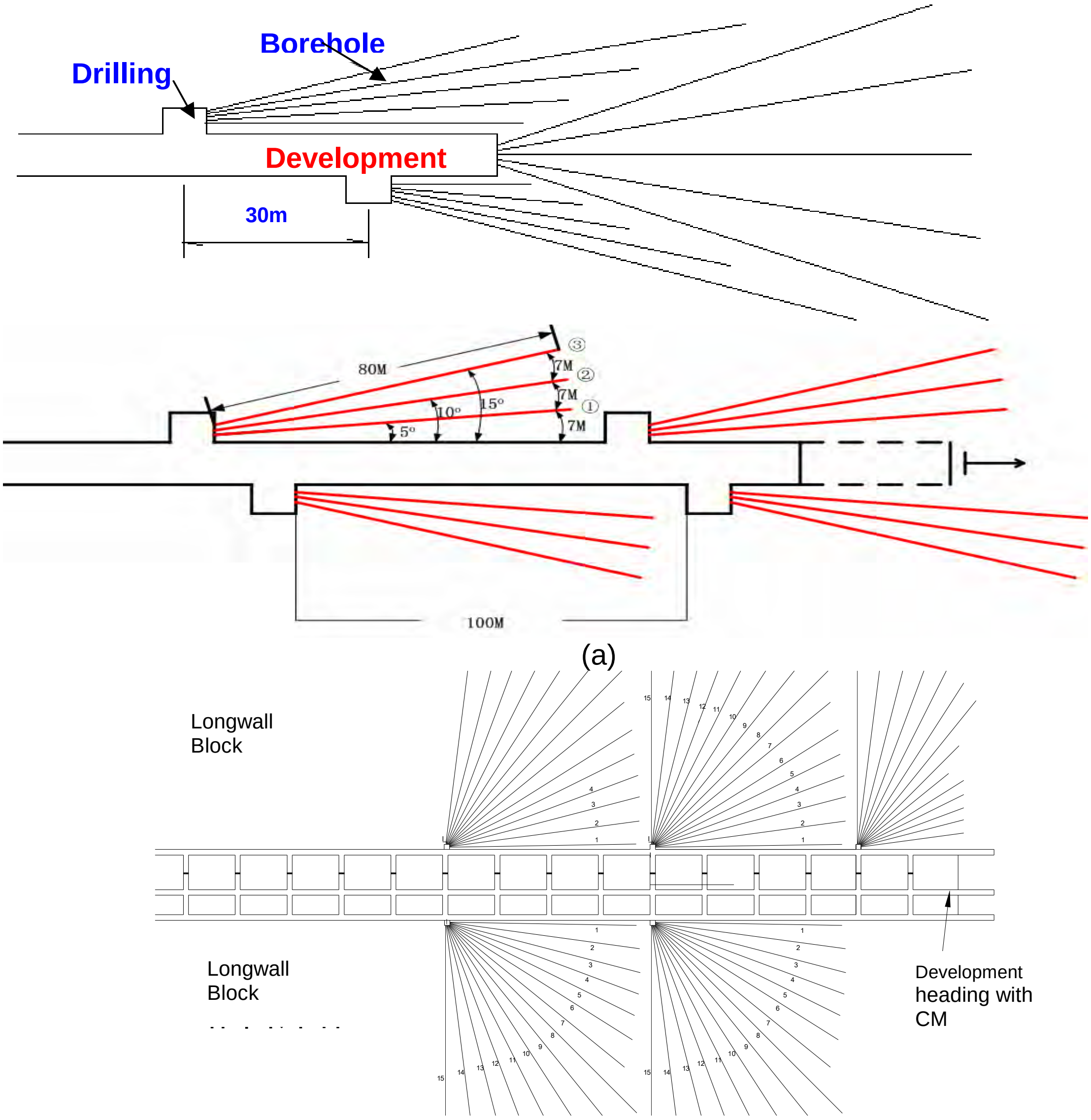


Figure 4.14 Pre-drainage in-seam drilling with development heading

Where the coal seam permeability is sufficient to allow gas to be drained from the unmined coal boreholes can be drilled in-seam before mining is carried out. The time frame between drilling and coal production will vary on the initial gas content, gas content identified for safe mining to be carried out is, number and spacing of boreholes, borehole design and coal permeability. The technique involves drilling boreholes across a coal panel (typically up to 200m as shown in Figure 4.15). At present boreholes are drilled with conventional rotary systems but the application of more advanced guided drilling systems that allow the borehole to be steered in the coal have been tried. Results of such technology suggest mixed success in China due to inappropriate and unsuitable geological setting for the application of technology and characteristics of coal seams in China that are highly variable, e.g., very soft, high gas pressure and rock stress. These characteristics can create difficult drilling conditions for in-seam drilling.

Higher ranked Chinese coal seams are also under saturated making conventional CBM drainage problematic from surface or CMM drainage from underground problematic. Some mines are now drilling in seam holes at 10 to 15m intervals for this reason.

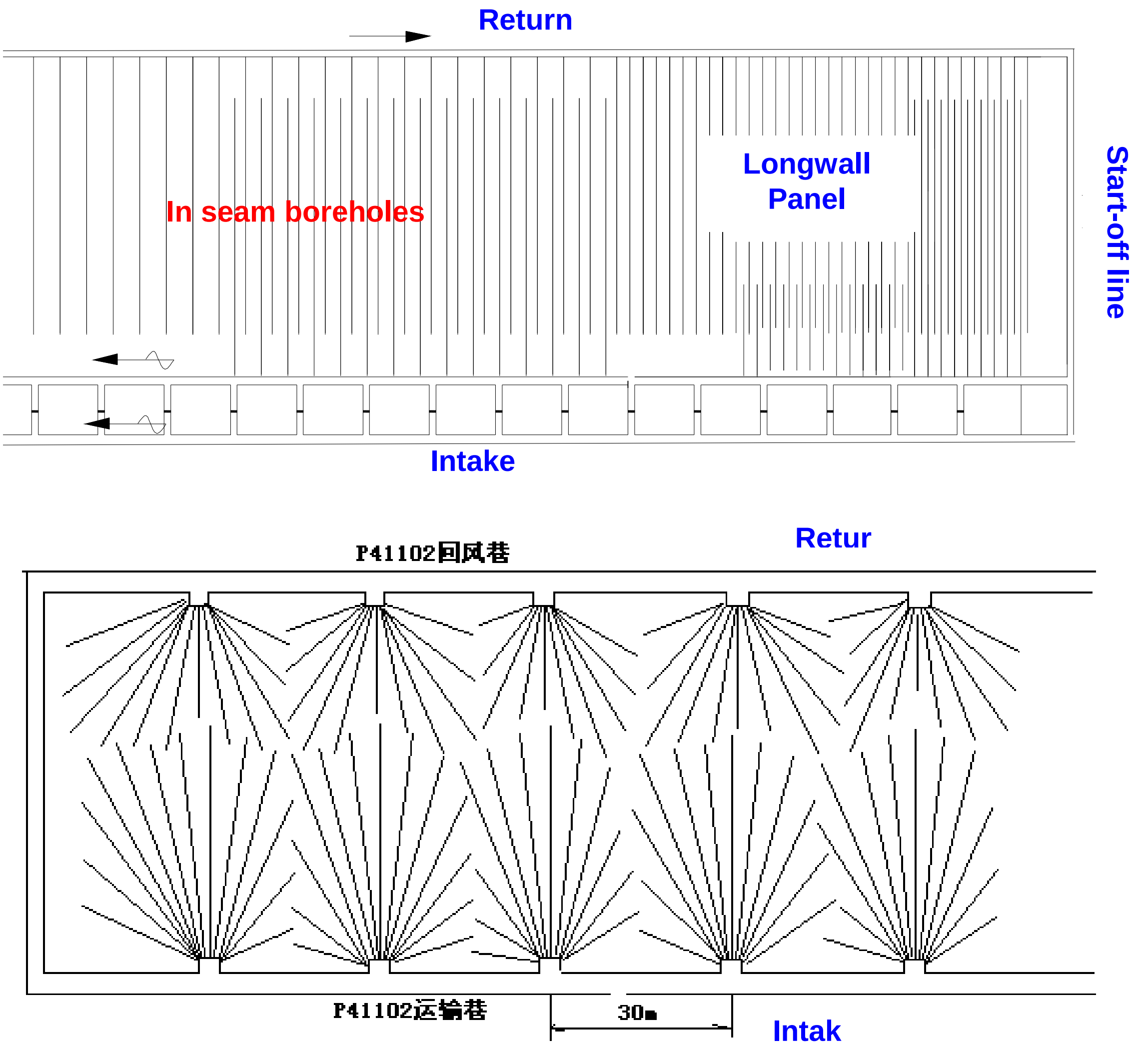


Figure 4.15 Pre-drainage in-seam drilling with longwall panel

4.7.2 Pre-drainage cross measures boreholes from adjacent roadway

The technique involves the construction of a dedicated roadway, usually driven in rock, beneath the coal seam. Boreholes are then drilled upwards to intersect the coal seam. Borehole spacing is estimated from the gas content and likely emissions from the coal seam, as shown in Figure 4.16. The use of a competent roadway from which to drill cross-measures boreholes provides good access for drilling and subsequent monitoring and regulation of boreholes. However, development of a dedicated roadway is likely to be costly.

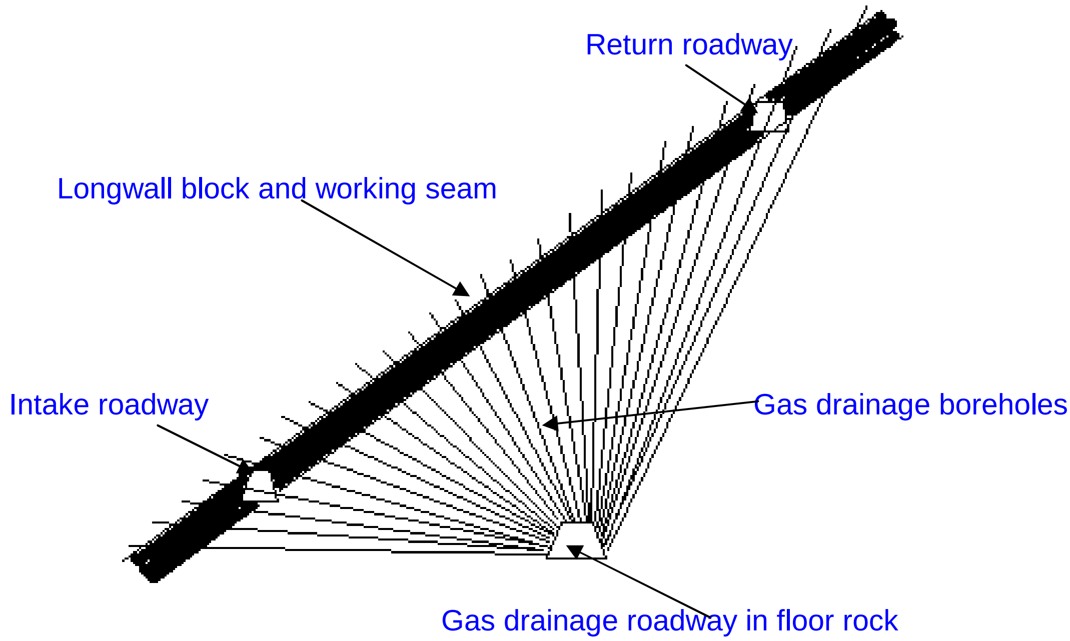


Figure 4.16 Pre-Drainage Cross Measure Boreholes from Adjacent Roadway

4.7.3 Post-drainage using super adjacent (overlying) headings or roadways

This method involves the development of a dedicated a roadway within the distressed envelope of a longwall face. Prior to starting the mining of a longwall panel, a roadway (super adjacent heading) is driven about 20~60m above the working longwall panel, 10~30m from the tailgate and parallel to the access roadways for almost the full length of the panel. As the coal face retreats, the coal bearing strata above the longwall is de-stressed and the released gas is collected in this roadway, as shown in Figure 6. The system has been successfully applied in a number of coal mines. Some mines are also experimenting with the use of long boreholes drilled from the return roadway angled back towards the face line to intersect the goaf area. This method is also widely used to degas adjacent seams before it can be safely extracted (as shown in Figure 4.17).

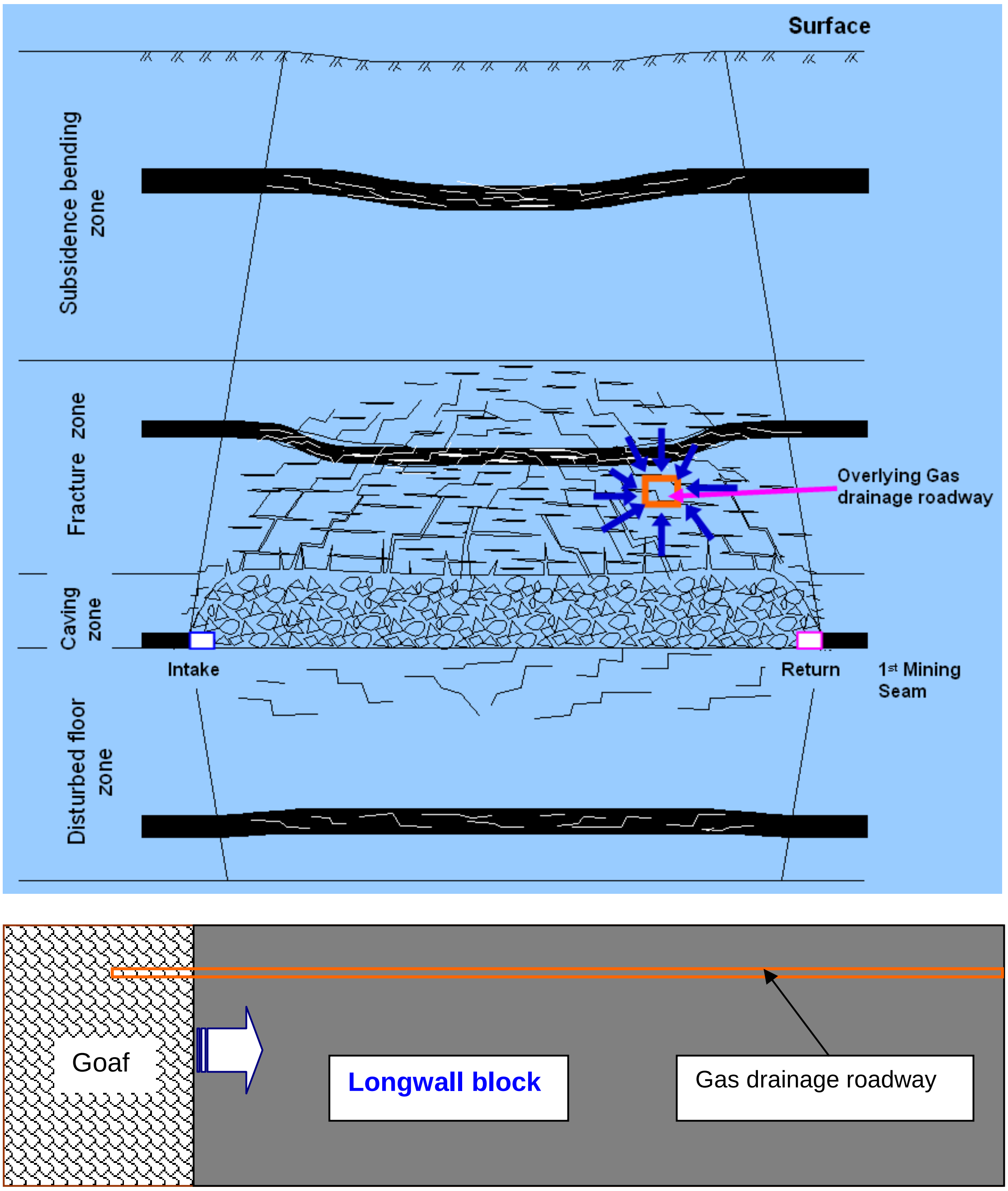


Figure 4.17 Post-drainage using super adjacent (overlying) headings or roadways

Another post drainage option is to drive roadways from the return ventilation roadway parallel to the coal face, as shown in Figure 4.18. Gas collection is achieved by driving short super adjacent roadways at about 100-150m intervals and extend some 20 to 50m from the return roadway at 150 to 200m centres. These roadways can also be replaced by large diameter borehole (230mm) drilled in advance of mining from the return roadway.

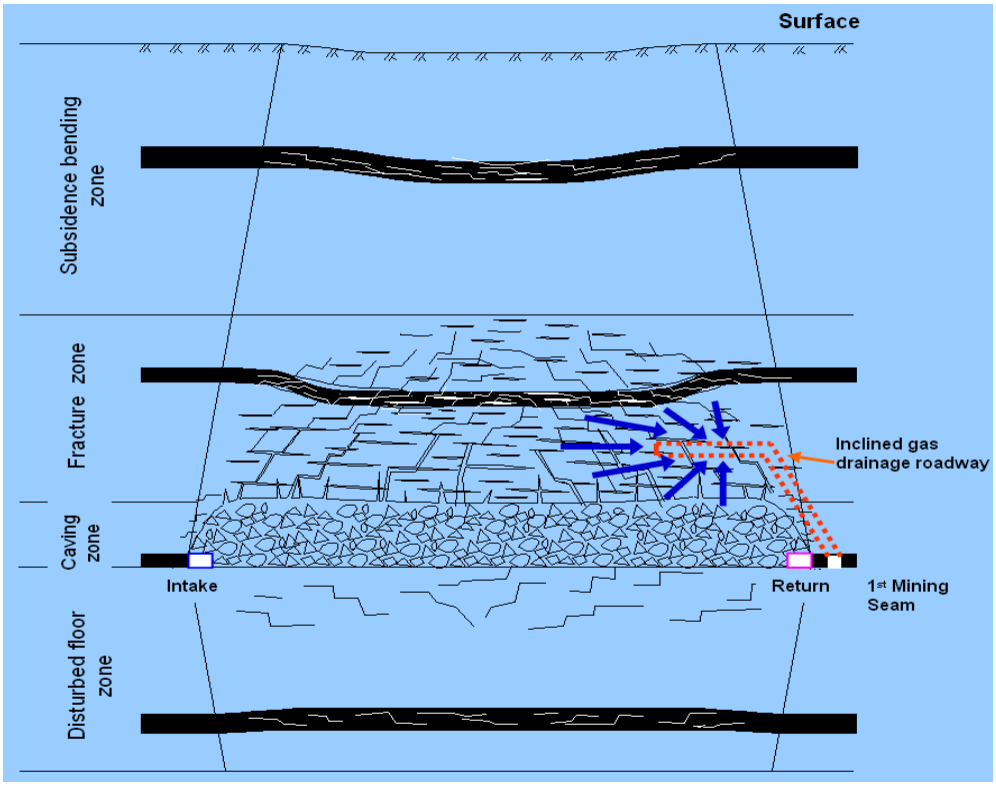


Figure 4.18 Post-drainage Using Super Adjacent Headings From Return Roadway

4.7.4 Post-drainage of goaf gas

A number of goaf drainage techniques have developed in China, and these include:

- boreholes drilled along the panel targeting caved zones above the return corner;
- boreholes drilled towards adjacent goaves via panel pillars;
- gas collected in pipes laid in the goaf area;
- gas collected from behind seals placed in the roadways;
- Surface boreholes drilled to goaf and sealed workings.

Figure 4.19 shows the application of roof boreholes drilled along the panel towards the fractured zone above the return corner of the longwall face (in combination with other techniques). This technique is currently widely used for draining methane from the goaf released from the working seam and adjacent seams. Boreholes are drilled from the return side into fractured roof strata at an upside angle of 10° ~18°, away from the return at an angle of 15°~20°, and 80-140 m in length. Two adjacent drilling insets are spaced 50-80 m. At each inset, 3 to 5 boreholes are drilled, and this leads to borehole overlapping around 40-65 m. Borehole diameter varies from 50 mm to 127 mm, the larger a borehole diameter, the higher gas flow rate from the borehole.

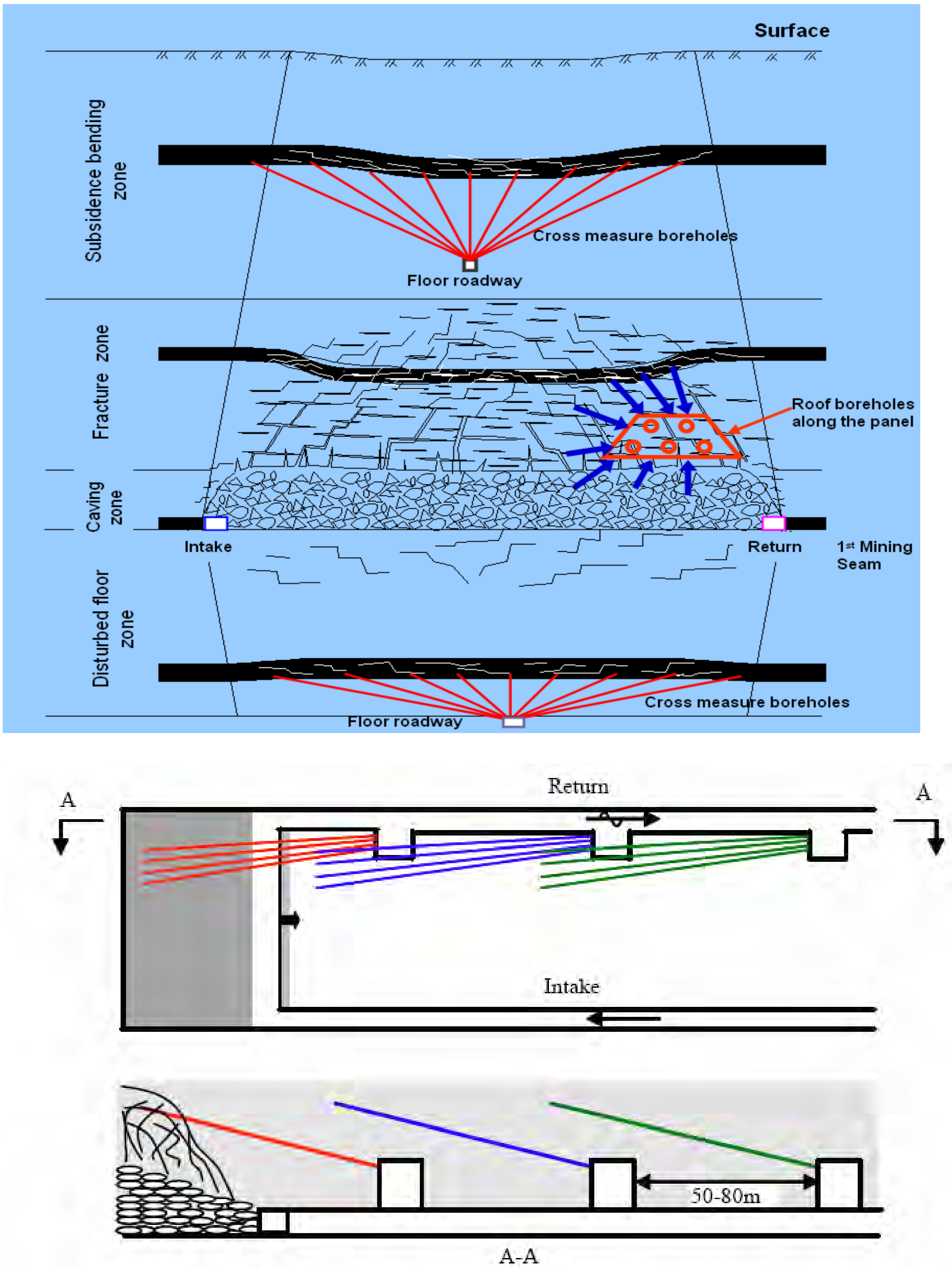


Figure 4.19 Post-drainage of Goaf Gas Using Boreholes Drilled Along the Panel

The use of surface goaf wells was demonstrated as part of the UNDP sponsored CBM/CMM Development Project at Daxing coal mine, Tiefa Coal Mining Group. Three vertical surface wells were drilled to a depth of 532m in advance of the face production. The boreholes were located 50m from the main return roadway at spacing of 150m. These were then connected to surface extraction pumps, Initial results showed that high purity gas could be captured as the face passed. Figure 4.20 shows the structure of goaf well used in Tiefa.

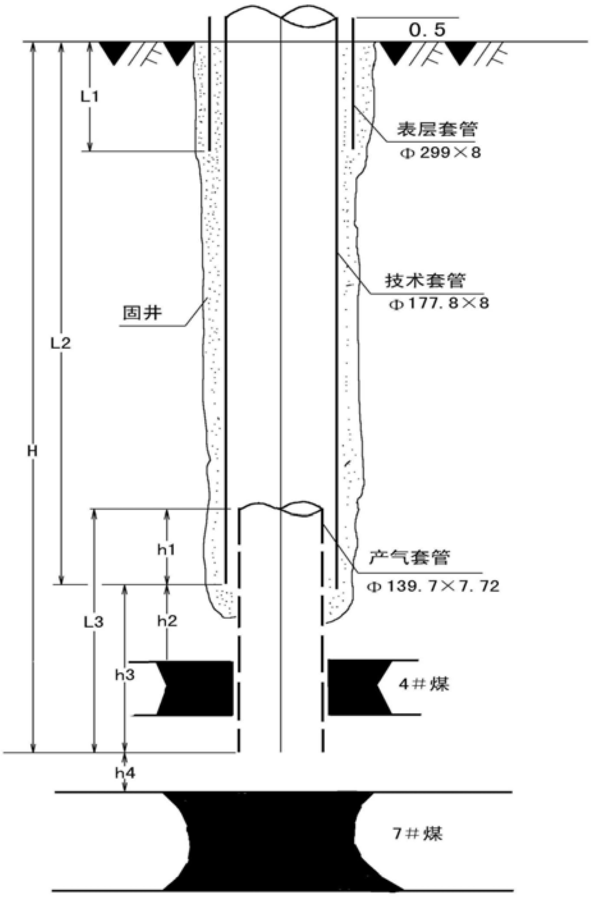


Figure 4.20 Geometry of Surface Goaf Well Used in Tiefa, China

Surface borehole goaf drainage has been trialed in other mines in China with mixed results. Operational experiences in Huainan Coal Mining Group indicate that ff coal seams are more than 600 m below the surface, its application may be also complicated with borehole stability.

4.8 Gas Utilisation Issues

The various uses of CMM employed internationally are summarised in Table 4.19.

Table 4.19 Potential Uses for Gas Produced in CMM Drainage Operations

Btu Quality	Recovery Method(s)	Utilization Options
High-Btu Gas (>950 Btu/scf)	Vertical Wells Horizontal Boreholes	Natural gas pipeline fuel (>97% CH4) Chemical feedstock for ammonia, methanol, and acetic acid production (>89% CH4) Transportation fuel as compressed or liquefied gas.
Medium-Btu Gas (350-950 Btu/scf)*	Gob Wells Cross-measure Boreholes	Spiking with propane or other gases to increase Btu content to pipeline quality Co-firing with coal in utility and industrial boilers Fuel for internal combustion engines (>20% CH4) Enrichment through gas processing Brine water treatment (>50% CH4) Greenhouse heating Blast furnace use (as supplement to natural gas) Production of liquefied gas (>80% CH4) Fuel for thermal dryers in a coal processing plant Fuel for micro-turbines (>35% CH4) Fuel for heating mine facilities Fuel for heating mine intake air Use in fuel cells (>30% CH4)
Ventilation Air	Ventilation Air	Combustion air in power production (<1.0% CH4) Combustion air in internal combustion engines or turbines (<1.0% CH4) Conversion to energy using oxidation technologies (<1.0% CH4)

* In some countries (e.g., China) drained gas may be 350 Btu/scf or lower, but in the U.S. drained gas is well above 350 Btu/scf.
Source: EPA-430-K-04-003,USEPA, SEP 2008.

4.8.1 Drained Gas

When drained gas has a methane concentration of not less than 30%, it can be easily used by conventional gas utilisation technologies including conventional internal combustion gas engines. According to application purposes drainage gas utilisation technologies can be divided into three categories: (1) purification for town gas (pipeline gas), (2) power generation, and (3) chemical feed-stocks. Some of the options are limited in application due to mine site locations, e.g. use of coal mine methane in blast furnaces and in agricultural greenhouses. Coal mine methane can be used as a chemical feedstock for different chemical processes for the production of synthetic fuels and chemicals. Two potential applications in this field are methanol production and carbon black production. These two applications have been demonstrated at mine sites, but have now ceased. In general, drainage gas utilisation technologies in the first two categories are commonly used at coal mines worldwide.

Gas drained from Australian coal mines may contain 30-98 % methane. Enrichment facilities have been successfully upgrading medium-quality gas from natural gas wells to pipeline specifications, but it is generally not economically possible to remove nitrogen, oxygen, carbon dioxide, and water vapour in an integrated system. There are four basic processes that are commonly used for gas purification activities, namely solvent adsorption, pressure swing adsorption, cryogenic separation and membrane separation. In Australia, underground gassy mines are in remote area, and there is almost no pipeline infrastructure. Technically the pipeline is suitable for taking all of the drainage gas regardless the continuality of the gas extracting process. Some mines flare coal mine methane both from pre- and post-drainage gas, resulting in a significant waste of energy. In most countries a minimum of 95 percent methane is required to meet the quality specifications for natural gas pipeline sales.

Generating electricity is an attractive option because most coal mines have significant electricity loads. Electricity is required to run nearly every piece of equipment including mining machines, conveyor belts, coal preparation plants, and ventilation fans. To date, gas engine power generation schemes have been successfully implemented in a number of coal mines world wide, though the conventional gas turbine has also been trialed for drainage gas. Gas supply continuity is an important factor affecting the determination of gas engine plant capacity and the percentage of drainage gas used by gas engines if there is no substantial gas storage facility.

4.8.2 Ventilation Air Methane

In general, the low concentration of methane in mine ventilation air presents a major challenge for utilisation and mitigation. Technologies for VAM mitigation and utilisation may be classified as either thermal oxidation or catalytic oxidation. Table 4.20 summarises VAM mitigation and utilisation technologies in terms of fundamental mechanisms, technical principles and application status.

VAM mitigation/utilisation requires either treatment in its dilute state, or concentration up to levels that can be used in conventional methane fuelled engines. Effective technology for increasing the concentration of methane is not available but is being researched. Most work has focussed on the oxidation of very low concentration methane.

Ancillary uses of VAM generally involve substituting the ventilation air for ambient air in combustion processes. Energy recovery is feasible for the ancillary-use technologies. When the ventilation air is used instead of ambient air as combustion air for conventional pulverised coal fired power station boilers, there are several potential operational issues including possible damage to boilers. This could be due to sudden temperature rise by a quick increase in CH_4 concentration, which could result in slagging and fouling, and slag falls down. Hence, these issues need to be investigated thoroughly before its full-scale implementation can occur. Additionally, the lack of availability of power stations convenient to mines limits the suitability of this technique. Conventional gas engines using mine ventilation air as combustion air have been demonstrated at Appin mine in Australia, but use of these has ceased. This may be due to an economic issue related to gas cleaning because the gas engines require strict cleaning limits.

Table 4.20 Technologies for Ventilation Air Methane

Technology	Oxidation mechanism	Principle	Application status
Ancillary uses			
Combustion air for conventional p.f. power station	Thermal	Combustion in p.f. power station boiler furnace	Mitigation Utilisation – demonstrated in a pilot-scale unit, and being considered for a full-scale demonstration (ceased)
Waste coal/methane combustion in a fluidised bed	Thermal	Combustion inside a fluidised bed and freeboard	Mitigation Utilisation – investigated at a laboratory scale rig
Combustion air for gas turbine	Thermal	Combustion in conventional gas turbine combustor	Mitigation Utilisation – studied
Combustion air for gas engine	Thermal	Combustion in gas engine combustor	Mitigation Utilisation – demonstrated but ceased at Appin Colliery
Principal uses			
Thermal flow reverse reactor (TFRR)	Thermal	Flow reverse reactor with regenerative bed	Mitigation – demonstrated Utilisation – 5.5MW TFFR plant commissioned at West Cliff colliery, commissioning results not publically reported yet
Catalytic flow reverse reactor (CFRR)	Catalytic	Flow reverse reactor with regenerative bed	Mitigation – demonstrated Utilisation – not demonstrated yet at a mine site
Catalytic monolith combustor (CMR)	Catalytic	Monolith reactor with a recuperator	Mitigation – demonstrated Utilisation – not demonstrated yet at a mine site
Catalytic lean burn gas turbine	Catalytic	Gas turbine with a catalytic combustor and a recuperator	Mitigation – combustion demonstrated Utilisation – development of a 25kW demonstration unit
Recuperative gas turbine	Thermal	Gas turbine with a recuperative combustor and a recuperator	Mitigation – demonstrated Utilisation – demonstrated in a pilot-scale unit (?, ceased)
Porous burner	Thermal	oxidation inside porous ceramics	Mitigation – lab scale study Utilisation – not demonstrated at a mine site
Biofilter	Biological	Oxidation inside composts	Mitigation – proposed concept, same as used for landfill gas etc.
Concentrator	N/A, adsorption	Multi-stage fluidised/moving bed using adsorbent, and a desorber	Mitigation Utilisation – the development work stopped in 2004 due to technical issue.
	Centrifugal	Gas centrifuges	Mitigation & utilisation – being proposed as a concept (Bose, 2005).
	adsorption	Novel carbon fibre composites	Investigated at a laboratory scale rig.

Principal uses of VAM involve combustion of the methane in ventilation air as the primary fuel, and some technologies require supplementary fuel when recovering energy to generate power if the primary methane concentration is too low. Both TFRR and CFRR employ the flow-reversal principle to transfer the heat of combustion first to a solid medium then back to incoming air to raise its temperature to the ignition temperature of methane. The two systems differ only with respect to the use of a catalyst. Recently, Biothermica Technologies Inc. has been promoting a VAMOX system for the VAM mitigation. In fact, the VAMOX system is also a type of TFRR, and uses the principle of regenerative thermal oxidation, but its configuration in a U shape is somewhat different to that of the MEGTEC system. This U type configuration of the VAMOX system may result in a higher pressure drop than the MEGTEC system from a point view of fluid dynamics.

CMR technology is a honeycomb-type monolithic reactor which is often used, and is known for its low pressure drop at high mass flows, high surface area, and high mechanical strength. ~5.5MW MEGTEC TFRR pilot-scale demonstration plant at West Cliff Mine in Australia has been commissioned, and the commission results of this are not yet in the public domain. CSIRO has developed and demonstrated a 25kW 1% methane catalytic combustion gas turbine demonstration unit. A 1% methane turbine can use a much greater proportion of ventilation air compared with a 1.6% methane gas turbine, which allows for mitigation and utilisation of much more VAM for typical gassy mines. Thermodynamic analyses indicates that lean-burn catalytic turbines can be operated at lower methane concentrations, perhaps to 0.8%, but it is difficult to generate power efficiently below this concentration.

Concentrators have been applied to several industries to capture volatile organic compounds. A concentrator of this type could be used to enrich methane in mine ventilation air to levels that meet the requirements of the utilisation technologies. The concentrator could also act as a buffer to cope with variations in methane concentration and ventilation air flow rate.

In summary, to address technical feasibility of the abovementioned VAM technologies and other proposed concepts particularly for the VAM principal use, the most important questions are:

- What is the minimum methane concentration required for the VAM mitigation technologies?
- What is the minimum methane concentration required for the VAM mitigation and utilisation (power generation) technologies?

In addition, the VAM mitigation and/or utilisation plant size is quite big as it processes air flow rate of 200-400m^3/s. For example, for a 220-250MW pulverised coal fire power station, its flue gas flow rate is about 300m^3/s. This could be used as an indication of any potential VAM plant size.

4.8.3 Effect of Carbon Dioxide on Combustion Stability

This section provides a brief discussion on the effect of CO_2 contained in drainage gas on combustion stability in gas engines, turbines and flares. Table 4.21 summarises the principles and differences of these two coal mine methane power generation technologies, and flares (sources: US EPA, 1998; DTI, 2004; US EPA, 1999; Walsh and Fletcher, 1998).

Table 4.21 A comparison of mine methane-fired stationary power generation technologies

Technology	Gas engines	Gas turbines	Flares
Mechanism	Combustion	Combustion	Combustion
Operating temperature	1600~ 2000°C	1400~ 1650°C	1000~1200°C (enclosed) 1200-1650°C (open)
Minimum CH_4 requirement	40% (Spark-ignition) 25% GE Jenbacher	30%	20% enclosed flare 25% enclosed flare 30% open flare

Internal combustion gas engines

Generally speaking, when the drainage gas has a methane concentration of not less than 30%, corresponding to a heating value of approximately 10MJ/m^3, the combustion can be stabilised in the gas engines, gas turbines and flares, no matter what other gas compositions such as CO_2, O_2, N_2 are in the drainage gas. When CO_2 is contained in the drainage gas, CO_2 will not react during the combustion, the CO_2 is emitted from the combustion systems into atmosphere if there is no CO_2 capture. Below are combustion calculations for two types of drainage gases to demonstrate that there is no effect of CO2 on combustion temperature. The two types of drainage gases are assumed as:

- 30% CH_4, 70%N_2
- 30% CH_4, 40% CO_2, 30%N_2.

The methane in drainage gas is oxidised with oxygen as follows:

$CH_4 + 2\ (O_2 + 79/21N_2) \rightarrow CO_2 + 2H_2O + 2(79/21)N_2$.

When oxgen/CH_4 ratio is 1.2 in the combustion, summarises the detailed reactants and products in the combustion for the two drainage gases. It seems that the CO_2 has an effect on the combustion temperature, but this can be avoided by adjusting oxygen/CH_4 ratio if needed. It is clear that the CO_2 concentration is higher in the combustion product for drainage gas 2 than that of drainage gas 1 as the CO_2 occurs in the drainage gas.

Parameters		**Drainage gas 1**	**Drainage gas 2**
O_2/CH_4 ratio in the combustion at atmospheric pressure		1.2	1.2
Fuel	CH_4, kg/s	1	1
	CO_2, kg/s	0	3.67
	N_2, kg/s	4.08	1.75
Air	O_2, kg/s	4.8	4.8
	N_2, kg/s	15.8	15.8
Combustion product	Temperature, K	1826	1760
	CO, ppm	109	127
	CO_2, %	6.75	15.78
	H_2O	13.5	13.49
	O_2	2.66	2.68
	N_2	76.87	67.86
	NO	1697	1281
	NO_2	1.35	1.18

Internal combustion engines commonly use medium-quality gas to generate electricity. There are two primary reciprocating engines designs of interest: the spark ignition Otto-cycle engine and the compression ignition Diesel-cycle engine. The essential mechanical components of the Otto-cycle and Diesel-cycle are the same. The primary difference between the Otto and Diesel cycles is the method of igniting the fuel. Spark ignition engines (Otto-cycle) use a spark plug to ignite a pre-mixed air fuel mixture introduced into the cylinder. Compression ignition engines (Diesel-cycle) compress the air introduced into the cylinder to a high pressure, raising temperature to the auto-ignition temperature of the fuel that is injected at the high pressure.

At Appin and Tower mines complex (NSW, Australia), 94 one-megawatt Caterpillar 3516 spark-fired engines are installed, and two sources of methane, gas from in-seam bore holes in advance of mining and gas from gob wells, supply the primary fuel for the project. The fuel gas composition varies from 50-85%, 0-5% CO_2, and up to 50% air. Typically, it is assumed that a minimum methane concentration of 40% is required for spark ignition engine operation. The best gas engine technology, developed by GE Jenbacher, promises to use a CH_4 content of 25%. It is to be noted here that if CO_2 concentration is more than 45%, gas engines won't run even if the CH_4 gas concentration is more than 25%.

Conventional gas turbines

Gas turbines are a complex device based on advanced mechanical design work and have a major application in aircraft propulsion. Other types of gas turbines are also used extensively in the power generation industry for more flexible distributed power systems, base-load power, peak lopping engines, combined heat and power systems, and standby generators for emergency use (Walsh and Fletcher, 1998). The basic principle of gas turbine operation, involves a working gas (air) being compressed and heated by the combustion energy released from injected fuel, the turbine then converts the energy of the working gas into rotating energy through interaction between the gas and the blades.

When considering methane combustion, it is feasible to design a standard combustor with a stable flame when the heating value of the drainage gas is higher than approximately $10MJ/m^3$, which corresponds to about 30% CH_4. Therefore, conventional gas turbines with modified combustors should be able to use post- and pre- drainage gas that contains methane over 30% where there are no problems with supply continuity.

Flares

Two principal types of flare systems are available namely open and enclosed. Open flare systems are relative simple devices burning gas as an open flame with little to control the rate of combustion; hence emissions from such systems can be variable. The enclosed system has been developed to provide more stable combustion conditions with minimum temperatures of 1000^oC and a retention time of at least 0.3 seconds (DTI, 2004), for example, HOFGAS-CFM4c. The open flame with combustion efficiencies of 98 percent is more suitable for a post drainage well application than an enclosed ground-level flare The enclosed flares are used typically at landfills and burn low quality gas more efficiently and emit less NO_x, but have higher capital and operating requirements (US EPA, 1999).

In summary, based on the above discussions on the gas engines, gas turbines and flares, it is fair to select 30% CH_4 as a minimum methane concentration required by these processes to stabilise the combustion.

4.8.4 Economic Parameters for VAM

To date, several pilot-scale VAM mitigation and utilisation demonstration plants including the MEGTEC VAM plant at WestCliff Colliery have been/are being commissioned in Australia, China and USA. Nevertheless, economic factors are an import issue for methane destruction in this manner. It is necessary to use the methane energy and/or by carbon reduction revenue to make VAM abatement economically viable. In general, major economic parameters for a VAM mitigation and utilisation plant are:

- Capital cost
- Operational and maintenance cost
- Benefits by using the methane energy and/or by carbon reduction revenue.

Therefore, it is very important to develop cost-effective technologies for the VAM mitigation and utilisation by reducing the capital, operational and maintenance costs, and by producing the revenue benefits. At present, as two best options of available technologies can be drawn out: 1% methane power generation technology and the VAM mitigation only technology. This also allows a combined plant configuration of 1% methane power units and mitigation units depending on mine site specifications. The 1% methane power units will not only generate power for plant operation, but also push the ventilation air flow through the mitigator units. The 1% methane power units target poor drainage gas, VAM from longwall bleeding fans and VAM with a higher concentration of CH_4 (say $\geq$0.5%), and mitigator units targeting 0.3-0.5% CH_4.

Mitigation at <0.3% CH_4 is most likely to be not an economically viable option. Hence, irrespective of 1% methane power or mitigator units, there is a need to develop such processes with lower capital, operational and maintenance cost through use of simplified processes, reduction of power consumption for the operation and use of cheap materials for the construction.

Based on a series of scientific and technological studies carried out by CSIRO in the last ten years or so, it can be concluded that the development of the cost–effective 1% methane power unit and mitigator unit is a best way to move forward in the respect. In addition, a recent study indicates that VAM adsorption is promising, but it needs a methane desorption study to fully evaluate the whole process of VAM capture.

In general, if the coal industry needs to employ methane abatement, the mitigation plant size and economics are important issues to consider. When the plant size is inevitably large, the coal industry has to have cost effective technologies to lower the capital, operational and maintenance costs, and the carbon trading revenue is required in additional for overall viability. On another hand, the coal industry could enhance drainage gas extraction to reduce the methane emission through ventilation air as drainage gas with high concentration methane being easier to use. The coal industry could improve the mine ventilation air system to make the methane in VAM, or part of it, easier to mitigate.

4.9 Economic Drivers for Improved Gas Management

The contents of this section have been obtained from a number of sources, Creedy, 2009 in particular, and provide an international perspective on economic drivers for coal mine GHG emission mitigation.

4.9.1 Gas Drainage

As with ventilation systems, the cost of pre and post drainage directly increase production cost ($/t), which is why it has been avoided where possible prior to CO_2-e having a financial value. Modern high-production longwalls working a typical seam thickness (circa 3m) can produce 2-4Mtpa in good geological conditions with current Australian targets being more than double this figure. If the coal was worth a net US$40/t then 10% of time gas constrained production would lose some US$8-16million per year. Of course, the fraction of time that a longwall is gas constrained depends on the capacity of gas management systems compared to the volume of gas emitted which is largely dependent on production rates. It is therefore inevitable that, in gassy mines, periods of gas constrained production will increase with production rate unless there is a commensurate increase in the capacity of gas management systems. This is an international problem being addressed by a number of strategies dependent on site specific factors and associated costs, including CO_2-e emission charges.

The cost of gas drainage depends on the magnitude of capacity employed including contingency or operational spares (pumps, flares and engines etc). This could increase the capital cost by 1.5US$/t of production capacity. Gas engines and flares could add up to a further 1.4US$/t to the capital cost and 0.2US$/t to the operating costs, i.e. it is very gas emission rate and therefore production dependent. Drainage operating costs in very gassy mines in difficult geology with directional drilling technology can reach 4-5US$/t. The operational cost range for extracting CMM from underground on a pure methane basis is 0.06 – 0.24 US$/$m^3$.

Equipment, service, labour, surface access and land acquisition costs vary from country to country and even within a country. These cost differences are further influenced by variations in geological and mining conditions which make actual costs, as with gas emission, a site specific issue. Table 4.22 provides an approximate range of costs for methods employed internationally using Chinese and Australian costs as a benchmark. It is understood that these values are broadly consistent with those in the US.

Costs for surface based methods increase with depth of working at which time underground methods will become increasingly attractive, particularly in countries with lower labour costs. Conversely, underground methods require drill sites to be available and are therefore tied to development thus reducing available lead times and requiring increased hole pattern intensity. In gassy mines a combination of methods may be required before high (>>4Mtpy per block) production rates can be safely achieved.

4.9.2 Gas Utilisation

Gas utilization for power generation requires additional investment but generates revenue or reduced power costs to the mine. Issues to consider are the variability of gas supply and quality, opportunity cost and source of financing.

Cost per MW_e of installed capacity for a CMM power plant is around US$1.0-1.2million for international standard high-efficiency IC engines. Capital equipment costs for power generation are typically equivalent to around 0.7-1.4 US$/t of annual coal production capacity or 0.02-0.13 US$/$m^3$ methane utilisation capacity. An estimated operating cost for CMM power generation plant is 0.2 US$/t of production capacity or 0.01 US$/$m^3$ of methane utilisation capacity.

Table 4.22 Cost of Various Gas Drainage Methods

Method	***Technology***	***Major cost items***	***Major cost variables***	***Estimated cost US$/t***
Underground pre drainage	*Guided long boreholes, in-seam along panel length*	*Specialist drillers and equipment*	*Borehole diameter and length*	*0.4-3.2*

	Rotary drilled boreholes across the panel	*Rotary drilling rig and equipment*	*Borehole diameter and length*	*0.6-4.0*
Surface pre drainage	*Vertical well with conventional fracture stimulation*	*Contract drilling, casing and fraccing services. Sealing on abandonment*	*Borehole depth and number of seams to be completed*	*1.2-9.6*
	Surface to in-seam well with multiple laterals	*Contract drilling, casing and specialized, steered down-hole drilling services. Sealing on abandonment*	*Borehole depth and total length of in-seam laterals drilled. Cost can escalate rapidly where drilling difficulties arise*	*1.0 – 8.0*
Underground post drainage	*Cross measure boreholes (from existing roadways)*	*Rotary drilling rig and equipment*	*Borehole diameter and length*	*0.1-1.6*
	Drainage galleries	*Additional roadway development*	*Distance above/below worked seam and roadway dimension*	*0.3-11.2*
	Super-adjacent (or sub adjacent) boreholes	*Specialist drillers and steered down-hole drilling equipment*	*Drilling difficulty for the radius bend*	*0.5-4.0*
Surface post drainage	*Goaf wells*	*Contract drilling and casing. Sealing on abandonment*	*Depth*	*1.4-15.2*

Note: The above are highly generalized, rounded values and take no account of variation of cost of surface methods with depth.

4.9.3 CO2-e Financing Models

Emission reduction credits in the form of Voluntary Emission Reductions (VERs) or Certified Emission Reduction (CERs) provide, where available, an additional financing option to supplement conventional project financing through bank loans or private investment.

Issues to consider with carbon finance include the crediting mechanism, process and transaction costs, time, complexity, local rules and price uncertainty over the future value of emission reduction credits. Investment costs for CMM co-generation plant in terms of emission reduction potential are around 3-5 US$/t$CO_2$ equivalent charge avoided.

A medium gassy mine could earn 0.037 CER/t of coal produced and a very gassy mine (specific emission of 40m^3/t) could earn 0.147 CER/t of coal produced. The calculation assumes that 40% of the gas emitted from the mine is extracted of which 70% is utilised.

CERs involve CDM project preparation, arrangement, annual verification and service costs together with the methane utilisation/destruction equipment and its maintenance. These CDM project costs can amount to 0.8-2.0 US$/CER.

4.9.4 Marginal Cost Benefit

The marginal costs for extracting gas that is utilised, having allowed for emissions from methane combustion, range from 4.0-15.9 US$/t CO_2-e or 4.6-18.3 US$/CER. The lower values are those in projects employing surface drainage techniques or those with lower labour costs, for example in northern China. The upper values represent gas drainage projects in more challenging gas drainage conditions where poor hole stability combined with low permeability require very intense underground hole patterns to be employed.

A gassy mine with an effective gas drainage system could earn an extra 0.55 US$ for each additional tonne of CO_2-e emission reduction depending on marginal costs, CER price and cost. There would be no net financial benefit from CERs from marginally increasing methane extraction and destruction at a very gassy coal mine with a complex drainage system until the CER price exceeds about 19US$ due to the high costs. However, there are also safety and social benefits to consider, including project approval in more environmentally or politically sensitive areas.

4.9.5 Increased Production

The mean capital cost of methane utilisation plant depends on scale and type of process but is of the order 1 US$/t of coal production capacity. In comparison, marginal costs to extend coal production capacity can be around 12 US$/t. Investment in mining instead of gas use would finance a marginal increase in coal production capacity of 1/12 = 0.083. So, for example, the capacity of a gas constrained 4.0Mtpa mine could be increased to 1.083 x 4 = 4.332Mtpa. At a coal price of 30 US$/t the additional annual revenue would be approximately US$10 million.

In some locations, such as gassy Australian mines, it is fundamentally important that the pro rata cost benefit of increasing production beyond historical norms is correctly assessed. For example, one Australian longwall mine achieves production rates of 6 to 7Mtpy in very low gas emission conditions on a two heading gate road basis. This production rate would most likely not be achieved operationally or be economically viable in gassier conditions, as for example is the case in Australian South Coast outburst prone mines.

4.9.6 Environmental Costs

As climate change mitigation and clean energy recovery become an intrinsic part of the coal mining process, mine operators will need to take a more holistic view of these factors. Mine owners may in the future be required, through financial penalties of development consent, to raise gas drainage performance beyond the safety needs of the mines to meet environmental protection targets. It is already the case that constructing an underground coal mine in environmentally sensitive areas results in more stringent consent requirements, in part to appease public opposition, than would have been the case some 5 to 10 years ago.

According to Stern (2006), the generalised social cost of emissions under a business as usual scenario is US$85/t CO_2.e emitted. If efforts are made to stabilise CO_2 in the atmosphere at 450-550ppm then the social cost of carbon will be reduced to US$25-US$30/t CO_2.e. The predictions, although dependent on broad assumptions, do provide some basis for estimating the potential contribution of future mining activity to the social impact of GHG emissions.

Estimations for China (ESMAP 2007) show that the cost of internalising the methane emission impact of coal mining would be around US$12/t. No country has attempted to impose such a cost as yet but the figure provides an indication of the potential future cost to a coal mine which fails to minimise environmental emissions.

4.10 Summary of International Coal Mine Gas Management

The main pertinent issues arising from international coal mine gas management are as follows;

- Improved management of seam gas emissions in and from underground coal mines for safety and GHG mitigation reasons is a worldwide issue of increasing concern.
- There are a wide variety of methods employed but all have the intention of firstly maintaining safe working conditions, and secondly, providing strategies for improved gas capture in order to minimise the VAM load.
- A number of options and management practices used internationally may not be acceptable in Australia, for example, additional non working seam development, stone drivage or operation of gas drainage systems in or close to the explosive range. However, the main issue is to consider how the increased intensity and focus of these systems on capturing goaf and close face gas emissions may be modified in Australian conditions.
- In additional to environmental and safety issues, the economic drivers are also changing significantly as various CO2-e charges and financing models are applied internationally. This will make a number of previously unpalatable options (e.g. pre drainage when below outburst thresholds) economically viable in some mines but also introduce a significant risk to projects when these charges are not fixed in the long term or seam characteristics are not adequately quantified at the start of a project.
- The main lessons to be learnt from international practices is the need to increase gas capture but pre and post drainage techniques so as to minimise the VAM burden, then employ utilisation techniques for methane destruction commensurate with local financing models i.e. cost of capital, cost of labour and cost or sale of power. Clearly, there is not a level playing field internationally in this respect with a greater emphasis on technological solutions required in Australia where labour costs are high and current cost of power is low.

5. GAS DRAINAGE OPTIONS IN AUSTRALIA

Essentially all gas drainage options used internationally are available to the Australian coal industry although application is subject to site specific conditions, various requirements of New South Wales and Queensland coal mine safety regulations together with outcomes of site specific risk assessment.

In this respect, it must be recognised that some techniques practiced internationally with success, such as high capacity direct goaf bleeders (MSHA, 2007) or roof drainage galleries, would not be acceptable in some parts of Australia due to concerns over explosive goaf atmospheres even where the risk of spontaneous combustion is low. It is a valid argument that the fact that underground Australian coal mines are, on a production and employee exposure basis, the safest in the world (Moreby et al, 2008), supports the approach currently taken to coal mine safety through proactive inertisation, although at inevitably higher cost.

Of course, the cost of continuous miner development also limits the application of techniques requiring additional airways or drainage galleries to be installed.

In the context of reducing fugitive emissions from underground coal mines, the purpose of pre and post drainage strategies is simply to reduce the fraction of total gas released from the reservoir reporting to the underground ventilation system or to atmosphere from stockpiles. Historically, only the working seam has been pre drained when necessary in Australian coal mines to control seam gas emission during development and to mitigate the risk of outbursts when threshold limits are exceeded (typically 6 to 9 m^3/t dependent on gas composition).

For economic and operational reasons, custom and practice in low to medium gas emission mines not prone to outbursts, is to use ventilation to avoid the need for underground pre drainage of the working seam, then introduce goaf drainage to manage tailgate concentrations to the point that two heading ventilation circuits can be employed. Higher gas emission mines with a low propensity to spontaneous combustion, such as those in the Bulli seam, employ high capacity bleeder systems but maintain two heading circuits with U or Z configurations. These mines are subject to gas constrained production and, without changes to historical gas management practice, would unlikely be able to exceed production rates of circa 4.5Mtpy.

Three heading gate road circuits provide for increased ventilation capacity and a number of alternative circuit configurations although, unlike mines in the USA, most Australian seams are not suited to place changing development techniques and additional development for three heading gate roads would impose significant economic and operational burdens on mine projects. In any event, increased two heading development rates will be required to support planned for increases in longwall retreat rates (>5Mtpy) making three heading development even less attractive. With consideration to fugitive gas emission, three heading circuits with bleeder systems increase the volumetric capacity of the ventilation system with the design intent of increasing the fraction of seam gas emission reporting as VAM. In these circumstances there may be opportunity to avoid pre drainage of the working seam but higher VAM emission then have to be managed.

5.1 Pre Drainage Options

It is not within the scope of this report to review and analysis each and every aspect of underground pre drainage practice as these are covered by numerous other current and completed ACARP projects together with work from other active research groups. The purpose of this section is to establish when pre drainage may be employed as a means of reducing fugitive gas emission and what considerations need to be made for decision making.

Pre drainage is essential when the gas content of the working seam exceeds outburst thresholds (in some cases frictional ignition thresholds) or development rib emission rates exceed the practicable dilution capacity of the ventilation circuit. However, if a monetary value has been placed on the CO_2-e emissions, it is necessary to consider the value of this gas and where it will be emitted, Figure 5.1.

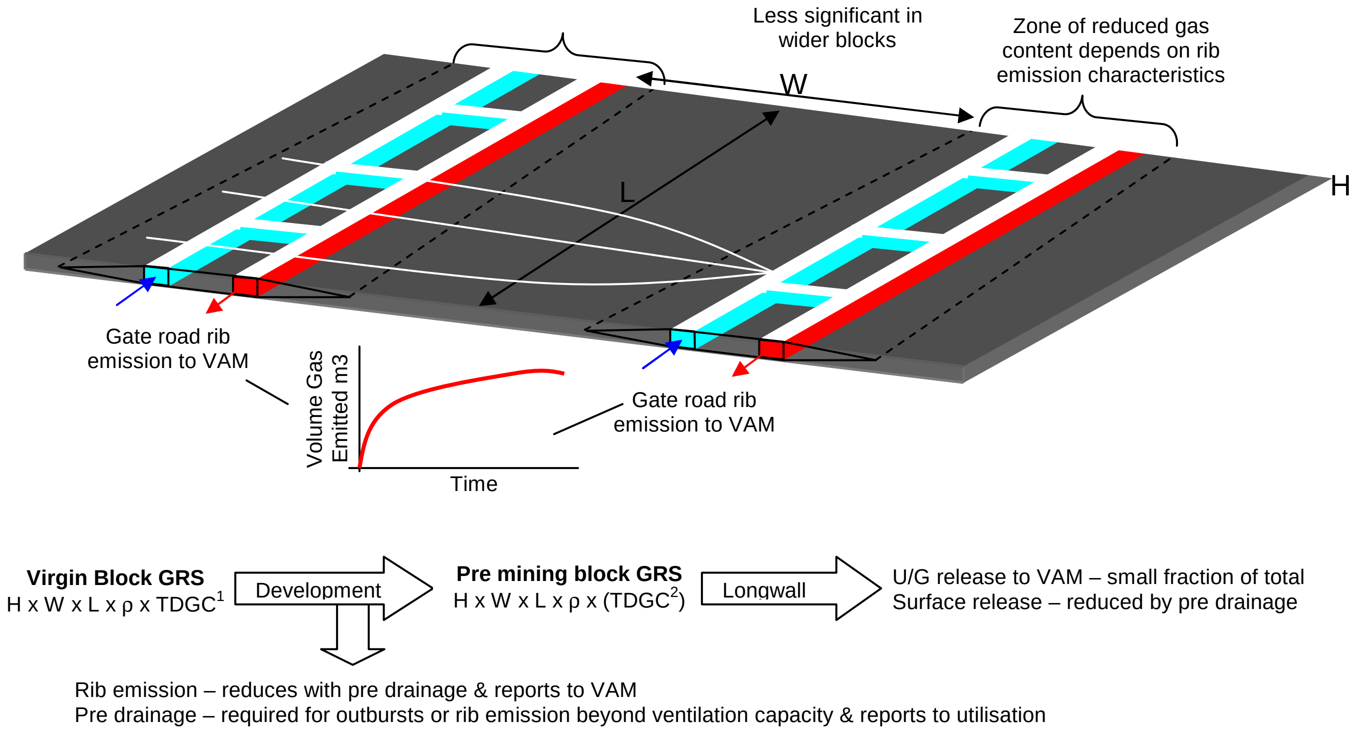


Figure 5.1 Emission Distribution from Working Seam Gas Reservoir

For all mines there will be a total volume of gas in place before mining, some fraction of this will report to ventilation during development, some fraction will report to pre drainage if employed, a small fraction (5 to 15% depending on gas content and residence time underground) of gas in place at time of longwall extraction will also report to ventilation with the balance reporting to surface stock piles in production coal. The effect of gas content of the working seam on these calculations is shown in Table 5.1.

Table 5.1 Volume of Methane and CO_2-e within the Working Seam

Residual 1.2 m3/t

Annual Volume of Methane in Working Seam for a Gas Content (m3/t CH4) of

Production Mtpy	Unit	Ventilation normally adequate				Ventilation maybe adequate			Drained for outbursts in any event			
		1	2	3	4	5	6	7	8	10	12	14
3.0	Mm3 CH4	0	2	5	8	11	14	17	20	26	32	38
	Mt CO2-e	0.00	0.03	0.08	0.12	0.16	0.20	0.24	0.29	0.37	0.45	0.54
4.0	Mm3 CH4	0	3	7	11	15	19	23	27	35	43	51
	Mt CO2-e	0.00	0.04	0.10	0.16	0.21	0.27	0.32	0.38	0.49	0.61	0.72
5.0	Mm3 CH4	0	4	9	14	19	24	29	34	44	54	64
	Mt CO2-e	0.00	0.06	0.13	0.20	0.27	0.34	0.41	0.48	0.62	0.76	0.90
6.0	Mm3 CH4	0	5	11	17	23	29	35	41	53	65	77
	Mt CO2-e	0.00	0.07	0.15	0.24	0.32	0.40	0.49	0.57	0.74	0.91	1.08
7.0	Mm3 CH4	0	6	13	20	27	34	41	48	62	76	90
	Mt CO2-e	0.00	0.08	0.18	0.27	0.37	0.47	0.57	0.67	0.86	1.06	1.26
8.0	Mm3 CH4	0	6	14	22	30	38	46	54	70	86	102
	Mt CO2-e	0.00	0.09	0.20	0.31	0.43	0.54	0.65	0.76	0.99	1.21	1.43
9.0	Mm3 CH4	0	7	16	25	34	43	52	61	79	97	115
	Mt CO2-e	0.00	0.10	0.23	0.35	0.48	0.61	0.73	0.86	1.11	1.36	1.61

CO2-e reduction depends on degree of gas drainage

For example, if coal has an average gas content of 5.0m^3/t CH_4 (residual 1.2m^3/t CH_4) and is mined at a rate of 3.0Mtpy, it would contain 11Mm^3 of gas (net of residual), with annual fugitive emissions of around 0.16 Mt CO_2-e and substantial charge if and when applicable. Without pre drainage, this would report to ventilation as development rib and production coal emission and to the surface stockpile from development and longwall production.

If, for arguments sake, the entire working seam's gas content could be reduced to 2.0m^3/t then the annual fugitive emissions would be reduced to 0.03 Mt CO_2-e, with significantly lower carbon charge and providing a substantial financial incentive to pre drain. In seams with lower permeability, it is likely that the intensity of conventional underground pre drainage necessary to achieve this result would cost considerably more.

If instead, the coal had a gas content of 14m^3/t CH_4 it would have an annual emissions rate of around 0.54 Mt CO_2-e, with substantial annual charge in millions of dollars for the same production rate. Pre draining the working section from 14 to 5m^3/t CH_4 with gas utilisation, would therefore reduce the annual CO_2-e emissions by about 0.54 – 0.18 = 0.36 Mt, but the cost to the mine would then be the value of gas left in the coal at time of production plus the cost of gas drainage. At a production rate of 3.0Mtpy the benefit would be substantial for every m^3/t reduction in pre mining gas content and pro rata for higher production rates.

For mines to make decisions in this respect it is important that an overall volumetric balance is obtained for the life of a longwall block. For example, the distribution of gas emission for various face width from a coal seam containing 5m^3/t and at a permeability of 1 to 2md is shown in Figure 5.2.

Total gas m^3 = gas to vent during development + gas to vent during LW production + gas to pre drainage + gas to stockpile development + LW

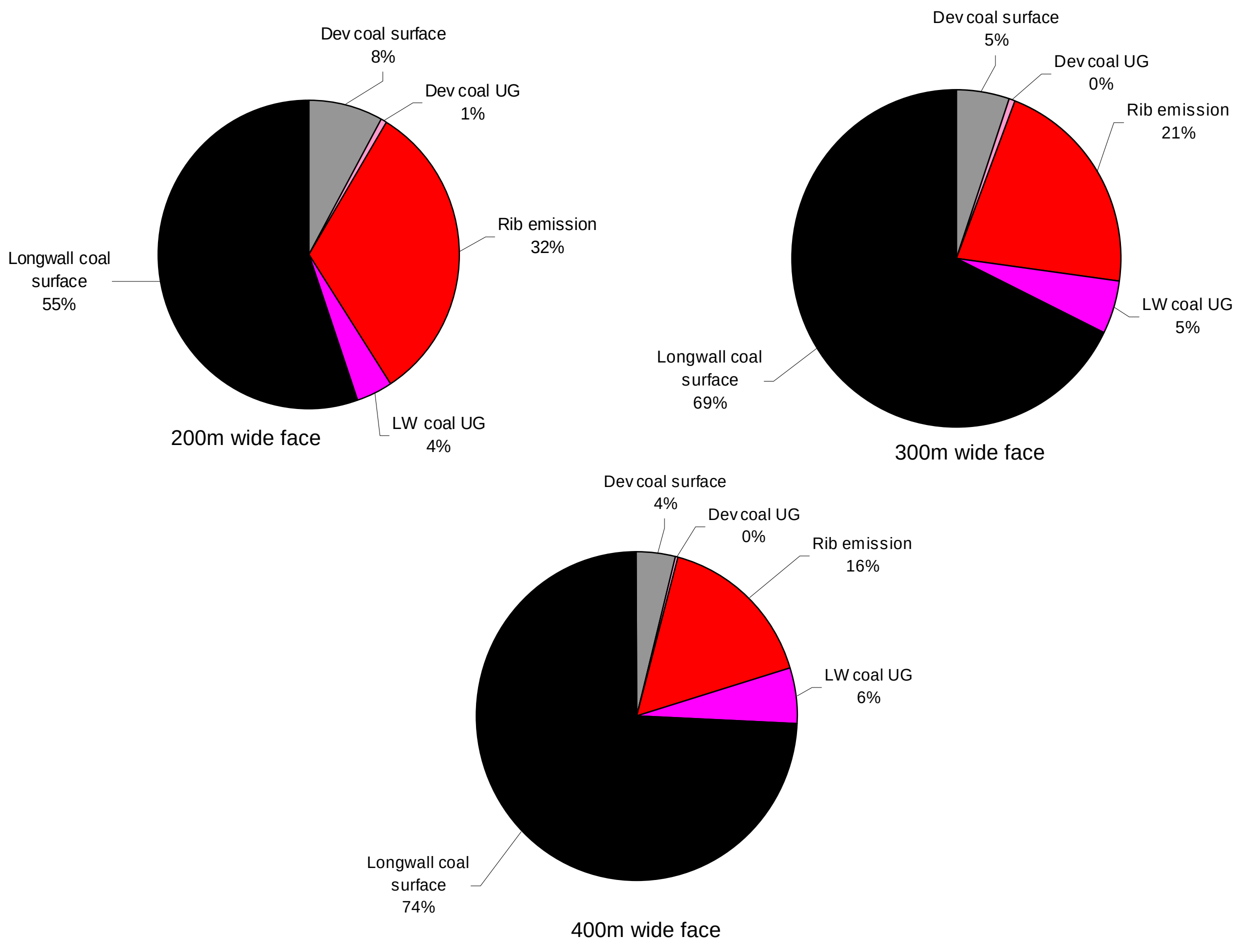


Figure 5.2 Indicative Emission Balance of Working Seam Coal ($5m^3/t$)

The fraction of gas in coal reporting to surface stockpiles will increase in wider faces as the emission from development ribs, as a fraction of the total in place, reduces. This is of course subject to seam permeability orthogonal to gate road direction but provides and indication of the orders of magnitude involved.

As described above, at gas contents well below outburst thresholds, the vast majority of gas in the working seam at time of mining will report to atmosphere from surface stockpiles or during transport on surface. This suggest two opportunities for improvement, firstly improving the efficiency of gas drainage overall but particularly in lower gas contents, and secondly, considering methods of capturing gas released from surface stockpiles.

5.1.1 Pre Drainage Characteristics

Research into pre drainage in Australia using directional drilling techniques has been undertaken for more than 20 years (Lama 1986, Hungerford et al, 1987), with many improvements being made from operational experience, improved modelling and modern drill rig capabilities. The basic function of pre drainage is to reduce the seam pressure adjacent to holes so that gas desorption takes place leading to gas flow from the seam into the hole from where is reticulated to some point of discharge. Without consideration to fugitive gas emissions, points of discharge are normally on surface although some Australian mines discharge directly into the underground ventilation system.

The rate at which pre drainage holes drain gas from a seam depends on a number of factors essentially the same as those determining rib emission rates in development panels. The most significant of these are seam gas content providing desorption pressure to drive the process and seam permeability determining the rate of flow into holes with time and hence the range of pre drainage effect at any point in time.

The two most important factors to consider when designing pre drainage systems are the spacing of holes to promote mutual interference and time allowed for pre drainage to occur. The spacing of holes obviously determines operational costs. Secondary, but also important considerations, are issues such as hole trajectories to avoid water accumulation, suction pressure applied to hole collars, standards of standpipe installation to minimise leakage and effect of hole length on limiting gas flow rates.

An example gas drainage hole flow rate profile is shown in Figure 5.3 for a virgin gas content of $8.5m^3/t$ and with time taken to reduce the gas content of the working seam to a target $4.0m^3/t$ at various hole spacing in Figure 5.4. Similar hole flow rate decline curves have been obtained by parametric studies of longer MRD holes drilled from surface, for example Figure 5.5 for 1300mm long holes (Humphries et al, 2006).

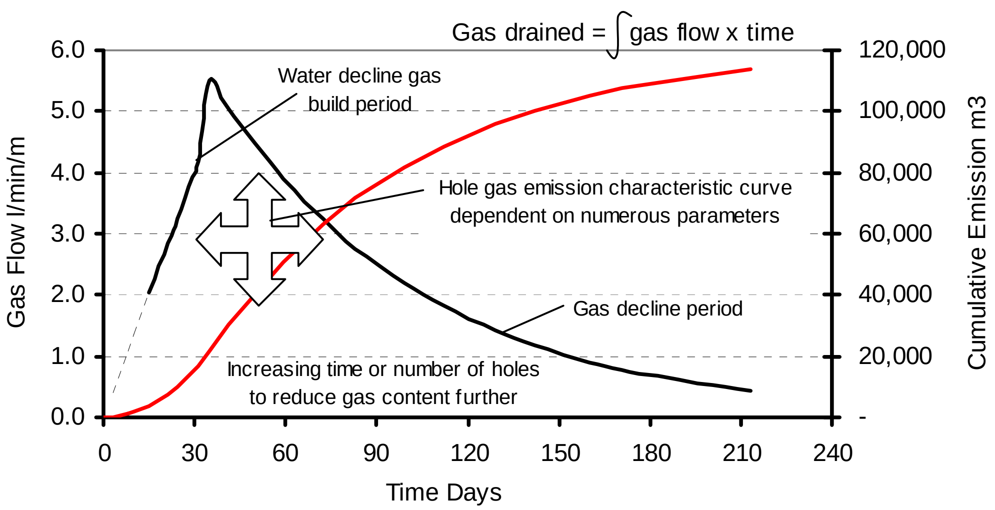


Figure 5.3 Example Hole Flow Curve (German Creek Seam $8.5m^3/t$ 5mD)

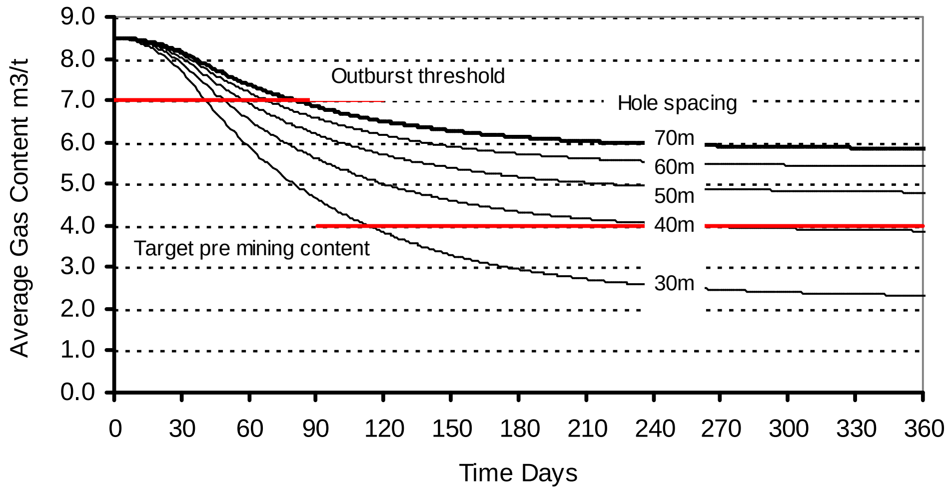


Figure 5.4 Drainage Times with Various Hole Spacing (German Creek Seam 8.5m^3/t 5mD)

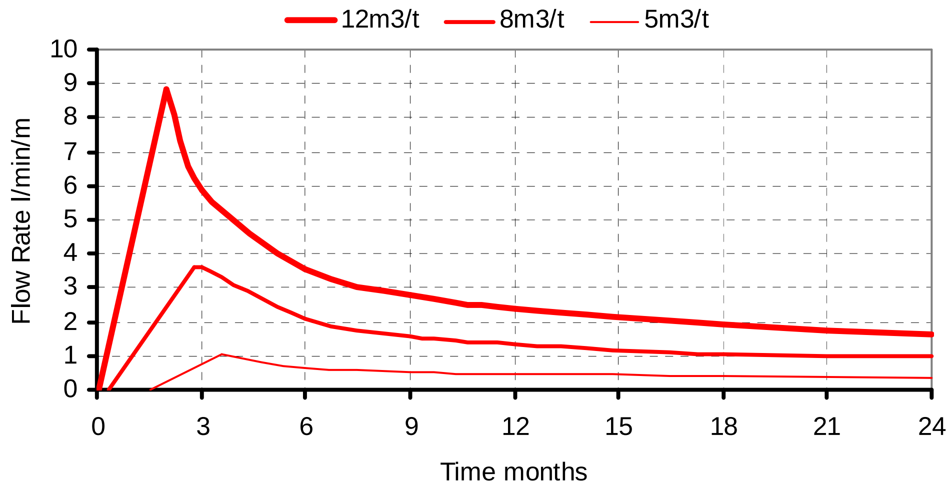


Figure 5.5 MRD Hole Performance (1300m 3mD)

The most significant issues arising from these and other similar generic curves are as follows;

- Peak gas flow rates only occur for 3 to 4 months depending on a number of properties of the gas reservoir, permeability in particular. This means that, in terms of benefit gained by reducing fugitive gas emission, there is a minimum average flow rate required for breakeven in the time available for pre drainage, Table 5.2. This would require unrealistic average flow rates if the hole were active for less than 9 to 12 months.

Table 5.2 Required Hole Flow Rates to Breakeven

		Life of hole months	2	3	6	9	12	24	36	48
Method	Drill $/m	Break Even m3 CH4 (1)	Average l/min/m	Average l/min/m	Average l/min/m	Average l/min/m	Average l/min/m	Average l/min/m	Average l/min/m	Average l/min/m
MRD	1000	2856	32.6	21.7	10.9	7.2	5.4	2.7	1.8	1.4
CIS	70	200	2.3	1.5	0.8	0.5	0.4	0.2	0.1	0.1

- The processes influencing development rib emission rates are essentially the same as those influencing pre drainage hole flow rates. This means that development rib emission rates will most likely be low in seams where low pre drainage rates occur due to low permeability and or low gas content. In this situation, the main function of pre drainage would therefore be to reduce gas reporting to surface stockpiles rather than that reporting to VAM. There may be opportunity to use various pre drainage hole orientations to improve hole flow rates by crossing cleat and or joint directions but this effect would be limited. Most importantly, the option to use VAM instead of pre drainage is not available as the majority of gas would remain in the coal.

- Reduced hole spacing at increased cost is required to achieve lower gas contents at time of mining if holes are drilled from adjacent underground workings. The benefit of MRD holes for pre drainage is the significant lead time available in addition to minimising operational interference underground. However, as described in point 2 above, the cost of both these strategies should be compared to the CO2-e value of gas in production coal rather than only that reporting as VAM.
- In high production rate thick seam longwall mines the time taken to reach lower gas contents (<< 5m3/t) can be similar to or greater than the development time of the panel i.e. closer hole spacing is required to effectively pre drain the block prior to longwall extraction. Pre drainage ahead of development is therefore always necessary.

The issue with pre drainage of the working section is that beyond a certain gas content (typically 4 to $5m^3/t$) there will be little reduction in the fraction of gas reporting to ventilation unless very close hole spacing or long lead times are employed. However the CO_2 –e value of gas reporting to the stockpile may remain high.

Pre drainage therefore becomes more important if return methane concentrations are below those required for VAM oxidation units (about 0.3%CH_4). This provides an important input to the various alternatives for applying VAM units using various ventilation circuit configurations.

With consideration to international practice (for example 1.5m spaced pre drainage holes in China) and these Australian mine issues, an opportunity for improvement may lie in re introducing much cheaper rotary holes to intensify capture potential and hole interference. For example, using widely (30 to 50m) spaced directional hole patterns for outburst control and exploration but in filled with rotary holes at say 5 to 10m spacing.

5.1.2 Non Working Seam Pre Drainage

The purpose of pre draining non working seams is to reduce the amount of gas reporting to ventilation during longwall production where it is emitted in close enough proximity to the face line to reduce the effectiveness of conventional goaf drainage strategies.

In theory, this is no different to pre draining the working seam with MRD or other surface drainage techniques although there may be more need to reduce fluid pressure in roof or floor seams not influenced by working seam development panels. In terms of economic considerations, the pay back or break even calculations provided in Table 5.2 are valid if the gas being drained from non working seams would in fact report to the active or sealed goaves within the working seam at some point in time.

Problems associated with planning and justification of non working seam pre drainage in Australian conditions are generally as follows;

- There is a considerable additional cost in performing multiple completion from single MRD collars leading to a large committed value being at risk in the event of hole failure.
- There is limited experience with multiple completions in Australian coal mining conditions simply because the strategy may be only justified in some mines if a carbon charge is introduced, otherwise mines would employ goaf drainage and a higher ventilation capacity.

- Typically the gas reservoir contained in non working seam strata is located in a number of stratagraphic members, these may be thin, faulted, soft and generally difficult to economically drill or to maintain hole stability over longer periods of time.
- There is very limited data available from existing Australian coal mines to demonstrate the actual nature of gas desorption from non working seams in the floor and roof of working sections. Other than seams in close proximity to the working seam (of the order 80m roof and 30m floor), there would be some degree of uncertainty concerning the actual contribution made. The 10m thick Wongawilli seam some 40 to 50m below Bulli seam workings is an example of this debate i.e. it is a large gas reservoir that may or may not contribute significantly to Bulli seam gas emissions but, it is understood, pre and post mining core samples are not taken for decision making in this respect.

There are two reasons to pre drain non working seams, the first is to reduce longwall gas emission rates to levels that can be managed by combined ventilation and goaf drainage methods, and secondly to reduce fugitive gas emissions. As, by definition, pre drainage of non working gas sources has no effect on the gas content of coal reporting to stockpiles, it may only be required if longwall gas emission rates are too high for available capture (e.g. above workings where surface goaf holes cannot be installed) or, for some reason, VAM oxidation strategies cannot be employed.

5.1.3 Reservoir Stimulation Options

Reservoir stimulation describes techniques employed to increase the rate and or total volume of gas drained from coal seams by changing one or more properties of the gas reservoir.

Hyrofracture is a method by which the permeability of coal seams, and adjacent strata, is increased by inducing fractures using high pressure water or other suitable fluid and has been considered for some time in Australia (Meaney, 1997). To maintain gas flow, the fracture may also be “propped” open by injecting a pumpable particulate solid such as sand with the purpose of promoting then maintaining higher matrix permeability.

Propped hydrofracture has been used in Australia (Jeffrey et al, 2005, Figure 5.6) and further trials are underway in underground mines (Mills et al, 2006) with applications possible using surface MRD holes or underground cross measure holes.

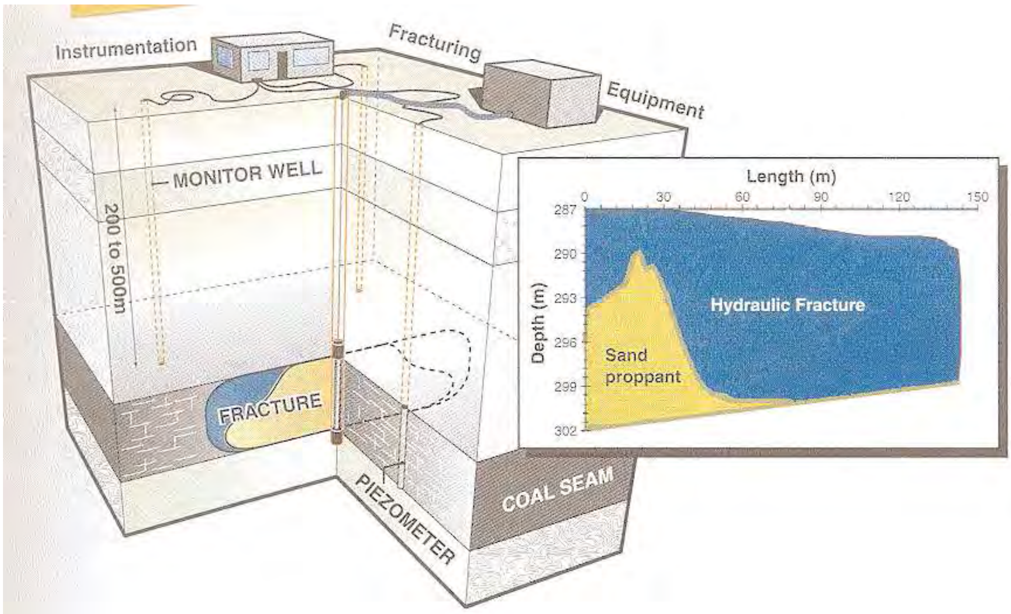
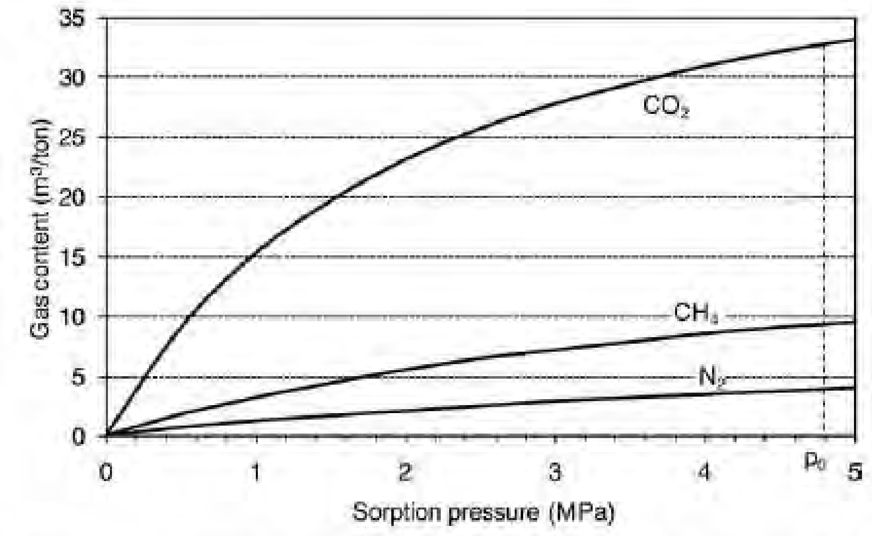


Figure 5.6 Coal Seam Hydrofracture

The application of hydrofracture in Australian coal mines would;

- Increase the permeability of working and non working seams to reduce pre drainage times where virgin state permeability is low and pre drainage is required.
- Increase connectivity between multiple thin but closely spaced non working seams to improve vertical permeability and therefore pre drainage of multiple targets, including interburden.

Nitrogen can also be employed to increase the rate and total volume of methane drained from coal seams (Packham, 2008, Durucan S and Ji-Quan, 2008) by altering the partial and absolute pressure within the coal seam, Figure 5.7. This serves to promote increased permeability due to the pressure applied and desorption of gas by reducing the partial pressure within the coal matrix, possibly resulting in a 5 fold increase in gas flow rates.



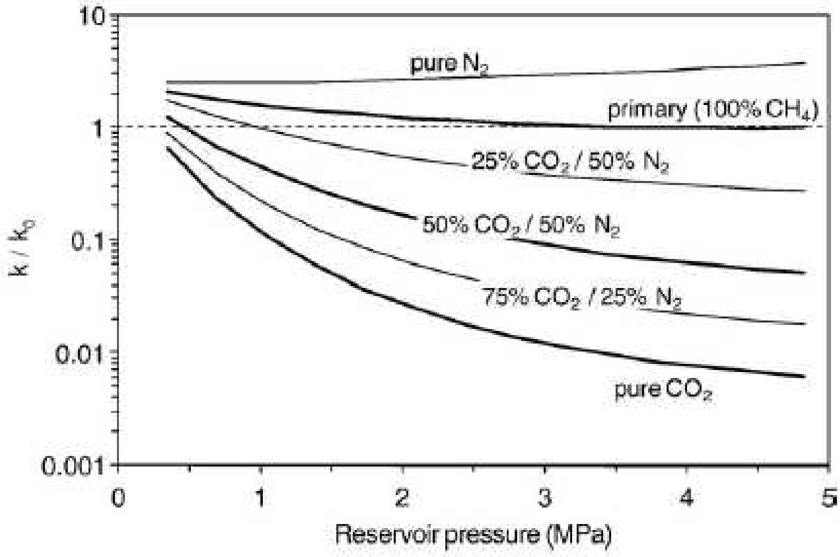


Figure 5.7 Effect of Nitrogen Injection on Pre Drainage Effectiveness (after Sams et al, 2004)

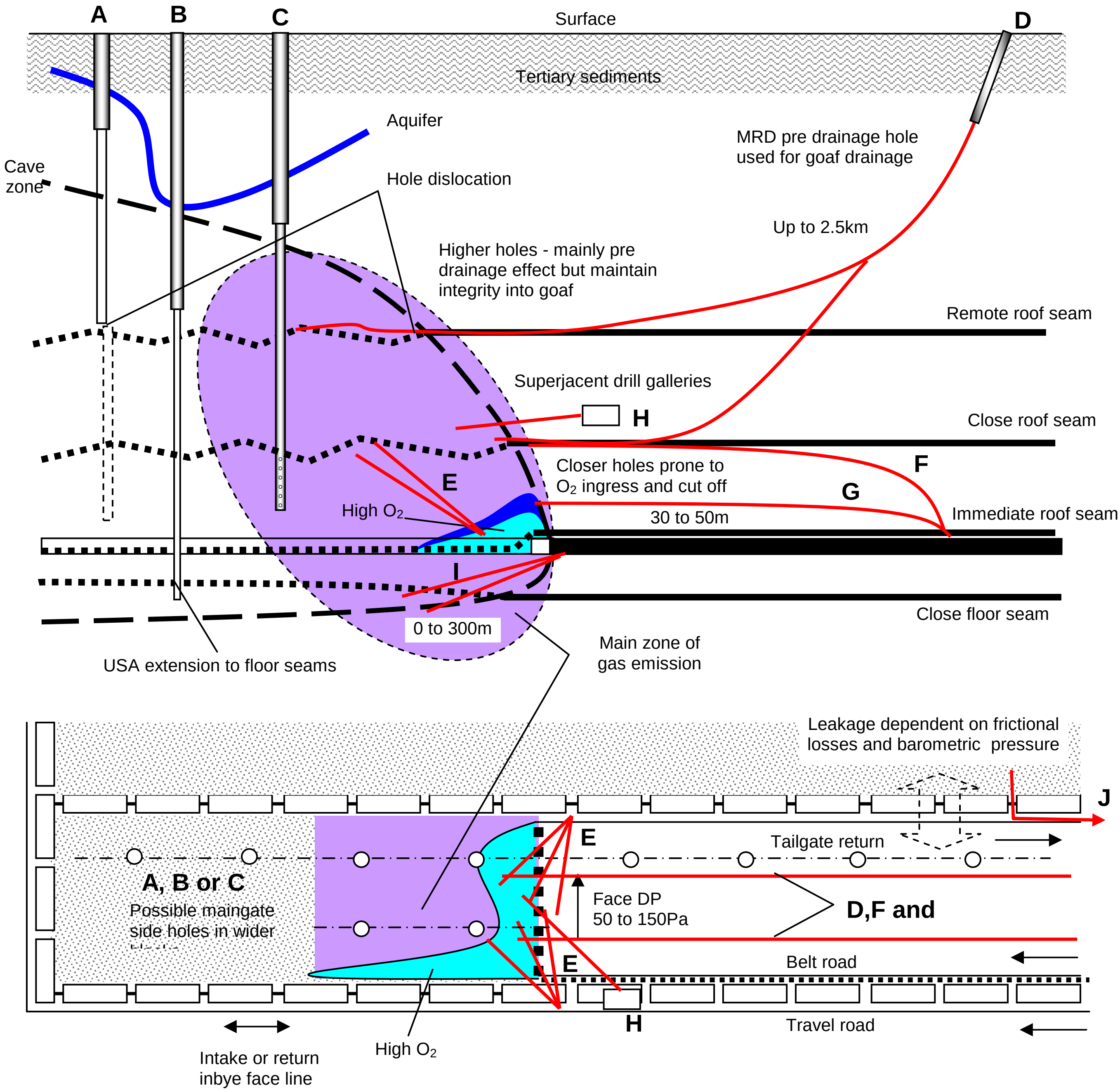
The main advantages of this technique are;

- The technique has been proven in the CBM industry where it continues to be developed together with carbon dioxide sequestration.
- It can be applied to seams using conventional underground pre drainage holes.
- The gas is inert and readily available to the industry using hired nitrogen producing membrane units.
- It promotes and maintains higher permeability in addition to increasing desorption rates.
- The technique enhances pre drainage rates even at lower gas contents that would otherwise not be economic to pre drain.
- Improved gas flow rates would also allow hole spacing to be increased in situations where sufficient nitrogen penetration can be maintained.

5.2 Post Drainage Options

There are a number of post or goaf (gob) drainage options employed worldwide, with selection dependent on mining depth, surface access, risk of spontaneous combustion and degree of gas emission, Figure 5.8. The main operational issues associated with these post drainage strategies in Australian conditions are;

- Maintaining non explosive gas mixtures in reticulation systems when attempting to increase capture efficiency by increasing goaf drainage rates.
- Managing the risk of spontaneous combustion when increased goaf drainage results in an increased ingress of oxygen to the goaf that also contains coal from immediate roof or floor seams.
- Providing sufficient volumetric capacity when dilution increases. With an explosive range nose point of 12% O_2 and an upper explosive limit of 15% CH_4, the limiting dilution ratio for fresh air and a pure methane seam gas would be approximately 57% air to 43% CH_4 i.e. the volume of the gas mixture to be reticulated would then be more than double that of the captured methane. These higher dilution ratios would most likely not be acceptable in the longer term in mines prone to spontaneous combustion although they do provide guidance for appropriate upper trip levels and necessary volumetric capacity.
- With consideration to points 1 and 2, it is often problematic to capture significant fractions of longwall gas emissions in close proximity (1 to 3 pillars) to the face line even though this is identified as the main source of gas emission from immediate or close gas sources (section 4.0 above). Gas capture efficiencies vary depending on the types of hole patterns employed and geometry of non working seam gas sources. As a fraction of total gas emitted during longwall extraction, values range typically between 40 and 60% world wide.
- When a significant fraction of gas emission originates from floor seams it is problematic to capture gas using holes in the roof cave zone as a result of ventilation flow paths and pressure differentials behind face supports.
- Horizontal and vertical displacement of strata may cause holes to be cut off or dislocated thus reducing their effectiveness, particularly when employing semi horizontal or cross measure holes.
- When a conventional U ventilation circuit is employed, seam gas emitted into the working horizon will tend to migrate to the tailgate side of the face. This results in higher oxygen concentrations on the maingate side resulting in more effective surface goaf drainage holes being located some 30 to 40m from the tailgate rib. Consequently, a large fraction of gas emission on the maingate side of the goaf centre line may not be effectively captured leading to the possibility of the gas fringe migrating onto the mid face area, particularly in wider blocks.
- With a conventional two heading U ventilation circuit, tailgate side management is problematic when employing closely spaced underground cross measure holes as collars need to be in the tailgate without protection from a pillar, as they would be if located in the maingate travel road. In addition, increasing flow rates by increasing collar pressure differentials will inevitably increase leakage of air through ribs adjacent to standpipes.



A. Surface goaf drainage open hole, cased through tertiary sediments

B. Surface goaf drainage hole with extended casing to target cave zone

C. Surface goaf drainage hole with slider case to extend draw through cave zone

D. Surface to inseam MRD pre drainage hole for close face environment

E. Underground cross measure rotary or short directional holes to face area and roof seams

F. Underground in roof seam pre drainage hole for close face environment

G. Underground cross measure goaf drainage hole

H. Superjacent drill galleries

I. Cross measure holes for floor seam gas capture.

J. Drainage through seals

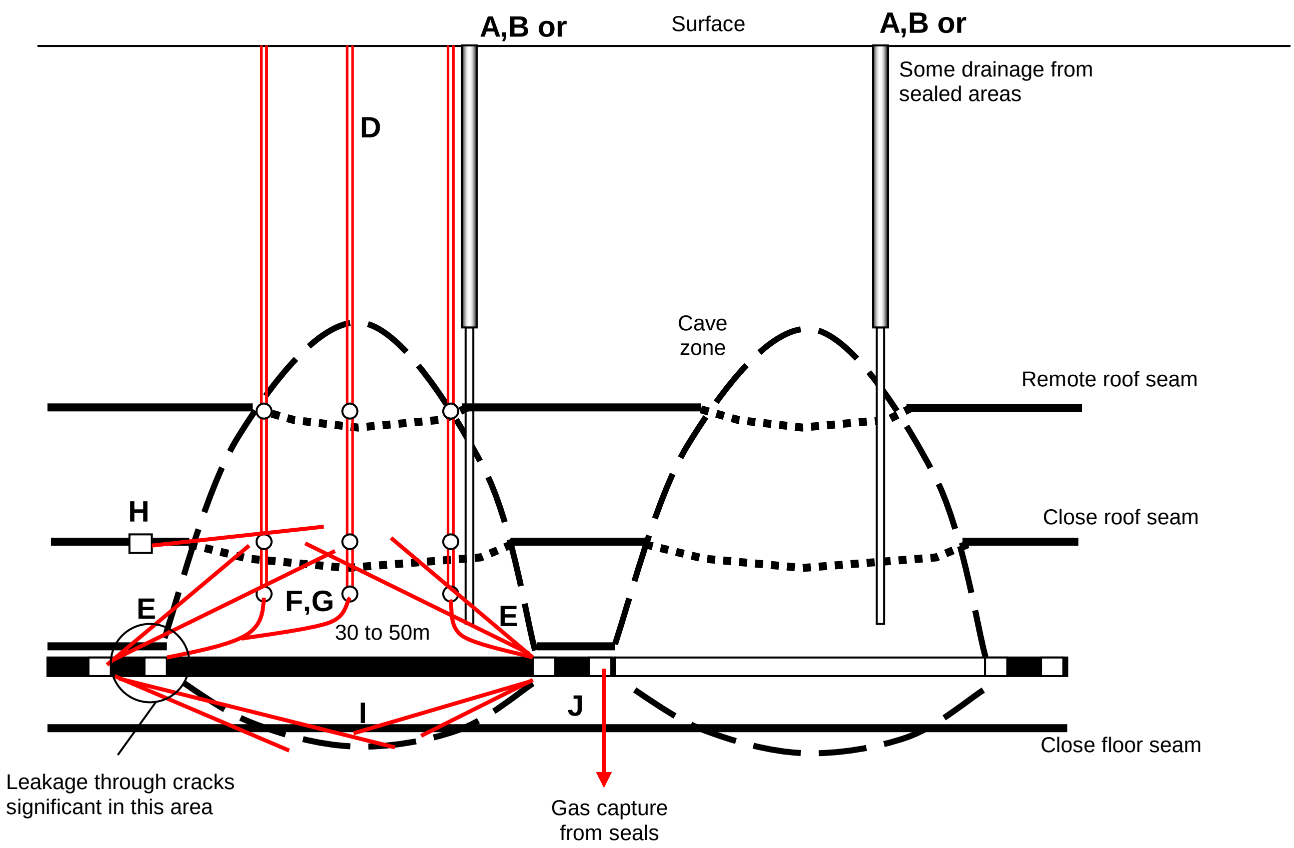


Figure 5.8 Post Drainage Options

- Superimposed on the dynamics of goaf caving are problems arising from pressure differentials and resultant flow paths caused by movement of the shearer, particularly in lower extraction heights. Barometric pressure also has an influence on open and sealed goaf area emissions.

The post drainage challenge is to provide sufficient volumetric capacity and sufficient drainage points to capture a practicable fraction of gas emission at an acceptable concentration for utilisation and at the same time not increasing the level of risk in the mine to, based on prevailing Australian standards, an unacceptable level.

5.2.1 Volumetric Capacity

The volumetric capacity of 300m long holes at various suction pressures are shown in Figure 5.9 for diameters available using currently available underground technology and in Figure 5.10 for larger diameter holes typically drilled vertically from surface although now within the capability of larger rigs from the petroleum industry.

The direct consequence of hole diameter and methane purity on the number of holes required to operate simultaneously for various capture rates is shown in Table 5.3. These values vary according to the carbon dioxide content of seam gas or if seam gas is mixed with oxygen depleted atmospheres but they do serve to indicate the orders of magnitude involved.

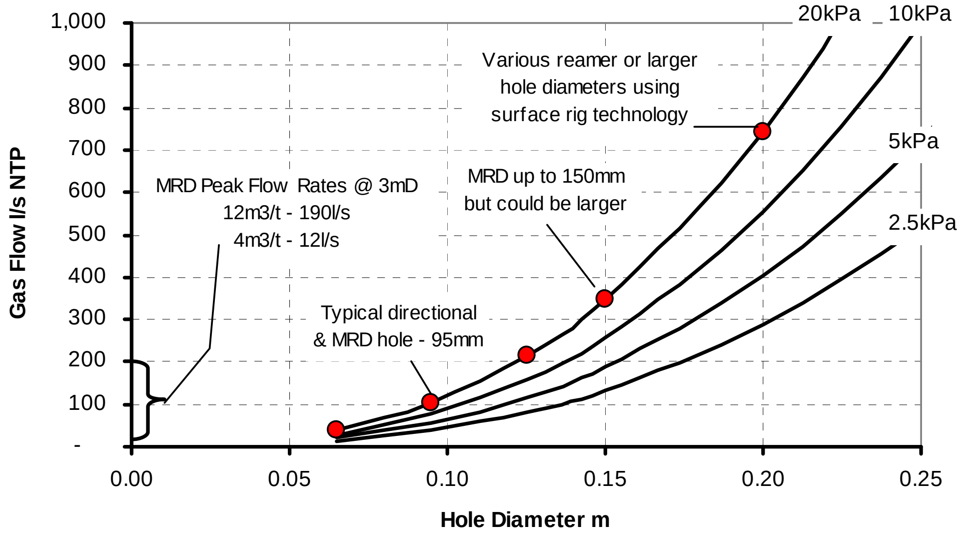


Figure 5.9 Smaller Goaf Drainage Hole Capacities (300m long)

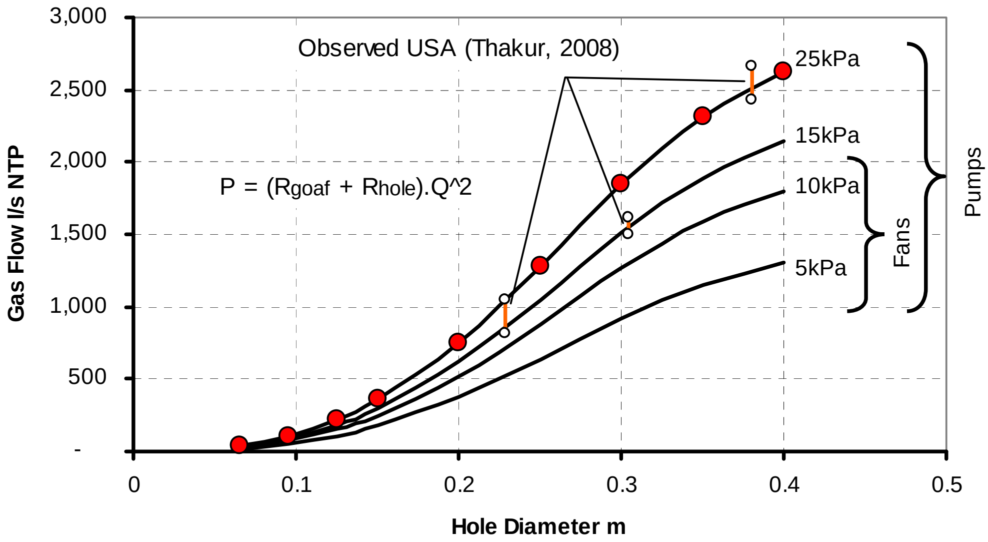


Figure 5.10 Larger Goaf Drainage Hole Capacities (300m long)

It is immediately apparent that the higher potential gas emission rates at increased production rates with only working section pre drainage will make underground post drainage strategies very problematic in gassier conditions and with current hole diameters. For this reason, underground strategies are appropriate at lower emission rates or to supplement surface goaf drainage to improve capture of close face gas emission.

In this respect, more recently available MRD technology with a 2.0 to 2.5km reach using 95mm to 150mm diameter holes may provide opportunities for improvement. The most significant issue is the shear magnitude of gas capture required in gassier Australian mines at higher production rates. This strongly suggests that a combination of whole reservoir pre drainage and high capacity post drainage will be required in many areas.

Table 5.3 Post Drainage Hole Pattern Design

Hole Diam m	15kPa Mixture NTP l/s per hole	100 l/s	500 l/s	1,000 l/s	1,500 l/s	2,000 l/s	3,000 l/s	4,000 l/s	5,000 l/s	6,000 l/s	7,000 l/s	8,000 l/s
		Number of holes required for mixture flow of										
0.095	85	2	6	12	18	24	36	47	59	71	82	94
0.115	143	1	4	7	11	14	21	28	35	42	49	56
0.125	179	1	3	6	9	12	17	23	28	34	40	45
0.150	292	1	2	4	6	7	11	14	18	21	25	28
0.200	617	1	1	2	3	4	5	7	9	10	12	13
0.250	1051	1	1	1	2	2	3	4	5	6	7	8
0.300	1511	1	1	1	1	2	2	3	4	4	5	6
0.350	1891	1	1	1	1	2	2	3	3	4	4	5
0.400	2149	1	1	1	1	1	2	2	3	3	4	4

	Mixed Methane %	100 l/s	500 l/s	1,000 l/s	1,500 l/s	2,000 l/s	3,000 l/s	4,000 l/s	5,000 l/s	6,000 l/s	7,000 l/s	8,000 l/s
		Capture achieved for mixture flow of										
Seam gas	100	100	500	1,000	1,500	2,000	3,000	4,000	5,000	6,000	7,000	8,000
	90	90	450	900	1,350	1,800	2,700	3,600	4,500	5,400	6,300	7,200
	80	80	400	800	1,200	1,600	2,400	3,200	4,000	4,800	5,600	6,400
	70	70	350	700	1,050	1,400	2,100	2,800	3,500	4,200	4,900	5,600
	60	60	300	600	900	1,200	1,800	2,400	3,000	3,600	4,200	4,800
	50	50	250	500	750	1,000	1,500	2,000	2,500	3,000	3,500	4,000
Typical	40	40	200	400	600	800	1,200	1,600	2,000	2,400	2,800	3,200
Minimum	30	30	150	300	450	600	900	1,200	1,500	1,800	2,100	2,400

Seam gas = 100% CH4 *Hole length = 300m* *Collar pressure = 15kPa*

5.2.2 Conventional Surface Goaf Drainage

In Australian conditions where many, but not all, mines have surface access, larger diameter vertical goaf holes provide the preferred solution due to simplicity, lower cost and limited impact on underground operations.

In mines with surface access, the custom and practice for goaf drainage in Australia is to use holes of various diameter (150 to 400mm) and various spacing (100m to 300m) on the tailgate side of the goaf with some means of applying suction to maximise flow rates for the available hole capacity. This is recognised as being the most effective in terms volumetric capacity and cost per m^3/s drained and, where employed with sufficient intensity, is normally sufficient to maintain tailgate methane concentrations below statutory limits. For example, in mines with specific gas emission rates less than $5m^3$/t CH_4, gas emission would range up to 2,000l/s at a production rate of 9Mtpy. With some 600l/s CH_4 reporting to the ventilation system, only 2 or 3 of 250 to 300mm diameter holes would be required to manage the excess for a capture efficiency of 70%.

At higher specific gas emission rates, an ever increasing goaf drainage capacity and capture efficiency is required for similar results. For example, with 600l/s CH_4 reporting to the ventilation system, a goaf drainage efficiency of 90% is required to capture 6,000l/s CH_4. Future deep longwalls producing in excess of 7Mtpy in Australian conditions could emit twice this volume of gas. It is in these circumstances that additional gate road headings are required to provide a higher ventilation capacity for dilution or pre drainage of non working seams must be undertaken to reduce the net specific gas emission rate.

There is a relatively good understanding of surface to goaf drainage characteristics in Australian conditions (Balusu et al, 2004) with a range of success generally dependent on drainage need and spontaneous combustion considerations. Surface to goaf drainage holes have also now been employed in Westcliff colliery at a depth of some 550m (Meyer, 2006). The highest known goaf capture efficiency is about 80% using 250mm holes at 100 to 200m spacing in goaves above the German Creek seam. This efficiency has been achieved due to most, if not all, gas emission originating from roof seams and no coal being left in the immediate goaf.

5.2.3 Underground Goaf Drainage

In mines with limited or no surface access due to topography (NSW Western districts) or for environmental reasons (some NSW Hunter Valley and South Coast) an underground strategy may be required with a consequential increase in the impact on mining operations.

Cross measure post drainage holes have previously been employed in Australia for roof capture (McKennzie and Renee, 1988) using techniques consistent with those employed in the UK and USA (Creedy, 2001, Mutmansky, 1999). Although these techniques do target gas emission in close proximity to the face line, they are labour intensive, relatively inefficient (20 to 40% capture) and are prone to high rates of leakage through coal ribs and goaf.

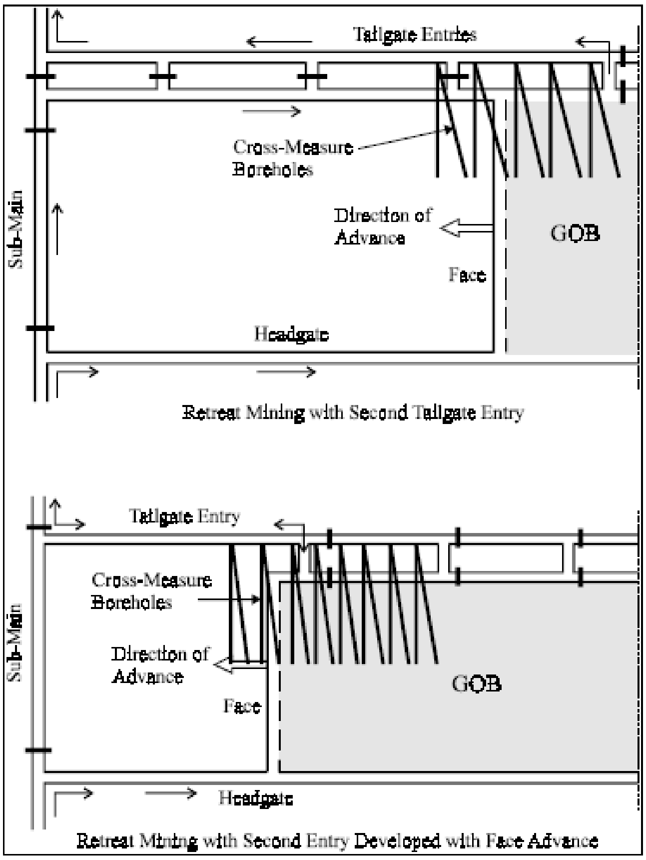


Figure 5.11 Cross Measure Boreholes USA

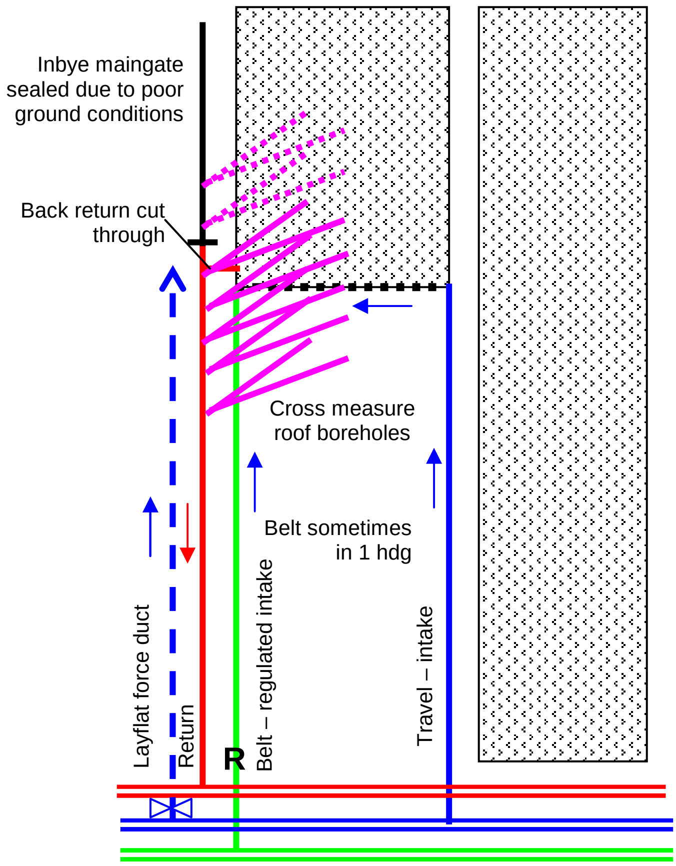


Figure 5.12 Chinese 2 hdgs with Cross Measure

Cross measure rotary holes are most effective when they are on the return side of the face line and protected from the goaf by a pillar, Figure 5.11 (Brunner, 2000). This would not normally be suited to a conventional 2 hdg U circuit in which the tailgate is the single return. In a number of Chinese coal mines the circuit shown in Figure 5.12 is employed albeit with longwalls producing less than 1.0Mtpy.

It is worthy of note that both these US and Chinese circuits represent the highest available ventilation capacity with two heading circuits because they provide intake capacity on both sides of the face, unlike the conventional Australian two heading U circuit. The configuration shown in Figure 5.12 is essentially the same as that employed at Dartbrook (Moreby, 2005) except that, at Dartbrook, the next gate was holed for access inbye the face line instead of using a force duct.

The main problems with return side rotary hole patterns are hole stability when entering the cave zone and lack of drainage on the intake side of the face which would become increasingly problematic in wider blocks. An alternative to rotary cross measure holes is to use directional goaf drainage holes drilled on, or nearly on, the axial direction of the block, for example Figure 5.13 (Brunner and Schwoebel, 2001, Brunner, 2005).

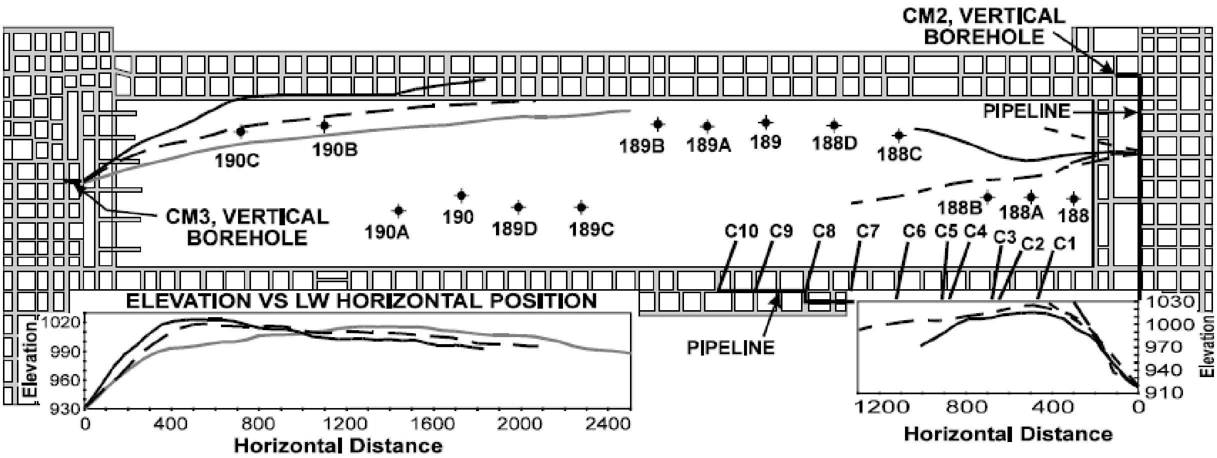
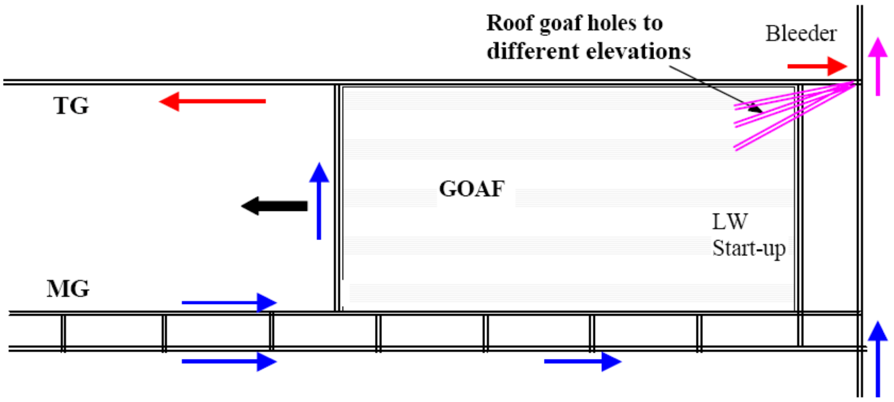


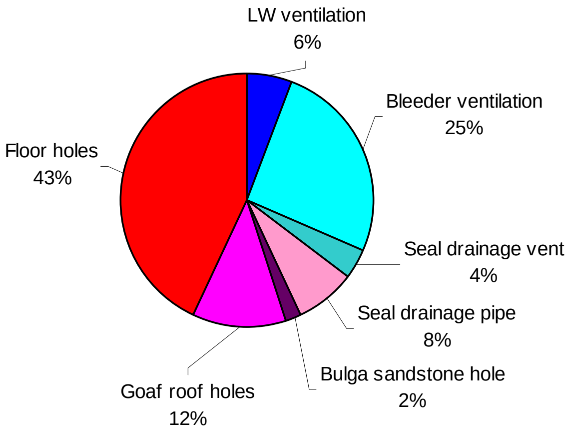
Figure 5.13 Underground Directional Goaf Drainage Holes

In these configurations, 75mm to 150mm directional holes drilled to 1,000m produced 10,000m^3/day (110l/s) gas mixture each, notably consistent with predicted capacity of holes of this diameter, Figure 6.9 above. The holes were located in the cave zone sufficiently high enough to maintain integrity over hole length, 40 to 60m, and required control and monitoring to manage oxygen concentrations.

A similar result from a limited trial of roof holes at Appin Colliery (Balusu et al, 2004) demonstrated that a combination of ventilation, cross measure roof and floor holes together with drainage directly through seals could manage over 5,000l/s CH4 for an overall capture efficiency of about 65%, Figure 5.14. In this situation, it was acceptable to apply high pressure differentials to the goaf in order to direct gas to the bleeder system, a strategy that could not be employed in mines with a higher propensity to spontaneous combustion. In general, a total gas capture efficiency of 30 to 50% is typical with the highest being about 75%. It is to be noted here that, if pre-drainage is ignored, then the gas capture efficiency reduces to 20 to 30% of the total coal mine methane emissions.

Roof hole elevation	CH4 - Gas (%)	Goaf hole operation	Combined gas flow rate (l/s)
45 m above the working seam (6" hole)	85 to 95 %	continuous	
35 m above the working seam (6" hole)	60 to 90 %	continuous	450 to 600 l/s
25 m above the working seam (4" hole)	25 to 70 %	ON and OFF	





Distribution	CH4 l/s
LW ventilation	300
Bleeder ventilation	1,300
Seal drainage vent	200
Seal drainage pipe	400
Bulga sandstone hole	100
Goaf roof holes	600
Floor holes	2,200
Total emission	5,100
To ventilation	1,800
Captured	3,300
Capture efficicency %	65

Figure 5.14 Appin Underground Goaf Drainage Demonstration Trial

Targeting close face goaf drainage by means of directionally drilled holes in the roof of working seams provides a significant opportunity for improvement in Australian mines where longwall gas emissions exceed the practicable dilution capacity of two heading gate road circuits with conventional surface goaf drainage. In mines with lower and more manageable gas emission rates, this gas would otherwise report to the ventilation system. Therefore the optimisation is between these alternative goaf drainage strategies and VAM oxidation systems.

In Australian conditions with surface access, available MRD technology would allow these holes to be drilled from surface possibly also serving a roof seam pre drainage function. When drilled from underground locations in a two heading gate road system, the issues to overcome are rig location and hole trajectories necessary to protect hole collars and initial hole length from leakage when the total hole length is significantly less than the length of the block, for example 1,000m long holes in blocks of 3 to 5km in length.

To overcome these problems, superjacent drilling galleries are employed in some Eastern European and Chinese coal mines in order to increase capture from goaf environments without holes having to pass through or close to the cave zone or stress induced fractures in and around pillars. In these countries, surface access is limited due to terrain or land use. Due to the additional development required, this technique may only be attractive in Australian mines where surface access is not available.

With existing or larger drill rigs, there is opportunity to intensify underground drill patterns for both pre and post drainage in mines that only employ in seam cross block drainage. There are a number of possible target seams in Australia for which this technique could be applied, for example the Aquila and Corvus seams above the German Creek seam or Wonagawilli seam below the Bulli seam.

5.2.4 Sealed Areas and Abandoned Mines

In Australian mines it is custom and practice to promote an inert atmosphere (ideally <4% O_2) in sealed areas in order to ensure that the goaf atmosphere is not in the explosive range and that oxygen concentrations are not sufficient to initiate a spontaneous combustion event. In gassy mines, this can be achieved by sealing and using residual seam gas emission to displace air or, in less gassy mines, by using oxidation of coal in the goaf to cause an inert atmosphere to develop. In both cases, the time taken for inert conditions to be reached can be reduced by introducing an inert gas such as nitrogen or combustion exhaust gases.

In most mine layouts, permanent final seals are adjacent to main returns which means that leakage of gas out of seals is diluted by a significant fraction of the mine’s ventilation capacity. In all but the gassiest Australian mines, this is normally sufficient for compliance purposes.

When they occur, operational problems due to leakage through seals normally manifest themselves in tailgates where seam gas leakage is superimposed on gas emission from the active longwall. Operational problems may also occur during periods of falling barometric pressure in larger bord and pillar mines.

There are four reasons for seam gas to be emitted from sealed areas, as follows;

- Frictional losses in the mine’s ventilation circuit cause pressure differentials to exist across sealed areas, for example tailgate to main return, resulting in continuous leakage into and out of the sealed area. The magnitude of this effect is dependent on characteristics of the ventilation circuit and may range widely from 100Pa up to 3.0kPa.
- During periods of falling barometric pressure the absolute pressure in a mine falls with respect to that inside the sealed area resulting in periodic leakage out of sealed areas at a rate dependent on the rate of change in barometric pressure, the size of the sealed area and integrity of seals. During periods of rising barometric pressure, the direction of leakage is reversed. In Australian east coast and Bowen basin areas, barometric pressure variations of 1.0 to 2.0kPa may occur in a 24 hours period at rates of change up to 500Pa per hour. It is the rate of change that determines the rate of gas emission and the total change in pressure that determines the total volume emitted.
- Changes in the elevation of sealed areas containing a goaf atmosphere with a different density to that in airways adjacent to seals, results in pressure differentials that will cause leakage to flow into and out of sealed areas. For example, in a NSW mines retreating up dip with a change in elevation of 200m over 3.0km, the methane rich goaf atmosphere (0.95kg/m3) causes a pressure differential of about 450Pa (breathing out) across outbye tailgate seals.
- When good quality final seals are placed in mines adjacent to voids without a large number of tailgate seals, such as final long walls or isolated bord and pillar panels,

sealed area pressure can rise towards that of the seam pressure. In one instance, seal differential pressures of 9.0kPa were observed for this reason.

All of these issues can be described mathematically using standard gas laws and ventilation pressure quantity relationships. In all cases it is application of appropriate gas monitoring systems that will provide the information necessary to quantify requirements. For example, variable gas emission from a sealed area in a Bowen Basin mine due to changes in barometric pressure are shown in Figure 5.15.

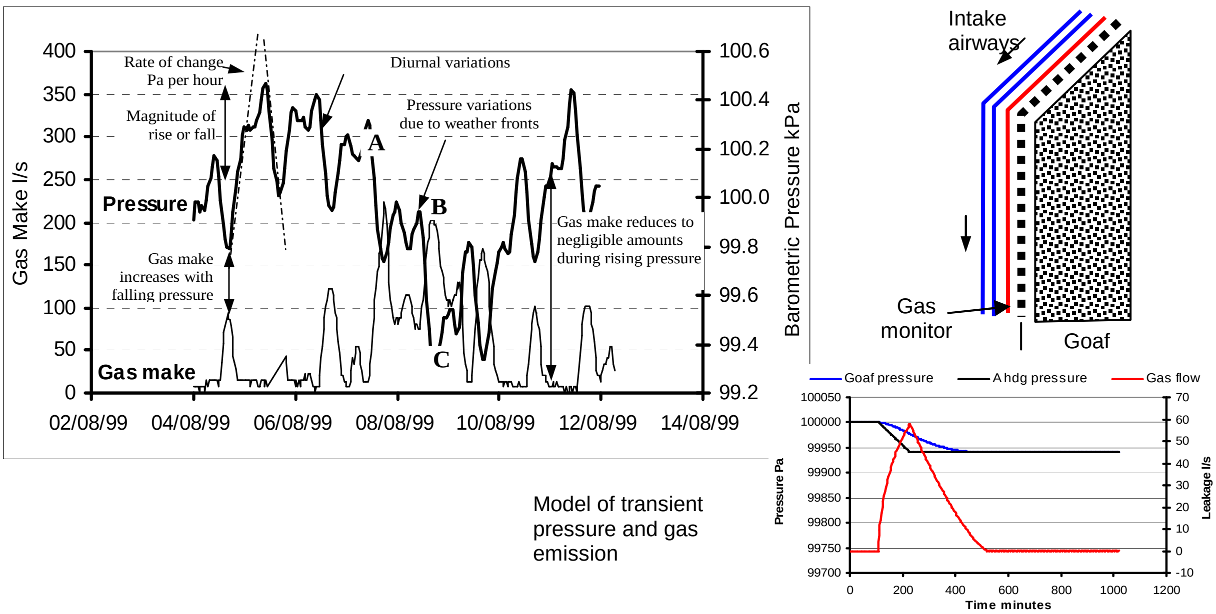


Figure 5.15 Observed Variable Leakage Through Seals

The issues highlighted by this data are;

- For this site, the maximum rate of rise and fall was circa 250 Pa per hour for a duration of 5 to 9 hours. The maximum change in pressure of 400 to 500 Pa was due to weather systems compared to 200 to 400 Pa due to diurnal fluctuations.
- Diurnal fluctuations of 200 to 400 Pa result in goaf gas leakage rates of 100 to 150 l/s. During periods of rising barometric pressure, gas make reduced to zero indicating flow reversal through seals.
- A 560 Pa fall due to a weather front (A – B) resulted in a gas leakage rate of 225 l/s.
- When this fall was followed by a further drop of approximately 500 Pa several hours later (B – C), gas leakage rates returned to about 225 l/s. This indicates stability of goaf pressures had occurred relatively quickly after the first fall, i.e. the effect was not cumulative due to leakage of gas through seals relieving pressure within the goaf.
- It is the rate of change of barometric pressure that determines rate of gas emission rather than the absolute change that occurs.
- In all mines, these transient effects will inevitably lead to variations in the volume of methane reporting to the ventilation system. As the magnitude of gas emission is related to the volume of sealed areas and pressure differentials, they will be more significant (as a fraction of mine average emissions) in low gas emission mines. This in turn will vary the performance and power generating capabilities of VAM units.

Observed gas emission rates through tailgate seals due to changes in elevation and frictional pressure loss are shown in Figure 5.16 for a gassy Hunter Valley mine. Of the total 1,216 l/s CH_4 reporting to the tailgate at 1c/t, 456 l/s or 37.5% was due to leakage through seals. This example is very important in highlighting the magnitude of sealed area emissions that are superimposed on that from the longwall goaf and will be variable during changes in barometric pressure i.e. they will at times be greater than 450l/s requiring at least 45m^3/s for 1%CH_4 due to this leakage alone. In gassier conditions and in longer blocks, this is a very significant operational issue in any event.

With respect to reducing fugitive gas emissions from coal mines, the issues with sealed areas are, firstly to determine how much gas is being emitted, and secondly to provide a means of capturing gas being emitted whilst at the same time maintaining an inert atmosphere within the goaf. Of course the base case is to do nothing other than use VAM units to destroy methane when emitted.

Capture of gas emissions from sealed areas in active mines requires a system that is responsive to absolute pressure, pressure differentials and methane concentrations. The sophistication can range from pipes with one way valves that only vent when seals breath out to variable speed goaf drainage pumps/fans that automatically or manually respond to changes in seal pressure differentials or general body methane concentrations. This latter method was successfully employed at Dartbrook to control CO_2 concentrations in the tailgate intake due to leakage through seals from adjacent goaves.

Control of leakage out of sealed areas, for the purposes of managing gas emission to atmosphere, is very similar to control of leakage into sealed areas for the purposes of mitigating the risk of spontaneous combustion (Brady et al, 2008). The concept of pressure balancing goaf seals is certainly not new and could easily be applied to tackle both issues at the same time. That is, use positive pressure balance chambers, with nitrogen injection in low gas emission mines, to minimise leakage into sealed areas during periods of rising barometric pressure then use the same chamber to capture gas emission during periods of falling barometric pressure. Gas could then be reticulated to some suitable point of discharge for utilisation.

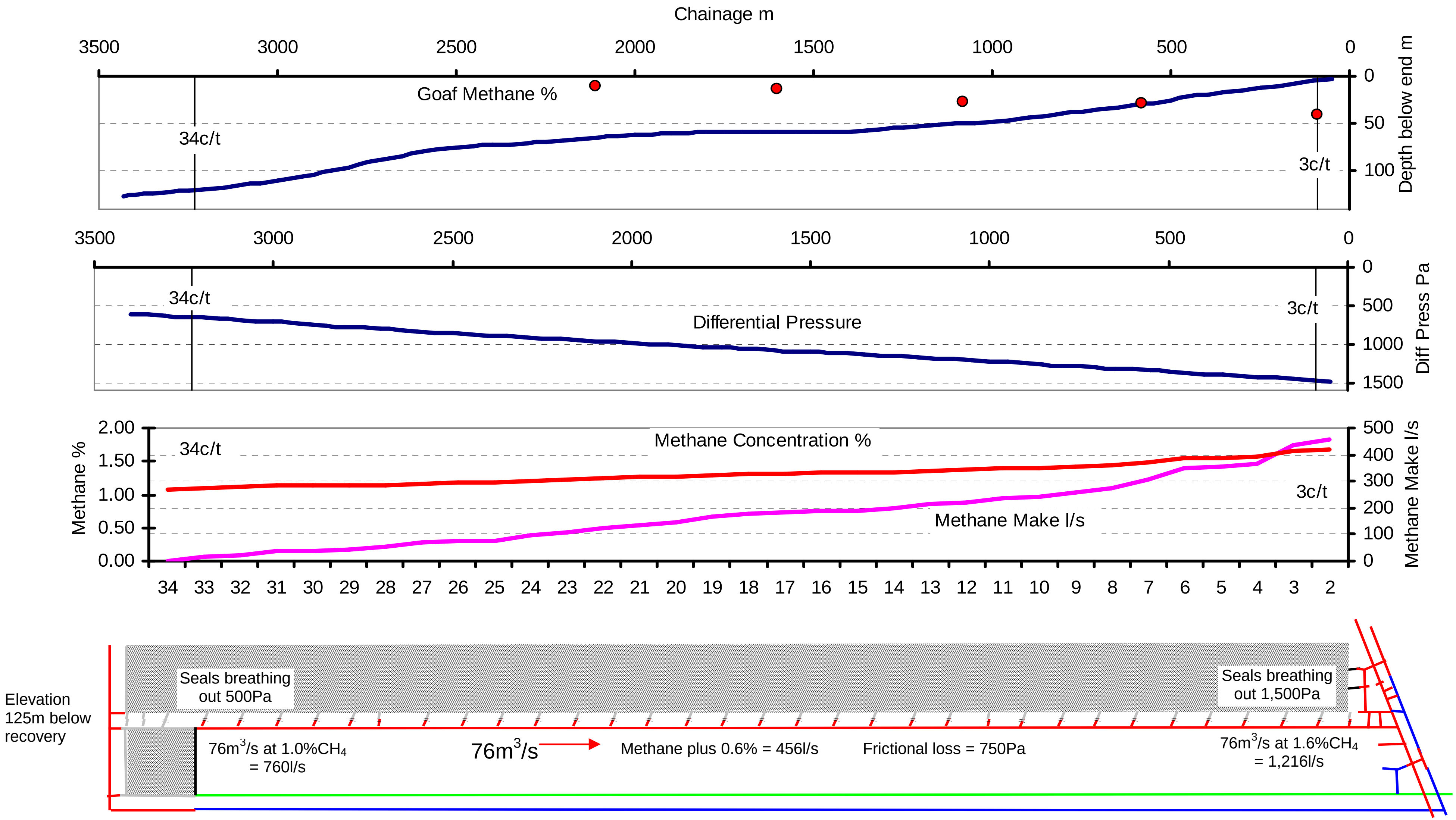


Figure 5.16 Observed Leakage Through Tailgate Seals

The technology required for this type of system is readily available and already employed in a number of mines for various applications. However, and as with most other aspects of this report, the design of an appropriate system will be site specific and may not be applicable in all cases. The main issues to consider are;

- Composition of gas within the sealed area, in particular N_2 and CO_2 composition.
- Relative magnitude of frictional pressure and density effects compared to changes in barometric pressure i.e. some seals may breath out or in under all circumstances where as leakage direction will change through others.
- Propensity for spontaneous combustion in the sealed area.
- Methods of utilisation available at the mine i.e. high purity pre drainage gas streams could be used to increase methane purity by blending if required. Does the mine have VAM units or IC engines.

With respect to health and safety in active coal mines, management of seals and sealed areas is also of particular importance and the subject of new legislation in the USA and international research (MSHA, 2008).

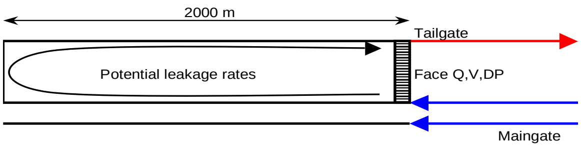
5.2.5 Management of Goaf Atmospheres

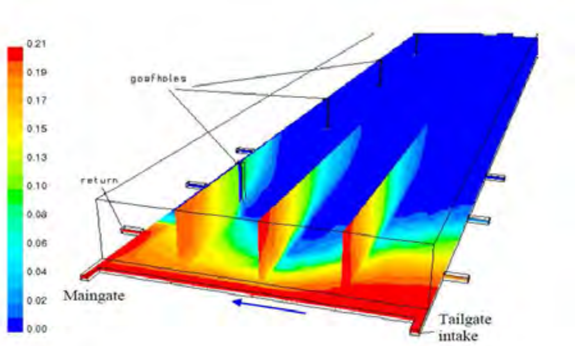
The main limitation on goaf drainage capacity is management of explosive mixtures and spontaneous combustion influenced by ingress of air promoted by drainage rates. It is however the case that significant migration paths exist within all active goaves due to the frictional loss across the face line. When increased ventilation rates are employed to dilute tailgate gas concentrations, it is inevitable that the face pressure drop will increase proportionally to the square of air quantity employed and so too will the degree of migration into and through the goaf. The effect is exacerbated by the fact that, in large goaves with relatively low flow rates (1.0×10^{-5} to 7.0×10^{-4} m/s, Yuan et al 2006), flow may tend towards laminar conditions where the exponent of the pressure quantity relationship will be less than 2, i.e. $P = R.Q^n$ where $1 < n \leq 2$.

For example, the potential for goaf perimeter leakage path flow rates due to longwall face frictional losses is described in Table 5.4. The plan migration distance depends on these frictional losses with vertical migration also affected by gas density, as described graphically by CFD simulation (Balusu et al, 2004).

Table 5.4 Potential Goaf Leakage Rates Due to Face Pressure Differential

Category	Face Quantity m3/s	Face Velocity m/s	Face DP Pa	0.25m2 Deep Leakage m3/s	0.50m2 Deep Leakage m3/s	1m2 Deep Leakage m3/s
Low	30	2.2	56	0.05	0.15	0.43
Typical	40	3.0	100	0.06	0.20	0.57
Typical	50	3.7	157	0.08	0.25	0.71
High	60	4.4	226	0.09	0.30	0.85
High	70	5.2	307	0.11	0.35	0.99
Very high	80	5.9	401	0.12	0.40	1.14





The most significant issue here is the inevitable conflict between increasing face ventilation rates for gas dilution, with or without goaf drainage, and simultaneously promoting inert goaf atmospheres for the control of spontaneous combustion and explosive atmospheres.

Injection of nitrogen can be used as a proactive measure when it is located at an appropriate location behind the face line where it intercepts deep goaf migration but does not get entrained in the higher flow rates behind face shields, Figure 5.17.

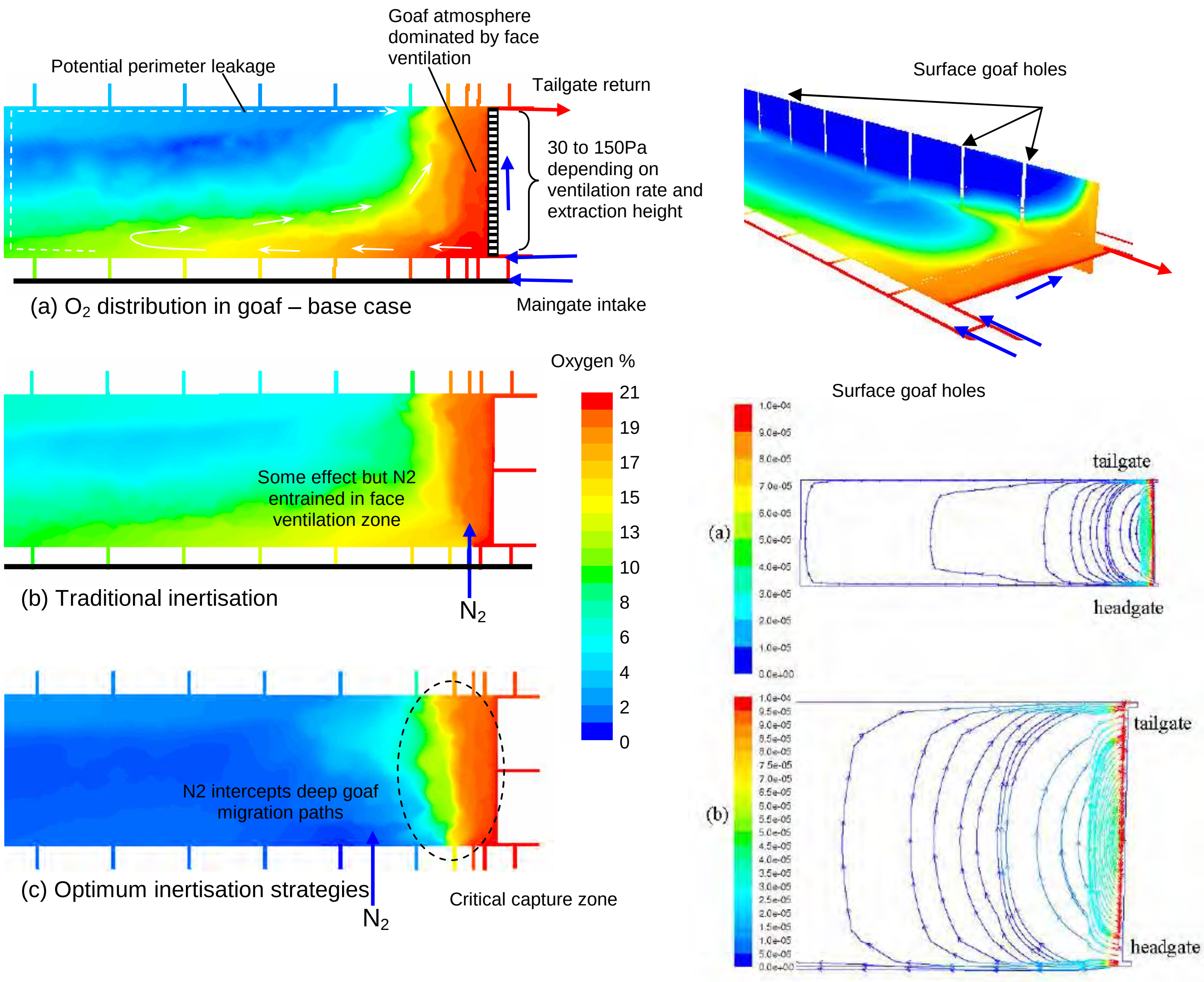


Figure 5.17 Goaf Inertisation Strategies and Flow Profiles

This mode of nitrogen injection is therefore appropriate for assisting the use of goaf drainage further back from the face line but would have limited application for assisting capture in closer proximity to shields. In this area, it is targeted roof holes that provides opportunity for improvement.

5.3 Summary of Gas Drainage Options in Australia

The main pertinent issues arising from gas drainage options in Australia are as follows;

- There is already a significant body of research and operational experience concerning pre and post drainage available to support decision making in Australian underground coal mines.
- No one technique will provide the optimum solution in all mines and a degree of trial and error will be required.
- For pre drainage of working and non working seams, the key issue is lead time. SIS techniques allow for very long lead times necessary for economic whole reservoir pre drainage.
- Surface to inseam post drainage is the most effective where surface access is available. Where it is not, then there are alternatives using rotary or directional holes from underground drill sites.

6. STRATEGIES SUPPORTING MITIGATION

The purpose of this section is to describe operational control strategies available to assist capture of methane at appropriate concentrations whilst also managing core risks such as explosive atmospheres, spontaneous combustion and gas emission to the general body of the mine.

6.1 Ventilation Circuit Configurations

The aim of pre and post drainage strategies is to reduce the amount of seam gas reporting to ventilation systems, historically to maintain compliance with statutory concentration limits but now, subject to feasibility including economic justification, to also reduce fugitive emissions.

The total volume of ventilation required in a mine is determined by the sum of that delivered to working development and longwall panels, standing gate roads, localised plant (TXs, pumps, fuel bays etc) and leakage through stoppings between intake and return airways. In all Queensland mines and some NSW mines, belt road segregation is also employed to provide two secure means of egress in intake air. Currently it is not required to totally segregate main belt roads in Australia and “belt road ventilation” is employed in working panels, otherwise it would be an additional ventilation requirement.

The absolute minimum ventilation rate employed in mines with low or negligible seam gas emission rates, for example some of those in the Western districts of NSW, is determined by specified minimum standards for auxiliary fans, diesel equipment and air velocity (0.3m/s in NSW) in panels multiplied by the number of panels in the mine plus an allowance for leakage.

The maximum ventilation rate is determined by limiting intake, face and return air velocities together with the practicable number of main and gate road headings employed. Currently only one coal mine in Australia employs three heading gate roads with others employing a variety of two heading configurations, some with but most without perimeter bleeders.

The range of Australian ventilation capacities for typical two heading development supporting a single longwall is shown in Table 6.1, which can be compared with the actual ventilation rates employed, see section 3.5. The nominal range for single longwall operations is a minimum 165m^3/s and a maximum of about 495m^3/s. It is understood that the maximum ventilation rate currently planned for in Australia is 550m^3/s in a gassy mine operating two sets of mains for a single longwall and with significant outbye leakage due to a surface fan pressure of circa c5.5kPa.

With consideration to a typical gassy Australian mine being one in which development gas emission is not problematic (due to low permeability and or pre drainage for outburst control) but longwall gas emission is problematic due to the limited volumetric capacity of a single tailgate return, this demonstrates that the longwall ventilation rate is typically about 30% of the mine total ventilation capacity but may contain about 70% of gas reporting to the ventilation system. It is also the case that gas dilution inefficiencies within the circuit arising from planned and unplanned leakage will dilute gas emission from panels in main returns. Where longwall gas emission is problematic, it is invariably the tailgate rather than main returns that exceed various methane concentration limits. This is more so the case today with improved standards for final seals in all mines i.e. reduced leakage from outbye sealed areas.

Table 6.1 Range of Australian Mine Ventilation Capacities

Activity	Minimum Quantity m3/s	Maximum Quantity m3/s	Typical Quantity m3/s	Typical Gassy Return CH4%	Typical Gassy Return CH4 l/s
Mains development	30	60	40	0.4	160
Leading gate development	25	50	35	0.5	175
1st Lagging gate development	35	70	50	0.6	300
2nd Lagging gate development	0	70			
Longwall tailgate	30	80	50	1.5	750
Longwall bleeder returns (1)	0	70	40	1.5	600
Total mining panels	120	400	215		1,985
Fuel bay	10	10	10		
TX and pumps	5	15	10		
Outbye leakage (2)	30	70	50		
Total miscellaneous	45	95	70		
Total mine	165	495	285	0.70	1,985
Volumetric efficicency % (3)	73	81	75		

(1) Bleeders or exhaust raises not always employed in gassy conditions
(2) Leakage depends on size of mine and quality of stoppings
(3) Efficicency is ventilatlion to panels compared to mine total

The operating envelopes describing ventilation rates and gas management capacity by dilution for all Australian mine conditions are provided in Table 6.2 for methane and Table 6.3 for carbon dioxide.

The main issues arising from this analysis is that, at production rates of 3.0 to 5.0+Mtpy, practicable total mine ventilation capacities can manage most gas emission rates (<5,000l/s CH_4) to less than 1.0%CH_4 but practicable two heading longwall ventilation capacities can only manage up to about 1.4m^3/s CH_4 without bleeder headings and about 2.2m^3/s CH_4 with bleeder headings. These values are of course subject to development profiles and characteristics of surface fans employed but they do provide a reasonable orders of magnitude assessment of practicable limits.

In terms of CO_2-e, 1.4m^3/s CH_4 in a tailgate represents about 0.6Mtpy CO_2-e. In gassy mines where longwall tailgate emission are high but the remainder of the mine is relatively low, there may therefore be significant benefit in targeting VAM solutions at the single tailgate return, or more gassy gate roads.

Table 6.2 Magnitude of Methane Emissions to Ventilation

			Range main returns									Range tailgate & bleeders				
					Lower Limit VAM						Upper Limit VAM	Limit for diesels	Limit for power			Limit for workers
		Vent	Methane Emission m3/s for General Body Methane % of													
	Description	**m3/s**	**0.1**	**0.2**	**0.3**	**0.4**	**0.5**	**0.6**	**0.7**	**0.8**	**0.9**	**1.0**	**1.25**	**1.50**	**1.75**	**2.00**
	Very low	20	0.02	0.04	0.06	0.08	0.10	0.12	0.14	0.16	0.18	0.20	0.25	0.30	0.35	0.40
	Low	30	0.03	0.06	0.09	0.12	0.15	0.18	0.21	0.24	0.27	0.30	0.38	0.45	0.53	0.60
		40	0.04	0.08	0.12	0.16	0.20	0.24	0.28	0.32	0.36	0.40	0.50	0.60	0.70	0.80
Range tailgate	Typical	50	0.05	0.10	0.15	0.20	0.25	0.30	0.35	0.40	0.45	0.50	0.63	0.75	0.88	1.00
+ bleeder quantity		60	0.06	0.12	0.18	0.24	0.30	0.36	0.42	0.48	0.54	0.60	0.75	0.90	1.1	1.2
	High	70	0.07	0.14	0.21	0.28	0.35	0.42	0.49	0.56	0.63	0.70	0.88	1.05	1.2	1.4
		80	0.08	0.16	0.24	0.32	0.40	0.48	0.56	0.64	0.72	0.80	1.0	1.2	1.4	1.6
	Very high	90	0.09	0.18	0.27	0.36	0.45	0.54	0.63	0.72	0.81	0.90	1.1	1.4	1.6	1.8
		100	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80	0.90	1.0	1.3	1.5	1.8	2.0
	Very low	150	0.15	0.30	0.45	0.60	0.75	0.90	1.1	1.2	1.4	1.5	1.9	2.3	2.6	3.0
Range total		200	0.20	0.40	0.60	0.80	1.0	1.2	1.4	1.6	1.8	2.0	2.5	3.0	3.5	4.0
mine quantity		250	0.25	0.50	0.75	1.0	1.3	1.5	1.8	2.0	2.3	2.5	3.1	3.8	4.4	5.0
	Typical	300	0.30	0.60	0.90	1.2	1.5	1.8	2.1	2.4	2.7	3.0	3.8	4.5	5.3	6.0
		350	0.35	0.70	1.1	1.4	1.8	2.1	2.5	2.8	3.2	3.5	4.4	5.3	6.1	7.0
		400	0.40	0.80	1.2	1.6	2.0	2.4	2.8	3.2	3.6	4.0	5.0	6.0	7.0	8.0
	High	450	0.45	0.90	1.4	1.8	2.3	2.7	3.2	3.6	4.1	4.5	5.6	6.8	7.9	9.0
		500	0.50	1.0	1.5	2.0	2.5	3.0	3.5	4.0	4.5	5.0	6.3	7.5	8.8	10.0
	Very high	550	0.55	1.1	1.7	2.2	2.8	3.3	3.9	4.4	5.0	5.5	6.9	8.3	9.6	11.0

Vent	Carbon Emission CO_2-e Mt/y for General Body Methane % of													
m3/s	**0.1**	**0.2**	**0.3**	**0.4**	**0.5**	**0.6**	**0.7**	**0.8**	**0.9**	**1.0**	**1.25**	**1.50**	**1.75**	**2.00**
20	0.01	0.02	0.03	0.04	0.04	0.05	0.06	0.07	0.08	0.09	0.11	0.13	0.15	0.18
30	0.01	0.03	0.04	0.05	0.07	0.08	0.09	0.11	0.12	0.13	0.17	0.20	0.23	0.27
40	0.02	0.04	0.05	0.07	0.09	0.11	0.12	0.14	0.16	0.18	0.22	0.27	0.31	0.35
50	0.02	0.04	0.07	0.09	0.11	0.13	0.15	0.18	0.20	0.22	0.28	0.33	0.39	0.44
60	0.03	0.05	0.08	0.11	0.13	0.16	0.19	0.21	0.24	0.27	0.33	0.40	0.5	0.5
70	0.03	0.06	0.09	0.12	0.15	0.19	0.22	0.25	0.28	0.31	0.39	0.46	0.5	0.6
80	0.04	0.07	0.11	0.14	0.18	0.21	0.25	0.28	0.32	0.35	0.4	0.5	0.6	0.7
90	0.04	0.08	0.12	0.16	0.20	0.24	0.28	0.32	0.36	0.40	0.5	0.6	0.7	0.8
100	0.04	0.09	0.13	0.18	0.22	0.27	0.31	0.35	0.40	0.4	0.6	0.7	0.8	0.9
150	0.07	0.13	0.20	0.27	0.33	0.40	0.5	0.5	0.6	0.7	0.8	1.0	1.2	1.3
200	0.09	0.18	0.27	0.35	0.4	0.5	0.6	0.7	0.8	0.9	1.1	1.3	1.5	1.8
250	0.11	0.22	0.33	0.4	0.6	0.7	0.8	0.9	1.0	1.1	1.4	1.7	1.9	2.2
300	0.13	0.27	0.40	0.5	0.7	0.8	0.9	1.1	1.2	1.3	1.7	2.0	2.3	2.7
350	0.15	0.31	0.5	0.6	0.8	0.9	1.1	1.2	1.4	1.5	1.9	2.3	2.7	3.1
400	0.18	0.35	0.5	0.7	0.9	1.1	1.2	1.4	1.6	1.8	2.2	2.7	3.1	3.5
450	0.20	0.40	0.6	0.8	1.0	1.2	1.4	1.6	1.8	2.0	2.5	3.0	3.5	4.0
500	0.22	0.4	0.7	0.9	1.1	1.3	1.5	1.8	2.0	2.2	2.8	3.3	3.9	4.4
550	0.24	0.5	0.7	1.0	1.2	1.5	1.7	1.9	2.2	2.4	3.0	3.6	4.3	4.9

Table 6.3 Magnitude of Carbon Dioxide Emissions to Ventilation

Range tailgate & bleeders: 0.1 to 3.00
Range main returns: 0.1 to 0.9

8hr TWA Limit (QLD): 0.5
8hr TWA Limit (NSW): 1.25
STEL Limit: 3.00

	Description	Vent m3/s	**Carbon Dioxide Emission m3/s for General Body Carbon Dioxide % of**												
			0.1	**0.2**	**0.3**	**0.4**	**0.5**	**0.6**	**0.7**	**0.8**	**0.9**	**1.0**	**1.25**	**2.00**	**3.00**
	Very low	20	0.02	0.04	0.06	0.08	0.10	0.12	0.14	0.16	0.18	0.20	0.25	0.40	0.60
	Low	30	0.03	0.06	0.09	0.12	0.15	0.18	0.21	0.24	0.27	0.30	0.38	0.60	0.90
		40	0.04	0.08	0.12	0.16	0.20	0.24	0.28	0.32	0.36	0.40	0.50	0.80	1.20
Range tailgate	Typical	50	0.05	0.10	0.15	0.20	0.25	0.30	0.35	0.40	0.45	0.50	0.63	1.00	1.50
+ bleeder quantity		60	0.06	0.12	0.18	0.24	0.30	0.36	0.42	0.48	0.54	0.60	0.75	1.20	1.8
	High	70	0.07	0.14	0.21	0.28	0.35	0.42	0.49	0.56	0.63	0.70	0.88	1.40	2.1
		80	0.08	0.16	0.24	0.32	0.40	0.48	0.56	0.64	0.72	0.80	1.0	1.6	2.4
	Very high	90	0.09	0.18	0.27	0.36	0.45	0.54	0.63	0.72	0.81	0.90	1.1	1.8	2.7
		100	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80	0.90	1.0	1.3	2.0	3.0
	Very low	150	0.15	0.30	0.45	0.60	0.75	0.90	1.1	1.2	1.4	1.5	1.9	3.0	4.5
Range total		200	0.20	0.40	0.60	0.80	1.0	1.2	1.4	1.6	1.8	2.0	2.5	4.0	6.0
mine quantity		250	0.25	0.50	0.75	1.0	1.3	1.5	1.8	2.0	2.3	2.5	3.1	5.0	7.5
	Typical	300	0.30	0.60	0.90	1.2	1.5	1.8	2.1	2.4	2.7	3.0	3.8	6.0	9.0
		350	0.35	0.70	1.1	1.4	1.8	2.1	2.5	2.8	3.2	3.5	4.4	7.0	10.5
		400	0.40	0.80	1.2	1.6	2.0	2.4	2.8	3.2	3.6	4.0	5.0	8.0	12.0
	High	450	0.45	0.90	1.4	1.8	2.3	2.7	3.2	3.6	4.1	4.5	5.6	9.0	13.5
		500	0.50	1.0	1.5	2.0	2.5	3.0	3.5	4.0	4.5	5.0	6.3	10.0	15.0
	Very high	550	0.55	1.1	1.7	2.2	2.8	3.3	3.9	4.4	5.0	5.5	6.9	11.0	16.5

Vent m3/s	**Carbon Emission CO_2 Mt/y for General Body Carbon Dioxide % of**												
	0.10	**0.20**	**0.30**	**0.40**	**0.50**	**0.60**	**0.70**	**0.80**	**0.90**	**1.00**	**1.25**	**2.00**	**3.00**
20	0.00	0.00	0.00	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02	0.04
30	0.00	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.02	0.02	0.02	0.04	0.05
40	0.00	0.00	0.01	0.01	0.01	0.01	0.02	0.02	0.02	0.02	0.03	0.05	0.07
50	0.00	0.01	0.01	0.01	0.01	0.02	0.02	0.02	0.03	0.03	0.04	0.06	0.09
60	0.00	0.01	0.01	0.01	0.02	0.02	0.02	0.03	0.03	0.04	0.04	0.07	0.1
70	0.00	0.01	0.01	0.02	0.02	0.02	0.03	0.03	0.04	0.04	0.05	0.08	0.1
80	0.00	0.01	0.01	0.02	0.02	0.03	0.03	0.04	0.04	0.05	0.1	0.1	0.1
90	0.01	0.01	0.02	0.02	0.03	0.03	0.04	0.04	0.05	0.05	0.1	0.1	0.2
100	0.01	0.01	0.02	0.02	0.03	0.04	0.04	0.05	0.05	0.1	0.1	0.1	0.2
150	0.01	0.02	0.03	0.04	0.04	0.05	0.1	0.1	0.1	0.1	0.1	0.2	0.3
200	0.01	0.02	0.04	0.05	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.2	0.4
250	0.01	0.03	0.04	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.2	0.3	0.4
300	0.02	0.04	0.05	0.1	0.1	0.1	0.1	0.1	0.2	0.2	0.2	0.4	0.5
350	0.02	0.04	0.1	0.1	0.1	0.1	0.1	0.2	0.2	0.2	0.3	0.4	0.6
400	0.02	0.05	0.1	0.1	0.1	0.1	0.2	0.2	0.2	0.2	0.3	0.5	0.7
450	0.03	0.05	0.1	0.1	0.1	0.2	0.2	0.2	0.2	0.3	0.3	0.5	0.8
500	0.03	0.1	0.1	0.1	0.1	0.2	0.2	0.2	0.3	0.3	0.4	0.6	0.9
550	0.03	0.1	0.1	0.1	0.2	0.2	0.2	0.3	0.3	0.3	0.4	0.7	1.0

VAM oxidation units, such as those employed at Westcliff , each have a volumetric capacity of $17m^3/s$ of gas air mixture and can operate in the range 0.3 to 0.9%CH_4 (further technological development may move these boundaries further but only by 0.1 to 0.2%CH_4). Subject to recent changes in international exchange rates, these units are understood to have a capital cost of between $2.0M and $3.0M.

Although these units can be applied to main exhaust shafts, this would normally be inefficient due to dilution of principal methane sources within the mine's main returns. Possible solutions to this are to employ dedicated panel exhaust raises where surface access permits or to segregate returns to a separate exhaust shaft, Figure 6.1.

In practice, tailgates normally operate below 2.0% CH_4 but with diesel access being problematic at or above 1.0% CH_4. Consequently, most tailgate return concentrations are at or below 1.0% CH_4 in any event with the gassier mines planning to operate up to about 1.8% CH_4 to provide a buffer to the 2.0% trip level. In most mines, it is possible to pass 50 to $60m^3/s$ across the face line providing a gas dilution capacity of 500 to 600l/s at 1.0% CH_4 and 1,000 to 1,200l/s at 2.0% CH_4. A split return system would therefore only be attractive in mines where there are significant variations in return methane concentrations between development and longwall panels.

In mines employing longwall ventilation, in excess of that required for management of dust and heat, to dilute seam gas emission, there may be opportunity to employ a panel split system but with economic feasibility dependent on the quantity of gas involved. A further limitation would be the degree to which daily peak gas emissions exceed the daily average, although on a two heading system (single tailgate return) diesels normally enter the tailgate after rather than during production due to dust exposure considerations. The degree of variability in longwall gas emissions can be seen in an example from a currently operating mine in Figure 6.2. Split system dilution requirements and shaft sizes are shown in Table 6.4.

Operational issues to consider are:

- inadvertent plugs of high methane concentrations passing through the ventilation circuit;
- pressure differentials increasing the risk of spontaneous combustion;
- management of dust and heat;
- as gas emission rates increase so to does the volume of air required for dilution to <0.9% making shaft diameters larger and more expensive;
- Coupling VAM units to the ventilation system with out the risk of ignition; and
- VAM units are best operated with stable methane concentrations which means that the dilution would be easier to manage on surface rather than underground. This stabilisation could also include use of pre drainage gas to raise concentrations when necessary.

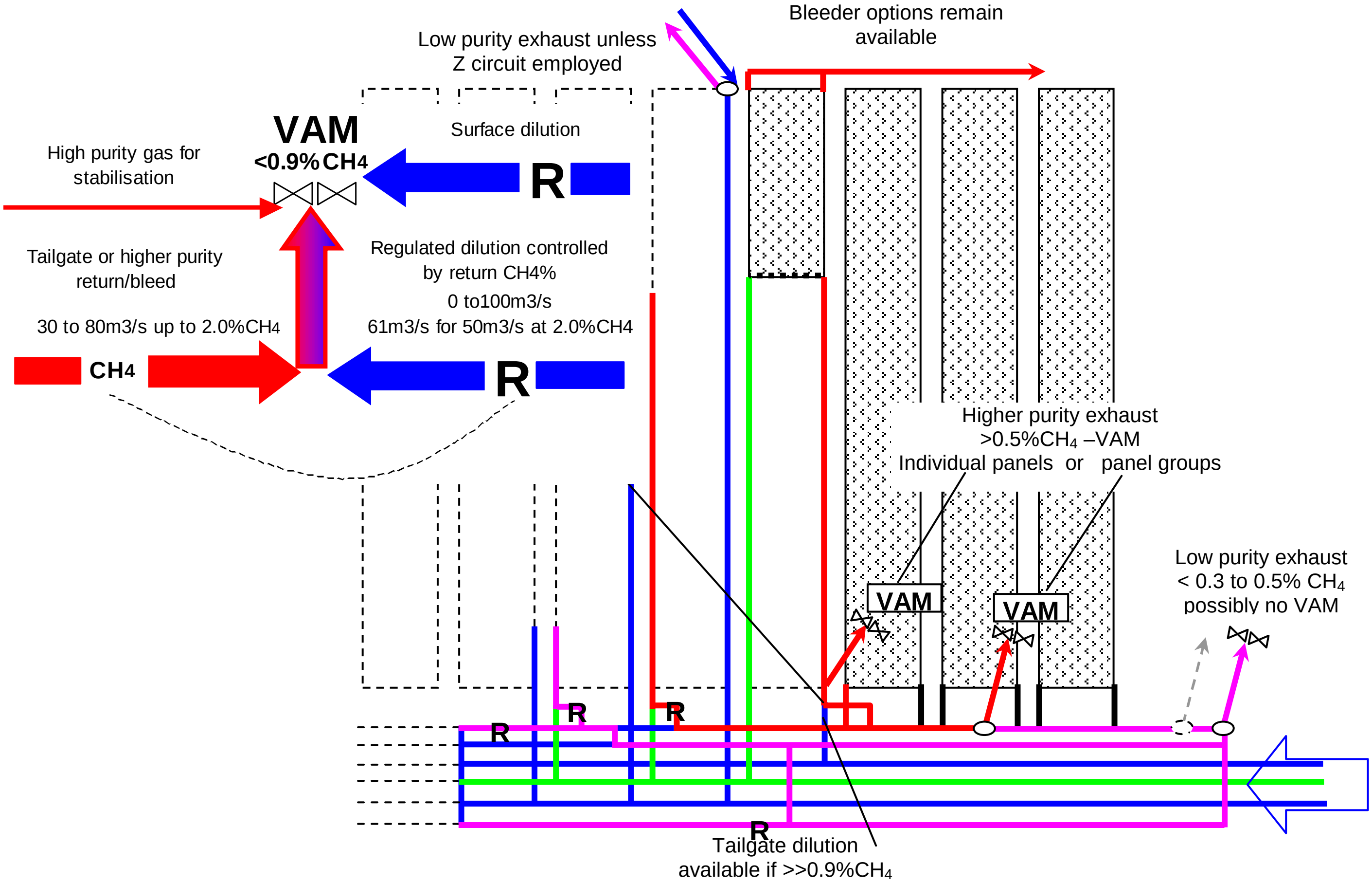


Figure 6.1 Location of VAM Units with Split Returns

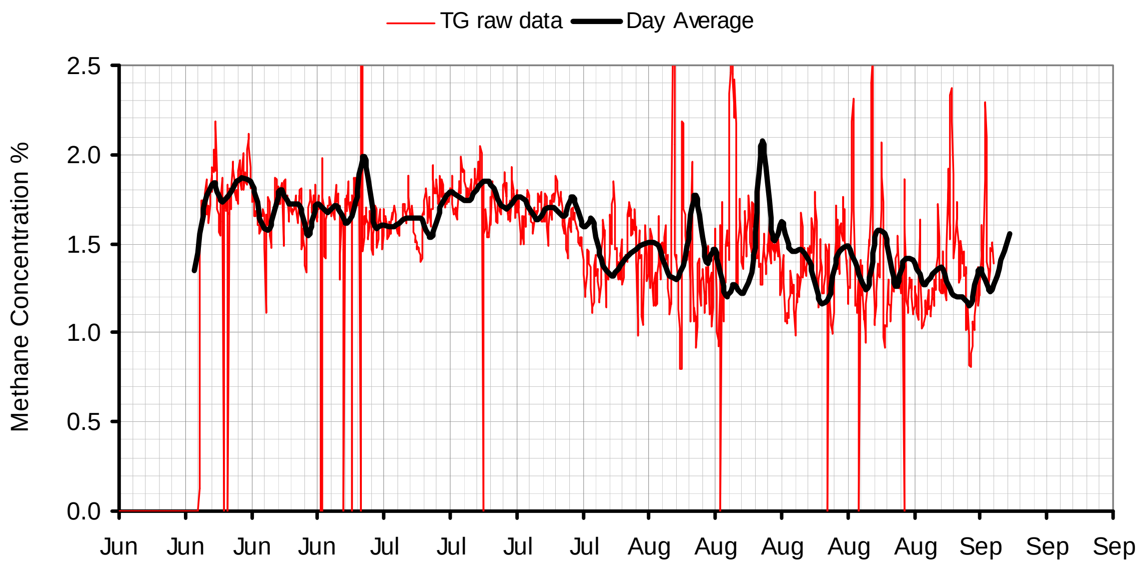


Figure 6.2 Variability of Longwall Tailgate Return Concentrations (30 min data interval)

Table 6.4 Split Ventilation System Dilution Requirements and Exhaust Shaft Sizes

Increasingly ineffective gas capture

					Diesel Limit	Power Limit				Personnel Limit
	Tailgate	Total Mixed Flow (m3/s) for Maximum 0.9% for Tailgate Concentrations of								
	Quantity m3/s	0.4 %CH4	0.6 %CH4	0.8 %CH4	1.0 %CH4	1.2 %CH4	1.4 %CH4	1.6 %CH4	1.8 %CH4	2.0 %CH4
Low	30	30	30	30	33	40	47	53	60	67
	40	40	40	40	44	53	62	71	80	89
Typical	50	50	50	50	56	67	78	89	100	111
Increasingly	60	60	60	60	67	80	93	107	120	133
ineffective High	70	70	70	70	78	93	109	124	140	156
gas capture	80	80	80	80	89	107	124	142	160	178

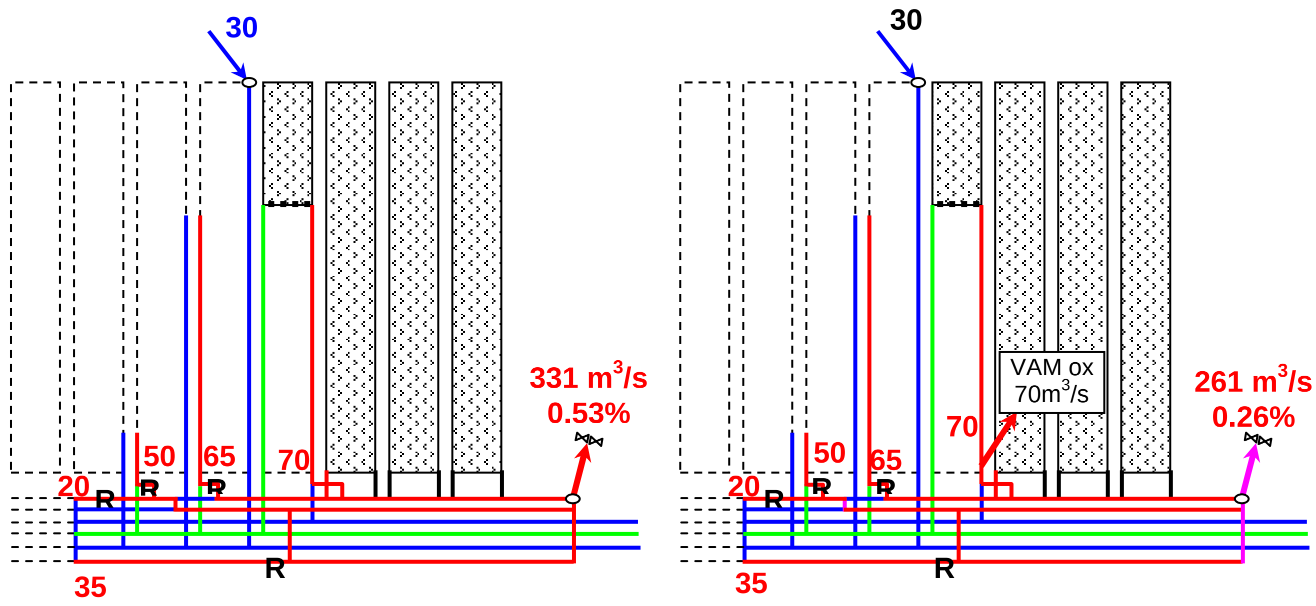
	Tailgate	Exhaust Shaft Diameter (m) for 16m/s for Tailgate Concentrations of								
	Quantity m3/s	0.4 %CH4	0.6 %CH4	0.8 %CH4	1.0 %CH4	1.2 %CH4	1.4 %CH4	1.6 %CH4	1.8 %CH4	2.0 %CH4
Low	30	1.5	1.5	1.5	1.6	1.8	1.9	2.1	2.2	2.3
	40	1.8	1.8	1.8	1.9	2.1	2.2	2.4	2.5	2.7
Typical	50	2.0	2.0	2.0	2.1	2.3	2.5	2.7	2.8	3.0
	60	2.2	2.2	2.2	2.3	2.5	2.7	2.9	3.1	3.3
High	70	2.4	2.4	2.4	2.5	2.7	2.9	3.1	3.3	3.5
	80	2.5	2.5	2.5	2.7	2.9	3.1	3.4	3.6	3.8

Quite obviously, the feasibility of split ventilation systems that employ additional surface shaft connections will depend on site specific conditions such as surface access, depth of cover and strata stability issues.

A basic decision making strategy is shown in Figure 6.3 for comparison between base case conventional and split return systems for the tailgate only.

In this example, splitting the return with destruction of tailgate VAM emissions would reduce the potential CO_2-e emissions from 0.78 million tonnes (1.74 m^3/s of methane) to 0.31 Mt (0.69 m^3/s of methane), which results in significant savings. The main exhaust shaft ventilation rate would then reduce to 261m^3/s at an average 0.26% CH_4.

Further reductions involving capture of development gas emissions are shown in Table 6.5 and Figure 6.4. This type of analysis should a then be compared with the need for and relative cost of pre and post drainage strategies. For example, in mines prone to outbursts, the optimum strategy may be pre drainage to just below outburst thresholds for minimum development emissions with VAM destruction strategies focused on tailgate emissions. As with all other aspects of decision making, preferred strategies will vary with site specific conditions and the actual cost of emission per t CO_2-e.



Location	Base Case Ciruit Parameters Allocated Ventilation m3/s	Methane %	Methane m3/s	VAM Shaft Applied	Modified Split Circuit Ventilation to exhaust # m3/s	Emitted Methane %	Emitted Methane m3/s
Development							
Mains	55	0.3	0.165	No	55	0.3	0.165
Leading gate	65	0.5	0.325	No	65	0.5	0.325
1st lagging gate	50	0.4	0.200	No	50	0.4	0.200
2nd lagging gate	0			No			
	170				170		
Longwall							
Tailgate return	70	1.5	1.050	Yes	0	0	0.000
Maingate homotropal	0		0.000	No	0	0	0.000
Bleeder return	0		0.000	No	0	0	0.000
	70				0		
Services	25				25		
Outbye leakage	66				66		
	331	0.53	1.740		261	0.26	0.690

Figure 6.3 Example Decision Making

Table 6.5 Example of Individual Panel VAM Destruction

Scenario	Main Shaft m3/s	Main Ex Shaft CH4 %	VAM Shaft(s) m3/s
No VAM destruction	331	0.53	0
Tailgate only	261	0.26	70
Taigate and lead gate	196	0.22	135
Taigate and both gates	146	0.11	185

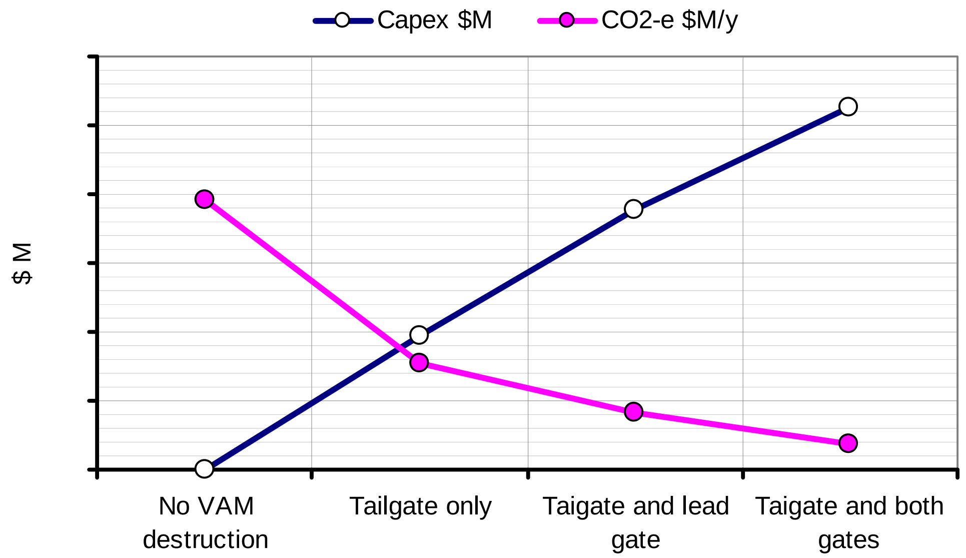


Figure 6.4 Individual Panel VAM Destruction

6.1.1 Bleeder Ventilation Systems

Although there are clearly exceptions in both countries, the main difference between the “norm” in Australia and the US, is the perceived or actual degree of risk introduced by direct goaf bleeder systems. The hazards being explosive mixtures and promotion of spontaneous combustion events by introduction of oxygen to goaf atmospheres. It is also the case that the cost of and lower advance rates achieved by conventional CM development in most Australian coal mines, leads to, all but one operating mine, avoiding three heading gate roads for operational reasons.

In essence, a bleeder system is a split ventilation system with the design intent of capturing most if not all longwall gas emission. The difference being that a true bleeder system draws ventilation through the goaf, or the goaf edge, in order to minimise the quantity of gas reporting to the face or tailgate end return. It is this aspect that brings Australian (particularly Queensland) mine regulations, guidelines and custom and practice into conflict with those in the US.

True bleeder systems are employed in a number of South Coast Bulli seam coal mines where all coal is extracted from the working section and the seam has a relatively low propensity to spontaneous combustion, Figure 6.5. In these circuits, high differential pressures are applied to the goaf by means of a maingate side regulator with additional dilution being provided by main bleeder intake airways. However, problems still arise when the face line is remote from rear goaf bleed points and the majority of floor gas emission (Belgownie and Wongawilli seams) reports to the tailgate return in any event.

The three heading alternative with controlled bleed is shown in Figure 6.6. In this circuit it is possible to use the bleeder shaft as the sole longwall return provided that an airway can be maintained in centre tailgate side road way. This may involve an additional line of seals to segregate the goaf.

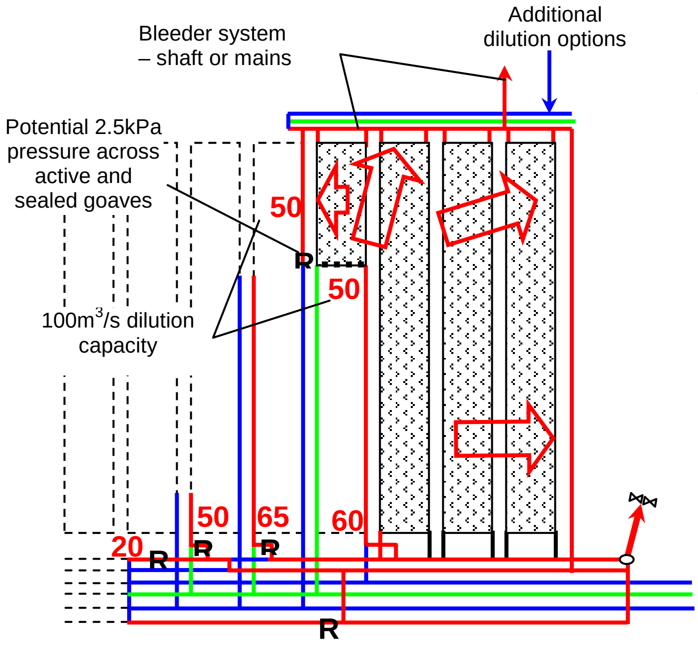


Figure 6.5 Australian Two Heading Bleeders

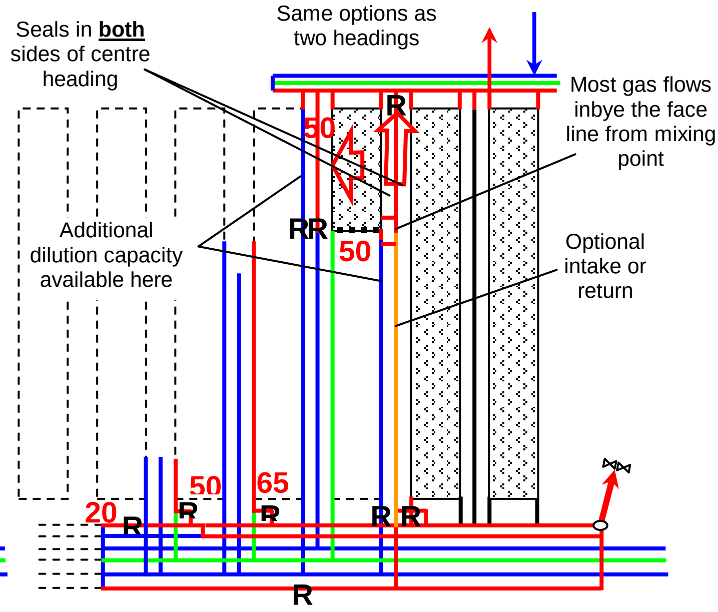


Figure 6.6 Three Heading Bleeders with Control

Both two and three heading bleeder options provide a means of capturing more longwall gas emissions and delivering it to a minor shaft where VAM abatement systems can be employed, as with a split return system. There are numerous operational and gas management advantages in employing three heading gate roads, particularly in longer blocks, and they would be appropriate in some Australian coal mines if costs and advance rate issue can be overcome.

It should be recognised that a number of other seams in Australia, for example the German Creek seam, could employ controlled goaf bleed systems on a two or three heading basis particularly if pro active injection of nitrogen was also employed. However, in all Australian coal mines with a higher propensity for spontaneous combustion and or significant quantity of coal in the immediate goaf, direct goaf bleed systems will most likely remain unacceptable and the split return system can be employed instead.

6.2 Ventilation Monitoring

The measurement of gas concentrations and ventilation flow rates in underground coal mines has been undertaken internationally for more than a century with the historical focus being on prevention of the occurrence of explosive mixtures in the general body of the mine and assessing seam gas emission rates from working panels or gaseous products of oxidation to monitor the spontaneous combustion or fire status of a mine.

All underground coal mines in Australia have underground real time telemetric gas monitoring systems together with gas sensors fitted to mine equipment. These are generally suitable for the measurement of gas emission concentrations for the purposes of assessing fugitive gas emissions and gas streams reporting to gas utilisation equipment (Day and McPhee, 2008, NGER, 2008). These real time telemetric systems are often supported by additional monitoring of gas concentrations by tube bundle systems using infra red sensors for CO, CO_2 and CH_4 and paramagnetic sensors for O_2 with further analysis by gas chromatograph available for confirmation and other gases such as H_2 and C_2H_4.

The main problem in coal mines is the measurement of ventilation velocity (m/s) required to calculate flow rates (m^3/s) for calculation of the volumetric or mass flow rate of gaseous emissions. In mines where gas emission is not problematic from a statutory compliance point of view, ventilation rates may only be measured once per month using handheld calibrated vane anemometers. In other mines, parts of the ventilation circuit may be surveyed more frequently although a mine wide balance is usually only obtained once per month. Regardless of the requirements for measurement of fugitive gas emission, this is a significant problem that can result in errors of gas make calculations for both seam gas emission and indicators of spontaneous combustion e.g. CO make l/min.

In addition to the frequency of measurement, an identified issue with current practice is that the Lambrecht type, or similar, mechanical anemometers are no longer available and a number of mines have resorted to using electronic devices with much smaller plastic impellors. It is not known if these devices have been or continue to be checked against calibrated anemometers as there is no way of adjusting the plastic vanes. These smaller impellors are unlikely to be suitable in lower air velocities.

There are two distinct issues to address. Firstly, calculation of fugitive gas emissions in ventilation air being emitted from the mine, in which case it is the volumetric flow rate through surface fans that is important, and secondly, calculation of seam gas emissions (and spontaneous combustion indicators) from various parts of the mine, in which case the issue is to provided a means of routinely measuring air velocity in underground airways.

6.2.1 Surface Fan Monitoring

It is understood that differential pressure devices fitted to most surface fans are provided by the fan manufacturers and meet, or could meet, international design standards. These devices provide accurate results when first installed or calibrated with the problem being one of ongoing maintenance.

Clogging with particulate matter and condensation is also a problem encountered with surface fan annubar pressure differential devices to the extent that it was "*universally felt* [by mine ventilation officers] *that, at best, these instruments were only suitable to indicate that the fans were maintaining flow*", Day, 2008. A possible solution is to fit a compressed air line with solenoid to provide a reverse flush of compressed air at an appropriate frequency, say 10 to 15 minute intervals. However, there are significant advantages in employing these pressure differential devices, as follows;

- There is no flow through the device and therefore the effect of clogging is reduced.
- IS versions of absolute and differential pressure devices are readily available.
- The pressure sensing device can be remote from the point of measurement allowing it to be located in intake airways, with flame arrestors in the connecting tubes if required.
- Pressure differential is normally proportional to the square of velocity, therefore changes in velocity are amplified making the measurement more precise.

Another problem with differential pressure devices fitted into the transition duct between exhaust shaft collars and fans is the significant degree of turbulence and often large pressure fluctuations that occur. In some metalliferous mines, shaft velocities are measured with large diameter (50 to 75mm ID) pitot tubes suspended in the shaft from surface. A similar approach has been used for measuring air velocity and overpressure resulting from underground windblast events (Fowler and Sharma, 2000). Using a back flush of compressed air, these devices may provide an alternative method for measuring the velocity in exhaust shafts and underground roadways.

In any event, the fan static pressure obtained by the difference between the shaft collar piezometer ring and shaft velocity pressure can be checked against the fan characteristics curves and absorbed electrical power.

It is understood that some metalliferous mines have installed ultrasonic devices across fan ducts with success. The problems with ultrasonic and other real time devices, such as those described by McDaniel et al, 1999, are the need for devices to be approved for use in hazardous zones and clogging by the continuous use of stone dust in coal mine return airways. Assuming that an approved device becomes available then these would also provide a solution to the problem, refer Figure 6.8 below for a possible solution.

6.2.2 Underground Monitoring

Trailing Hose and Pressure Differentials - the trailing hose method of measuring frictional pressure loss is appropriate for air quantity determinations in circumstances where the frictional loss is sufficient to provide a large enough value to measure. These may not be suitable for individual panel returns in which the air velocity was low.

For example, the frictional loss (Pa) and frictional loss per m^3/s (Pa per m^3/s) for 500m of typical 3m x 5m airway are shown in Figure 6.7. At higher velocities the frictional loss would be approximately 200Pa per 500m or some 1.5 to 2.0 Pa per m^3/s which could be measured with high quality DP devices.

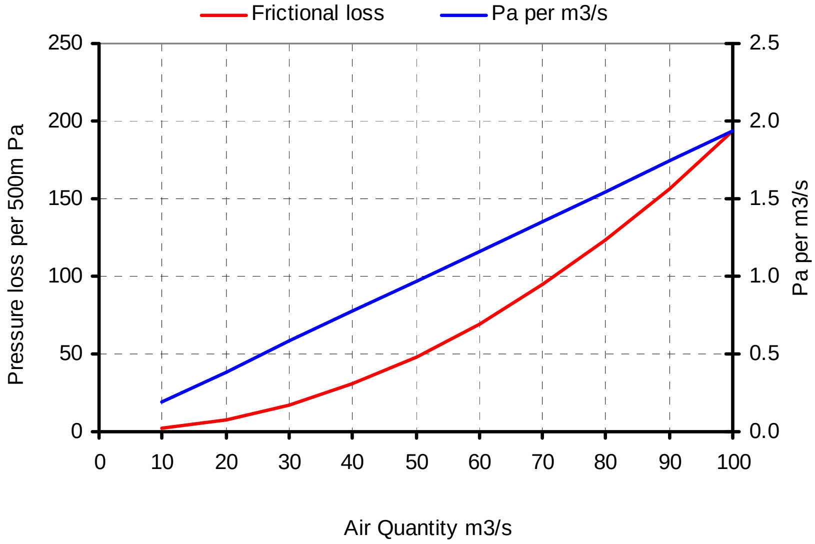


Figure 6.7 Air Quantity Determination from Frictional Losses per 500m

Electronic Velocity Devices - there are a number of electronic devices available for measuring air velocity in metalliferous mines with the cross airway ultrasonic devices including air reversal capabilities being favoured. With consideration to the problems of installation and maintenance in coal mine return airways, the obvious solution is to recognise the general simplicity of coal mine circuits, as compared to multi level metalliferous mine circuits, and install the devices in intake airways (non hazardous or NERZ) instead.

In most Australian coal mines this would only require two to four monitoring points to determine total mine airflow rates. Similar devices could also be located in the intake travel roads of longwall and gate road panels.

It is understood that at least one such ultrasonic device (Accutron Plus) is currently undergoing the Testsafe approval process and is on trial in at least one under ground mine. This device is reported to be insensitive to dust, humidity and temperature and is suitable for both airway and fan locations, Figure 6.8. If this, and other similar sensors, meet their specifications then they will be appropriate for this aspect of ventilation monitoring.

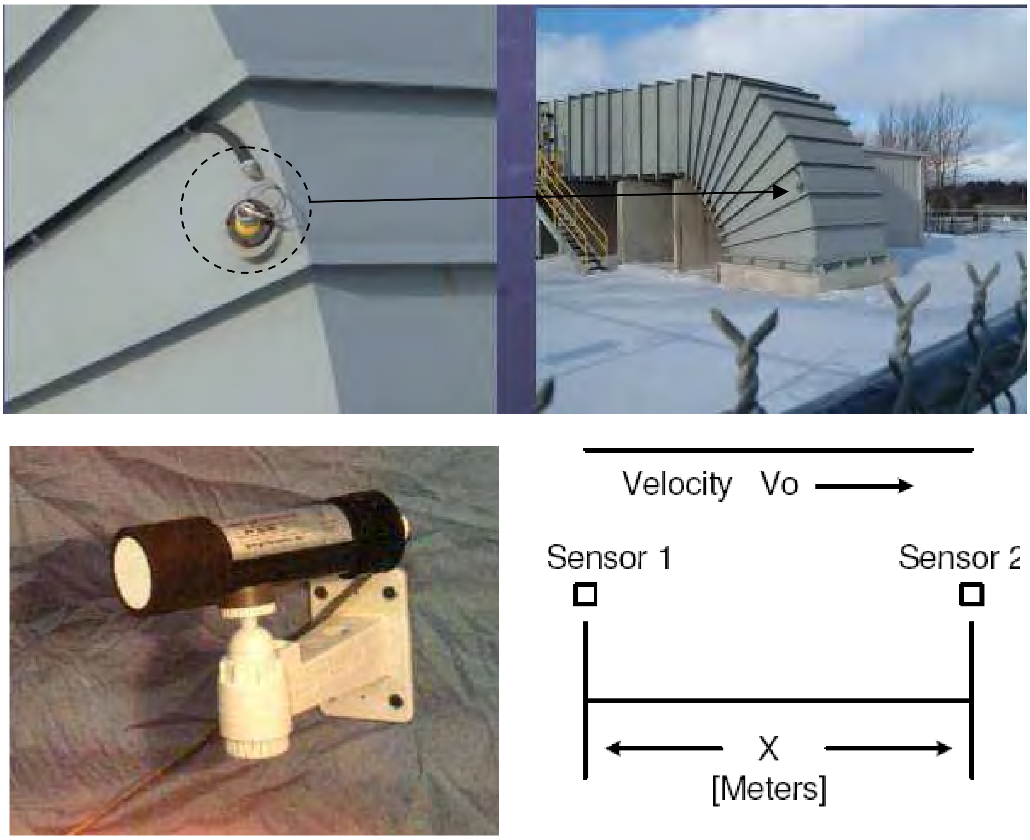


Figure 6.8 Ultrasonic Velocity Sensor (Accutron Plus)

6.2.3 Pressure and Temperature

In order to provide a mass balance or to report emission rates at a standard or normal condition, it is necessary to measure both temperature and absolute pressure.

Absolute pressure is normally obtained from a pressure transducer reporting to the mine control system. These devices are located on surface allowing the absolute pressure in surface intakes or underground workings to be calculated from monitored differential pressures, for example exhaust shaft collar static pressures.

Air temperature is not normally monitored in mine exhaust airways, even in Queensland mines in which active heat stress management is required. Dry bulb temperature probes are readily available for this purpose but wet bulb temperatures would have to determined indirectly by measuring relative humidity. Alternatively, for the purpose of calculating air density, it would most likely be acceptable to assume a constant wet bulb depression in most mine exhaust shafts i.e. evaporation of moisture in most coal mines combined with auto decompression in exhaust shafts would lead to relatively stable saturated or near saturated conditions. This is most likely also the case in gas drainage systems, particularly where virgin strata temperatures exceed ambient mine air temperatures.

Acceptable measurement of pressure and temperature should not be problematic with available instrumentation.

6.2.4 Gas Concentrations and Moisture Content

Although most mine gas monitoring systems are, if properly calibrated, suitable for measuring gas concentrations for the purposes of calculating GHG emissions, it is important to recognise the effect of moisture content. In monitoring systems that dry the gas mixture (by chilling or passing over a drying agent) the absolute gas concentrations reported are higher than actual by an amount equal to the volume occupied by water vapour. This is most important in gas drainage systems although also applies to ventilation monitoring.

The worked example in Table 6.6 indicates the effect in terms of gas flow rate and CO_2-e emissions. The dry gas methane concentration will always be higher than the true wet case value and therefore overestimate emissions if applied to wet gas flow rates. In this case, (300m^3/s ventilation at 0.5%) there would be an 43l/s overestimate with a significant annual CO_2-e charge, if applicable.

Table 6.6 Effect of Moisture Content on Gas Emission Calculations

Item	Unit	Value
Actual methane concentration (wet)	vol %	0.500
Moisture content (ASV)	g/kg	0.883
Moisture content	vol %	2.780
Dry gas methane concentration	vol %	0.514
Measured air flow rate	m3/s	300
Actual methane emission	m3/s	1.500
Methane emission (dry gas value)	m3/s	1.543
Difference	m3/s	0.043

It is therefore important that wet gas concentrations are used to calculate emission from ventilation and gas drainage systems.

6.3 Alternative Mitigation Strategies

The various ventilation and gas drainage strategies described above are concerned with capture and destruction of methane released as a result of mining activity and at a rate determined by the gas reservoir characteristics present. The following are alternative mitigation strategies to consider although very much dependent on economic justification.

Reduce permeability of non working seams	Water injection has been used to reduce rib emission and various other treatments are employed to stabilise ground, for example grout injection or freezing. If it were possible to inject an appropriate fluid into non working seams it may be possible to reduce gas emission during mining – either reducing the total emitted or delaying emission by increasing the effective time constant thus increasing emission to sealed areas which is more manageable.
Reduced caving	Material placement into the active goaf in order to reduce the zone of influence – in a similar manner to cemented fill (CAF) used in metalliferous mines. This could either be continuous or periodic placement to establish a pillar effect. Obviously, material volumes

	required would be problematic.
Covered Stockpiles and Silos	If stockpile emissions are significant gas could be captured on surface possibly with some additional treatment to increase gas desorption or gas stripping e.g. nitrogen or increased wash temperatures.
Out of seam development, pre drainage through mining LCBM, Large surface rigs in underground environment	Use significantly larger drill rigs underground in circumstances where surface access is limited or to increase the diameter of holes employed. This would require designed for purpose drill locations similar to the superjacent methods used internationally.
Top coal caving in thick seams	Reduces gas emission underground but this would report to surface. Significant when the immediate roof contains gas and a large proportion of the gas reservoir.
Sequestration of exhaust gases	Possibility of pumping surface engine and flare exhaust back into degassed seams or goaf areas – subject to adsorption characteristics.

7. MINE SITE DECISION MAKING

The purposes of this section is to provide an operational and economic rational for decision making at mine sites. The issues to consider are as follows;

- What is the basic economic position of a mine with respect to gas streams and consequential CO_2-e charges, if and when applicable?
- Should and when should mines introduce pre drainage if they could otherwise operate in compliance using ventilation alone during development or longwall phases?
- In mines where longwall gas emission rates are problematic, to what extent can pre drainage techniques be applied to both working and non working seams?
- To what extent can post drainage strategies be improved to increase the fraction of seam gas available for utilisation at high (>>30% CH4) purity in mines with and without gas management problems during longwall production?
- What is a realistic and optimum balance between pre drainage, post drainage and VAM oxidation strategies for mitigation of fugitive gas emissions at both existing and future production rates?

With consideration to all socio-economic factors affecting the underground coal mining industry, the aim should be to develop and operate “near zero emission” mines. In this context, the ideal future underground coal mine is one in which;

- All sources of seam gas emission are properly accounted for, monitored and quantified.
- There is good reconciliation between gas reservoir in place, effect of mining on all sources, including roof and floor seams, and therefore a defined reliable relationship between gas emission and production rates together with prioritisation of pre drainage targets.
- The volume of gas reporting to pre and post drainage systems is maximised with due consideration to the cost of capture compared to discharge. This includes gas emission from production coal on surface. All captured gas streams are at least flared with power generation or direct gas sales provided for when economically viable.
- Consequently, the volume of gas reporting to ventilation systems is minimised allowing less ventilation to be employed (lower fan power consumption), less development required for distribution, improved safety and reduced gas constrained production. Where appropriate VAM oxidation units are applied to some or all VAM streams with increasing use of reject heat (VAM units and IC engines) to reduce mine or local area power consumption.

7.1 Costs and Benefits of Drainage Strategies

The key issue is that fugitive emissions from coal mines may incur considerable carbon charges, if and when applicable, which would be in millions of dollars for gassy mines. Increasing the gas capture/drainage rates and destroying methane by any method (flares, engines or VAM) would substantially reduce the carbon charges.

For the purposes of this report, it is to be noted that to carry out cost benefit analysis on various gas management strategies, first the cost of CO_2-e emissions in do nothing scenario and revenue from power generation from pre or post drained gas are to be calculated. The basic or effective pay back periods can then be calculated based on revenue from power generation, if any, against capital cost and cost of CO_2-e, i.e. charge not paid due to utilisation or methane destruction. In addition, the cost and revenue for VAM thermal oxidation unit at various methane concentrations also can be calculated for different VAM methane concentration scenarios. At present, it is understood that the viable range of methane concentrations is 0.3 to 0.9% and that only heat recovery would be employed due to the high cost of generating power with reject heat. This situation could change in the future if cost of power increases.

Currently, accurate values for ongoing operational costs at mine sites, other than power generation, are uncertain. Actual effective payback periods will be longer subject to these values being assessed, and are of course also dependent on the actual CO_2-e charge.

Power generated from engines or heat recovered from engines and VAM units is an additional benefit but at significantly higher capital cost. For engines in particular, various project financing options are available including those in which a third party provides and manages the entire power generation business. In some cases this can involve selling gas for transmission in overland pipe lines. This basic analysis demonstrates the following;

- If gas is being captured at greater than 25 to 30% CH_4 then flaring is required in any event. However, this still incurs an ongoing cost due to post combustion emissions.
- Power generation using IC engines can be justified subject to revenue obtained from power sold or used on site. This would then be cost positive with a payback period dependent on how the project is financed.
- Standard VAM oxidation units may be attractive at higher methane purity but at a high capital cost and significant physical footprint on surface, subject to the ventilation rates employed and where the units are applied.
- Per m^3/s CH_4 emitted, reticulation with flaring or utilisation at high methane purity is more cost effective than VAM strategies if, and only if, the cost of the necessary pre and post drainage infrastructure is not prohibitively high, as it may be in low permeability seams and or at lower gas contents ($<<5m^3/t$). It is this issue that is fundamental in achieving optimum solutions for fugitive gas emission from underground coal mines that do not ordinarily pre drain the working seam.
- Gas contained in production coal will report to surface stockpiles and may be charged accordingly, subject to the definition of residual gas component and at what point the charge is made to the end user rather than producer. Other than somehow capturing this gas form surface stockpiles, only gas pre drainage can alter this value at a mine site.

7.1.1 Gas Management Costs

The approximate nominal costs of various systems components are shown in Table 7.1. Due to various site specific issues (ground conditions for drilling and shaft sinking in particular), current volatility of commodity prices and exchange rates, the values provided here are for an orders of magnitude analysis only. Quite obviously the cost of gas drainage is entirely dependent on the intensity employed and lead times available.

Table 7.1 Nominal Gas Management Option Costs

Group	Item	Value AU$	Unit
Pre drainage	CIS Rotary 65mm	40	per m
	CIS Directional 95mm	70	per m
	SIS MRD holes	1,000,000	per km
Surface goaf holes	Hammer to 400m	80	per m 150mm diam
	Case - black	40	per m 150mm diam
	Rotary to 400m	150	per m 300mm diam
	Case - black	75	per m 300mm diam
Reticulation	Gas retic pipe - black	100	per m 400mm diam
	Gas retic pipe - galv	155	per m 400mm diam
	Gas retic pipe - poly	165	per m 400mm diam
	Installed cost UG		multiplier = 1.5 x cost
Ventilation shafts	Blind bore	100,000	per m 5.0m diam lined
	Raise bore	925	per m per m2 cross sectional area
	Spray lining		multiplier = 0.75 x ream cost
Fans and pumps	Liquid ring pumps	250,000	per m3/s mix NTP
	Centrifugal fans (gas)	100,000	per m3/s mix NTP

The cost of introducing pre drainage to a mine can be estimated using annual block retreat rates and hole spacing together with fixed costs (system maintenance labour, pipes and fittings). Typical values for a range of block widths and hole spacing in Australian conditions are shown in Table 7.2. In additional to this will be start up capital for surface gas drainage pumps of AU$1.0M to AU$2.0M depending on flow rates and methane purity. Overall, pre drainage will add AU$ 1.0 to 2.0 per tonne to operating costs.

Similarly for surface goaf drainage, the annual cost depends on hole spacing and longwall retreat rate, Table 7.3. Of course, additional fixed costs will depend on surface reticulation and type of discharge. Overall, post drainage will add AU$ 0.5 to 1.0 per tonne to operating costs but most likely provide significantly high gas flow rates and therefore be more cost effective on a AU$ per m^3/s CH_4 basis.

Table 7.2 Cost of Introducing Pre Drainage

Seam Height	3	m
Density	1.4	t/m3
Drill cost	70	$/m
Annual fixed	2.7	AU$M (labour plus pipes)

Block width (m)	200	250	300	350	400
Annual production (	3,000,000	4,000,000	5,000,000	6,000,000	7,000,000
Annual retreat (m)	3,571	3,810	3,968	4,082	4,167
Hole spacing (m)	Annual cost of drilling AU$M				
10	5.0	6.7	8.4	10.0	11.7
20	2.5	3.4	4.2	5.0	5.9
30	1.7	2.3	2.8	3.4	3.9
40	1.3	1.7	2.1	2.5	3.0
50	1.0	1.4	1.7	2.0	2.4

Hole spacing (m)	Annual cost of drainage per tonne AU$/t				
10	2.57	2.35	2.22	2.12	2.06
20	1.73	1.53	1.38	1.28	1.23
30	1.47	1.25	1.10	1.02	0.94
40	1.33	1.10	0.96	0.87	0.81
50	1.23	1.03	0.88	0.78	0.73

Table 7.3 Cost of Introducing Goaf Drainage

Seam Height	3	m
Density	1.4	t/m3
Depth	300	m
Drill cost	120	$/m - 50% hole lined
Drill cost	36,000	$/hole
Annual fixed	1.5	AU$M (labour plus pipes)

Block width (m)	200	250	300	350	400
Annual production (	3,000,000	4,000,000	5,000,000	6,000,000	7,000,000
Annual retreat (m)	3,571	3,810	3,968	4,082	4,167
Hole spacing (m)	Annual cost of drilling AU$M				
50	2.6	2.8	2.9	3.0	3.0
100	1.3	1.4	1.5	1.5	1.5
150	0.9	1.0	1.0	1.0	1.0
200	0.7	0.7	0.8	0.8	0.8
250	0.6	0.6	0.6	0.6	0.6

Hole spacing (m)	Annual cost of drainage per tonne AU$/t				
10	1.37	1.08	0.88	0.75	0.64
20	0.93	0.73	0.60	0.50	0.43
30	0.80	0.63	0.50	0.42	0.36
40	0.73	0.55	0.46	0.38	0.33
50	0.70	0.53	0.42	0.35	0.30

The most significant issue for Australian mines is that the cost of pre and post drainage may, in a number of cases, be less than the potential CO_2-e charges (if and when applicable) and should therefore be introduced or intensified. In other cases, the cost of these strategies is less than that of potential CO_2-e charges and it will be cost effective to continue direct discharge. Here in lies the problem. The lowest capital cost method of destroying methane is with flares but flares require a pre or post drained gas stream at greater than 30% purity. The decision to pre and or post drain must be driven by the consequential cost of VAM emissions and what fraction of this can be captured.

7.2 Basic Mine Analysis and Screening

There are of course numerous methods of presenting analysis of GHG emission from underground coal mines. The method provided in this section is for basic analysis and screening using mine average values. From the analysis of costs provided above and the rates of CO_2-e emission (tCO_2-e per t ROM) for Australian coal mines, decision making needs to be undertaken on a site specific analysis of CO_2-e charged per t ROM.

The main issue is to quantity the distribution of gas streams in terms of CO_2-e mass flow rate (Table 7.4 and Figure 7.1) from basic input data (Figure 7.2) and analysis (Figure 7.3). The worked example is for a gassy mine employing VAM, engines and flares in ventilation, pre and post drainage systems.

Table 7.4 Analysis of Coal Mine Gas Streams and Utilisation

SOURCES	RAW EMISSION
VAM Sources	**VAM Streams**
• Longwall(s) • Gate development • Mains development • Sealed areas	• Main shafts • Bleeder shafts
Production Sources	**Production stream**
• Development coal • Longwall coal	• Main conveyor
Gas Drainage Sources	**Drainage streams**
• CIS pre drainage • SIS pre drainage • Surface goaf drainage • UG goaf drainage	• UG reticulation • Surface reticulation • Surface points

UTILISATION	FINAL EMISSION
All Streams	**ANALYSIS & DECISIONS**
NONE Minimise	tCO_2-e/yr or AU$/yr Cost of do nothing option? Can utilisation be increased? Benefit of reducing ventilation power?
VAM **0.3 to 0.9% CH_4** Optimise	Optimise fraction utilised Consider pre drain gas, split ventilation systems and heat recovery.
FLARES **>25% CH_4** Maximise	Can pre or post drainage reduce VAM and production streams ?
ENGINES **>30% CH_4** Maximise	Consider heat recovery, transmission and finance schemes.

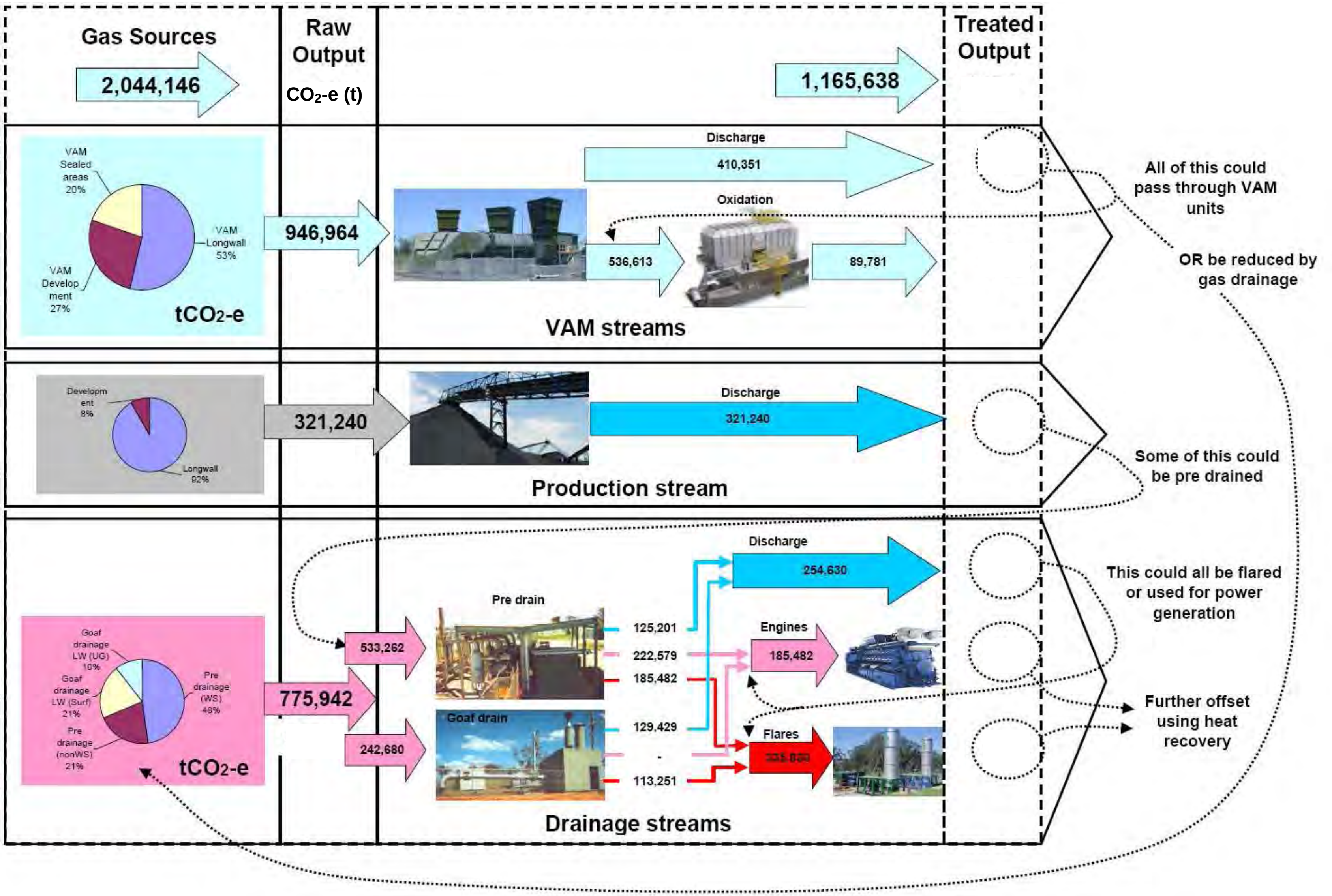


Figure 7.1 Basic Mine Analysis Flow Schematic

The main objective of this approach is to identify and prioritise opportunities for improvement or simply to quantify potential CO_2-e emissions. In this example, the main issue is the untreated emission from VAM, production stream and direct discharge from gas drainage systems. The obvious solutions would be increasing flare capacity, improving longwall capture efficiency and considering the benefit of increased pre drainage.

Input Data	**Unit**	**Value**
Dev height	m	3
Dev width	m	5.2
Longwall prod	tpy	5,000,000
Development	m/y	26,000
Development	tpy	567,840
Seam density	t/m3	1.4
Gas content compositon	%CH4	80
Gas content virgin	m3/t	6
Gas content LW production	m3/t	5
Gas content Dev production	m3/t	4

VAM Streams	Vent m3/s	Vent CH4%	Vent CO2%	CH4 l/s	CO2 l/s	VAM oxid m3/s (mix)	CH4 dest l/s
Longwall return 1	80	1.4	0.1	1120	80	0	-
Longwall return 2	0	0.0	0.0	0	0	0	-
Longwall total	80	1.40	0.10	1120	80	0	-
Gate 1	60	0.4	0.2	240	120	0	-
Gate 2	50	0.4	0.1	175	50	0	-
Gate 3	0	0.0	0.0	0	0	0	-
Mains face	40	0.3	0.1	120	40	0	-
Development total	150	0.36	0.14	535	210	0	-
Sealed areas				400	100	0	-
Misc vent & leakage	70						
Total shaft applied VAM		0.69	0.13			170	1,165
Mine total VAM	300			2,055	390	170	1,165

Gas Drainage Streams	Mixture Total l/s	Mixture CH4%	Mixture CO2%	CH4 l/s	CO2 l/s	Flares m3/s (mix)	Engines m3/s (mix)	CH4 dest l/s
Pre drainage (WS)	1,000	80.0	20.0	800	200	0.3	0.5	600
Pre drainage (nonWS)	500	70.0	15.0	350	75	0.4	0.0	280
Goaf drainage LW (Surf)	500	70.0	15.0	350	75	0.4	0.0	245
Goaf drainage LW (UG)	250	70.0	15.0	175	38	0.0	0.0	-
Pre drainage total	1,500			1,150	275	1	1	880
Goaf drainage total	750			525	113	0	-	245
Gas drainage totals	2,250			1,675	388	1.00	0.50	1125.00

Production Coal Streams	Prod Coal Gas l/s	Mixture CH4%	Mixture CO2%	CH4 l/s	CO2 l/s	CH4 dest l/s
Longwall	792.74	80	20	634	159	
Development	72.02	80	20	58	14	
Production coal total	865			692	173	
Overall mine total				4,422	950	2,290

Figure 7.2 Basic Mine Input Data

Analysis

Longwall		CH4	CO2
Longwall make vent + goaf	l/s	1,645	193
Capture goaf drainage	l/s	525	113
Capture efficiency	%	32	58
Specific gas emission	m3/t	10	1

VAM Streams	Source CH4 l/s	Source CO2 l/s		Destroy CH4 l/s	Discharge CH4 l/s	Discharge CO2 l/s	CO2-e Tot Pot t/y	CO2-e CH4 des t/y	CO2-e CH4 dis t/y	CO2-e CO2 dis t/y	CO2-e Emitted t/y
VAM Longwall	1,120	80		-	1,120	80	508,328	-	503,632	4,695	508,328
VAM Development	535	210		-	535	210	252,899	-	240,574	12,325	252,899
VAM Sealed areas	400	100		-	400	100	185,738	-	179,869	5,869	185,738
Total VAM streams	2,055	390		1,165	891	390	946,964	76,811	400,433	22,889	500,132

Production Streams	Source CH4 l/s	Source CO2 l/s			Discharge CH4 l/s	Discharge CO2 l/s	CO2-e Tot Pot t/y		CO2-e CH4 dis t/y	CO2-e CO2 dis t/y	CO2-e Emitted t/y
Longwall	634	159			634	159	294,485		285,180	9,305	294,485
Development	58	14			58	14	26,755				
Production coal total	692	173			692	173	321,240		311,090	10,150	321,240

Gas Drainage Streams	Source CH4 l/s	Source CO2 l/s	Flare CH4 l/s	Pwr Gen CH4 l/s	Discharge CH4 l/s	Discharge CO2 l/s	CO2-e Tot Pot t/y	CO2-e CH4 des t/y	CO2-e CH4 dis t/y	CO2-e CO2 dis t/y	CO2-e Emitted t/y
Pre drainage (WS)	800	200	200	400	200	200	371,475	39,576	89,934	11,738	141,248
Pre drainage (nonWS)	350	75	280	-	70	75	161,787	18,469	31,477	4,402	54,347
Goaf drainage LW (Surf)	350	75	245	-	105	75	161,787	16,160	47,216	4,402	67,777
Goaf drainage LW (UG)	175	38	-	-	175	38	80,893	-	78,693	2,201	80,893
Pre drainage total	1,150	275	480	400	270	275	533,262	58,045	121,411	16,139	195,596
Goaf drainage total	525	113	245	-	280	113	242,680	16,160	125,908	6,602	148,671
Gas drainage totals	1,675	388	725	400	550	388	775,942	74,205	247,320	22,742	344,266

	Source CO2 l/s	Source CH4 l/s		Destroy CH4 l/s	Discharge CH4 l/s	Discharge CO2 l/s	CO2-e Tot Pot t/y		CO2-e CH4 dis t/y	CO2-e CO2 dis t/y	CO2-e Emitted t/y
Total mine	950	4,422		2,290	2,132	950	2,044,146	151,016	958,842	55,781	1,165,638

Figure 7.3 Basic Mine Analysis

7.2.1 Decisions Concerning VAM Oxidation Units

For decision making, the issue is to minimise surface emissions at optimum cost. Clearly the problem with currently available VAM technology is the very high capital cost and, currently, the prohibitively high cost of power generation using reject heat.

Using the worked example provided below, the effect of applying VAM units to the longwall return alone, then using gas drainage to remove sealed area emissions is summarised in Table 7.5.

Table 7.5 Decision Making for VAM

Current siutation	Ventilation m3/s	CH4 l/s	CO2 l/s	CO2-e t/y	
VAM Longwall	80	1,120	80	508,328	
VAM Development	150	535	210	252,899	
VAM Sealed areas		400	100	185,738	
		2,055	390		
		CH4 %	CO2 %		
Total mine	300	0.69	0.13		

Longwall only VAM	Ventilation m3/s	CH4 l/s	CO2 l/s	CO2-e t/y	
VAM Development	150	535	210	252,899	
VAM Sealed areas		400	100	185,738	
		935	310		
		CH4 %	CO2 %		
Total mine	220	0.43	0.14	Annual saving	= 508,328 t CO_2-e

Longwall only VAM Drain sealed areas	Ventilation m3/s	CH4 l/s	CO2 l/s	CO2-e t/y	
VAM Development	150	535	210	252,899	
		535	210		
		CH4 %	CO2 %		
Total mine	220	0.24	0.10	Annual saving	= 694,066 t CO_2-e

The annual "do nothing" CO_2-e emissions would be 946,965 tonnes/year, with substantial cost, if carbon charges are introduced. If a single raise and 5 VAM units were to be installed in the longwall tailgate or in main returns so that a single installation could be used for two or more blocks then the annual saving in emissions would be 508,328 t CO_2-e, with substantial cost savings. Although clearly not a trivial task, there would therefore be substantial annual savings if VAM were to be applied to the longwall return alone. The alternative would be to apply about 18 units to the main ventilation shaft at higher capital cost. The strategy of splitting ventilation systems is therefore only likely to be attractive in mines where longwall emissions are very much higher than those from development panels or sealed areas.

In countries such as China, a significant part of VAM projects is the benefit of using heat rejection for winter intake air and bathhouse heating. In some countries it is also used by local communities. With the likely future increase in cost of power in Australia, the use of reject heat form VAM units and IC engines will become increasingly viable.

7.3 Mine Classification and Strategies

With consideration to the range of gas reservoir sizes present in Australian mines, a basic spreadsheet model has been developed to identify the orders of magnitude technical design values involved for pre and post drainage requirements together with VAM strategies, Table 7.6 and Figure 7.4. This has been established using a number of identified general assumptions and design criteria that can be changed to fit individual mines sites. The main relationships to consider are those associated with gas reservoir size within the zone of influence, gas content of the working section and annual production rate.

The results of this analysis for the distribution (m^3/m^2) of gas emission with increasing gas reservoir size, Figure 7.5, and rates (m^3/s) of gas emission, Figures 7.6 and 7.7, suggest the following classification of mines can be used for broad decision making (GRS and WS gas content values are generic and for guidance only, goaf drainage capture efficiency set at 50%);

1. Very low gas emission mines (GRS < $30m^3/m^2$ and WS < $3m^3/t$) - at any production rate
   - Consider VAM if part or all returns >0.3%CH_4
   - Consider sealed area drainage if proven to be significant
   - Conventional pre drainage not feasible
   - Possibly do nothing e.g. $150m^3/s$ at 0.2% CH_4 = 300l/s.
2. Low gas emission mines (30< GRS < $50m^3/m^2$ and 3< WS < $5m^3/t$)
   - As for category 1 with need for VAM increasing at higher production rates
   - Consider goaf drainage of active and sealed longwalls
   - Viability of stimulated pre drainage depends on techniques available, balance against reduced VAM load and cost of gas reporting to stockpiles.
3. Medium gas emission mines (50< GRS < $80m^3/m^2$ and WS < $7m^3/t$ and < outburst limit m^3/t)
   - VAM required with or without split ventilation systems
   - Goaf and sealed area drainage required for compliance in single tailgate returns, particularly at higher production rates.
   - Consider pre drainage characteristics and operational/ economic viability to off load development phase VAM and gas reporting to surface stockpiles.
   - Consider MRD pre drainage of large non working seam gas sources.
4. High gas emission mines (80< GRS < $110m^3/m^2$ and WS > outburst limit)
   - VAM required with or without split ventilation systems
   - Goaf and sealed area drainage required using goaf area management to maximise goaf drainage efficiency and alternative goaf drainage hole geometry for close face capture.
   - Pre drainage required in any event, consider intensity and lead times to offload development phase VAM and reduce gas reporting to surface stockpiles.

  - Subject to goaf drainage efficiency, alternatives such as pre drainage of non working sections may be required at higher production rates.

5. Very high gas emission mines (110< GRS m^3/m^2 and WS < outburst limit)
    - As for category 4 with increased need for pre drainage of gas reservoir in non working seam strata.

In the analysis provided here alternative strategies include;

- Introducing pre drainage when not otherwise required (appropriate in higher permeability seams).
- Increasing the time available and or intensity of pre drainage.
- Using reservoir stimulation techniques such as hydrofracture or nitrogen injection
- Pre draining non working seams or strata
- Introducing active and sealed goaf drainage when not otherwise required
- Increasing the intensity of goaf drainage, including strategies to improve capture close to face zone

Table 7.6 Basic Gas Emission Model and Range of Australian Design Values

WS height	3	m
Ws density	1.4	t/m3
Production rate	3,000,000	tpy
Weeks per year	46	
	65,217	tpw
Days per week	6	
	10,870	tpd
	0.03	m2/s
Outburst threshold	7	m3/t
Pre drained content	4	m3/t
Goaf drainage limit	50	%
Tailgate ventilation	60	m3/s
Tailgate limit	1.0	% CH4 average
Bleeder vent	50	m3/s
Bleeder limit	1.5	% CH4 average
Use bleeder	No	

Item	Unit															Note
Total GR in zone of influence, including WS	m3/m2	10	20	30	40	50	60	70	80	90	100	125	150	175	200	
Average methane composition	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	
Gas content working section (virgin)	m3/t	1.0	1.7	2.4	3.1	3.7	4.4	5.1	5.8	6.5	7.2	8.9	10.6	12.3	14.0	A
GR in non working section	m3/m2	6	13	20	27	34	41	49	56	63	70	88	106	123	141	B
GR in working section	m3/m2	4	7	10	13	16	19	21	24	27	30	37	44	52	59	B
Percent non working GR released	%	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	C
Specific gas emission - non working GR	m3/t	1.2	2.8	4.3	5.8	7.4	8.9	10.4	11.9	13.5	15.0	18.8	22.6	26.4	30.3	D
WS pre drainage requried (outbursts)	Y/N	N	N	N	N	N	N	N	N	N	Y	Y	Y	Y	Y	E
WS Gas content pre drained	m3/t	1.0	1.7	2.4	3.1	3.7	4.4	5.1	5.8	6.5	4.0	4.0	4.0	4.0	4.0	F
Nominal pre drainage rate (subject to time taken)	m3/s	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.6	0.8	1.0	1.3	
GR in working section pre drained	m3/m2	4	7	10	13	16	19	21	24	27	17	17	17	17	17	
Percent working seam release UG	%	0	2	3	5	6	8	9	11	13	7	7	7	7	7	G
Gas emission production coal (surface)	m3/s	0.13	0.21	0.29	0.37	0.44	0.51	0.58	0.65	0.71	0.47	0.47	0.47	0.47	0.47	
Gas emission production coal (underground)	m3/s	0.00	0.00	0.01	0.02	0.03	0.04	0.06	0.08	0.10	0.03	0.03	0.03	0.03	0.03	
Gas emission non working seam	m3/s	0.2	0.3	0.5	0.7	0.9	1.1	1.3	1.5	1.7	1.9	2.4	2.8	3.3	3.8	
Potential LW gas emission underground	m3/s	0.16	0.35	0.55	0.75	1.0	1.2	1.4	1.6	1.8	1.9	2.4	2.9	3.4	3.8	H
Operational Longwall																
Design ventilation capacity CH4	m3/s	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	I
Goaf drainage required	Y/N	N	N	N	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	J
Goaf drainage required (methane)	m3/s	0.0	0.0	0.0	0.2	0.4	0.6	0.8	1.0	1.2	1.3	1.8	2.3	2.8	3.2	K
Goaf drainage effectiveness required	%	0	0	0	20	37	48	56	62	67	69	75	79	82	84	
Goaf drainage employed	m3/s	0.00	0.00	0.00	0.15	0.35	0.56	0.38	0.49	0.60	0.66	0.90	1.14	1.38	1.62	
Other strategy required to support goaf drain	Y/N	N	N	N	N	N	N	Y	Y	Y	Y	Y	Y	Y	Y	L
Other strategy requriements (methane)	m3/s	0	0	0	0	0	0	0.3849	0.4908	0.59807	0.66	0.90	1.14	1.38	1.62	

A. Gas content of working section assumed to increase with gas reservoir size for this generic model.
B. Fraction of total gas reservoir in working and no working sections dependent on extraction height and gas content.
C. Percent of non working section released is the total fraction of gas released from the non working section gas reservoir during production as obtained from longwall gas emission models.
D. The specific gas emission rate is that from the non working section gas reservoir related to working section production rate.
E. Pre drainage required if working section gas contents exceed outburst thresholds otherwise ventilation employed.
F. If pre drained then gas content reduced to specified level with balance reporting to pre drainage system.
G. Percent of working section gas content at time of mining released underground and reporting as VAM as obtained from coal desorption rates.
H. Potential longwall gas emission is the sum of that from production and non working section gas reservoir.
I. Ventilation capacity is that specified by ventilation rate (m3/s) and average methane concentration (% CH_4). This can be single tailgate or tailgate plus bleeder.
J. Goaf drainage required when longwall emission rate exceeds ventilation capacity.
K. Goaf drainage required is the difference between longwall emission and ventilation capacity with the effectiveness as a percent capture being the ratio of goaf drainage capture rate required (m3/s) divided by potential longwall emission. The actual goaf drainage rate employed is that specified as a percent of total emission, for example 50%.
L. Other strategies are required when goaf drainage at specified effectiveness cannot manage longwall gas emission net of that reporting to ventilation systems. These alternative strategies could be;
   - Increase ventilation rates with consequential increase in VAM load.
   - Improved goaf drainage effectiveness using alternative hole configurations and nitrogen injection, section 6.2 below.
   - Pre drainage of non working section gas sources such as roof or floor seams.

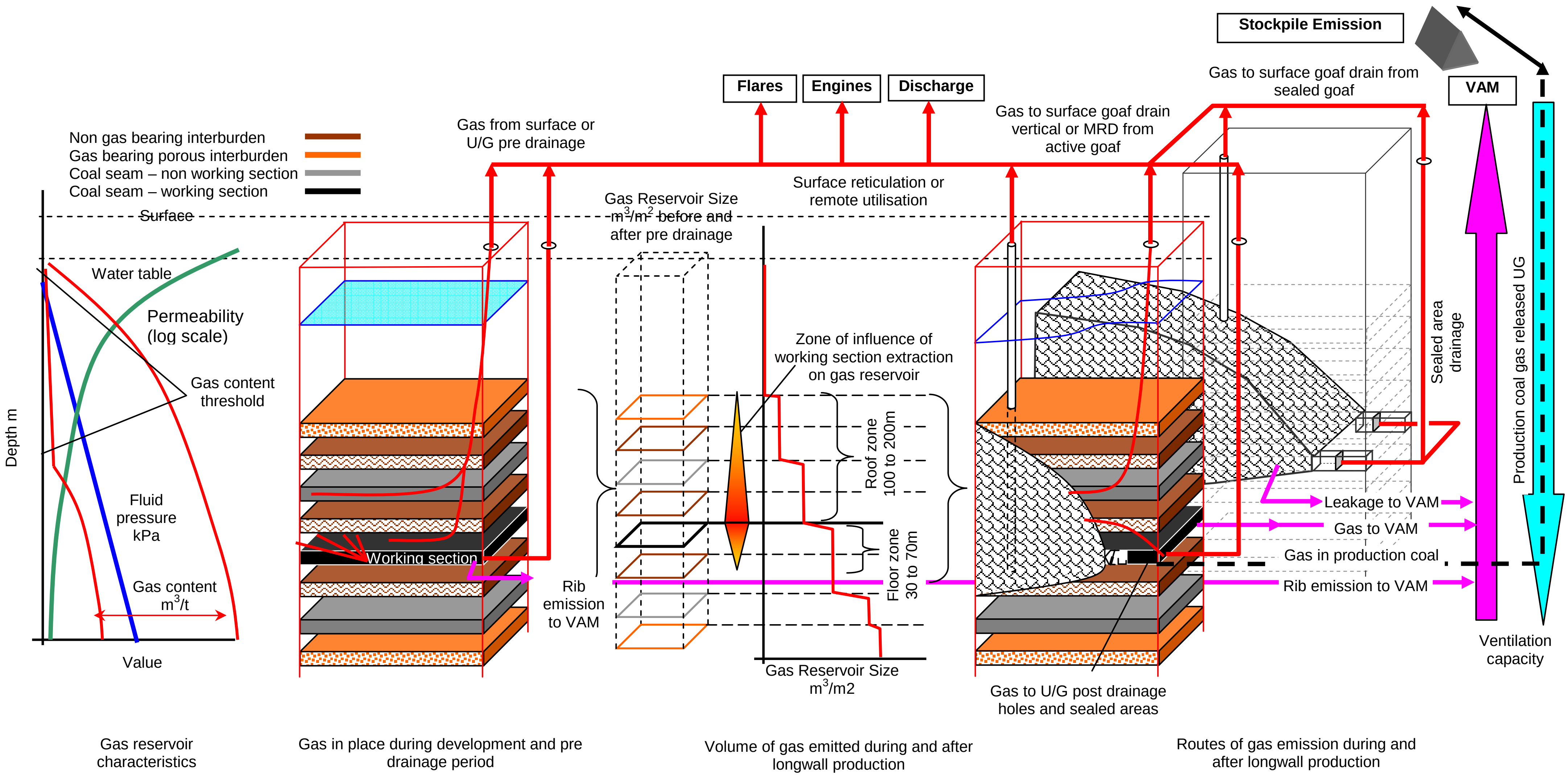


Figure 7.4 Longwall Gas Emission Models and Gas Emission Control

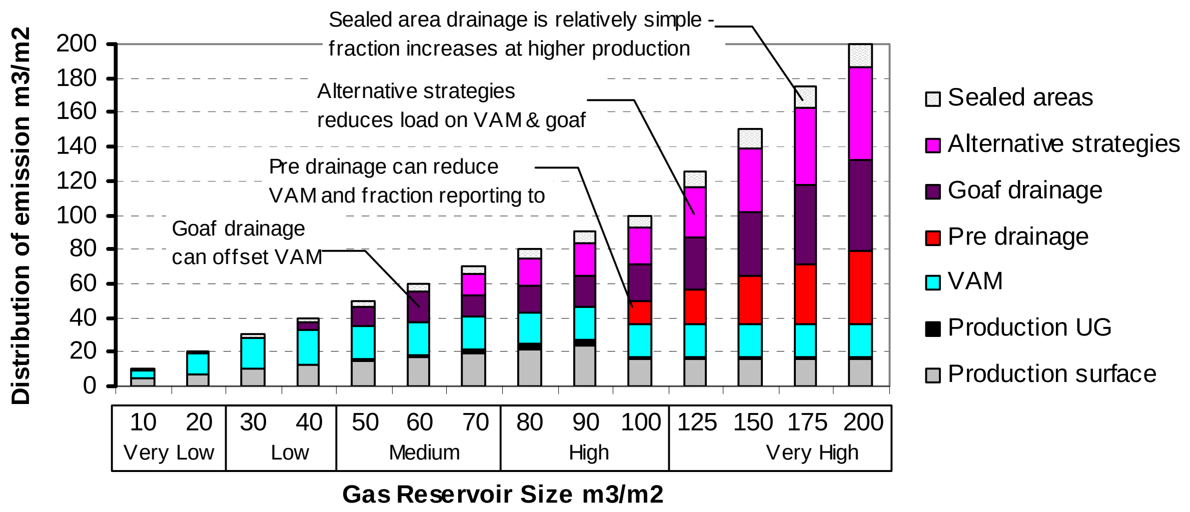


Figure 7.5 Distribution of Longwall Gas Reservoir Emissions m^3/m^2

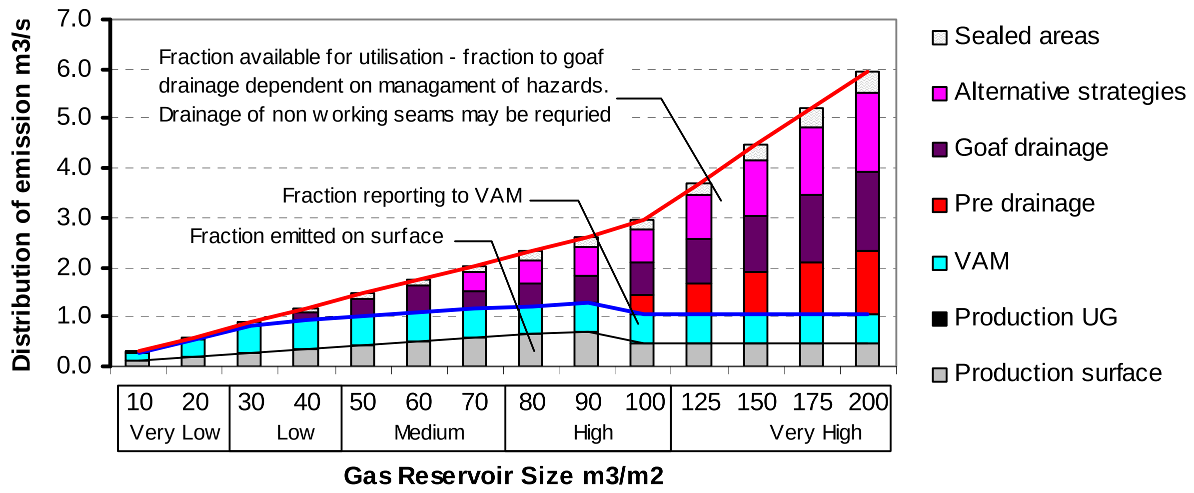


Figure 7.6 Rate of Longwall Gas Reservoir Emissions m^3/s at 3Mtpy

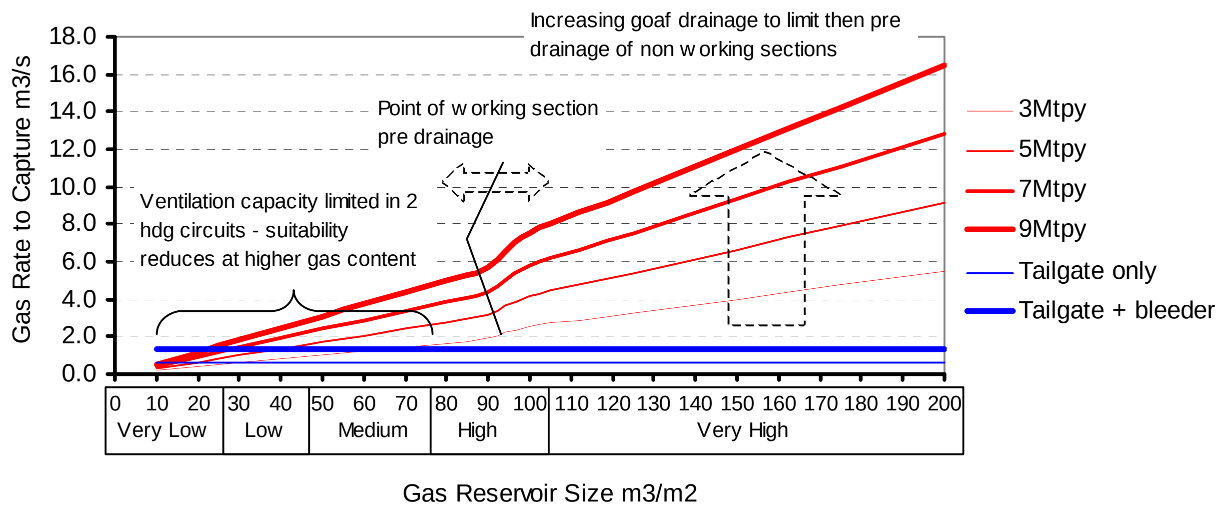


Figure 7.7 Rate of Longwall Gas Reservoir Emissions m^3/s at Various Production Rates

8. CONCLUSIONS AND RECOMMENDATIONS

8.1 Conclusions

The main conclusions from the review are as follows:

1. The National Greenhouse Accounts Factors and determination methods specified for use in coal mines provide options for assessment (method 1 using tonnage factors and method 4 using measurement). It is likely that the factor method for "gassy" mines may overestimate emissions from some mines not prone to outbursts and underestimate emissions from "non gassy" mines. The issue here is definition of gassiness based on main return ventilation concentrations being greater or less than 0.1% CH_4 without consideration to ventilation rates. A methane emission rate in m^3/s CH_4 is more appropriate as this is also used to determine a mines ability to access the transitional assistance fund.

2. The main issues yet to be adequately addressed in National Greenhouse and Energy reporting are:

   - Direct determination of emissions from stockpiles. Values provided may significantly underestimate that arising in gassier coal mines with short underground coal residence times. This does however depend on long term residual gas contents and the rate of desorption on surface.
   - Acceptably accurate measurement of ventilation flow rates in hazardous zones to quantify VAM, including surface fans. This is also identified as a significant issue for coal mine operational effectiveness and management of safety in gassy and or spontaneous combustion prone conditions.

3. There is a need to develop methodologies to quantify CO_2-e emissions from coal in stockpiles and surface transport systems. This will be fundamentally important for decision making in mines that otherwise do not have to pre drain for compliance with coal mine safety regulations.

4. Fugitive gas emission from coal mining activities are increasing with time and will continue to do so as more underground projects become available at increasing production rates and increasing depths.

5. The gas reservoir characteristics impacting on Australian coal mines lead to a wide range of actual emission rates but with an upper boundary to potential specific emissions (m^3/t) determined by known limits of the adsorption capacity of coal seams and gas content of interburden.

6. Based on observed gas emission rates and desorption characteristics of coal, it invariably the case that the majority of seam gas is emitted during the active production life of development and longwall panels. In particular, peak emission rates occur in relatively close proximity to the active pillar of development panels or longwall face line indicating where the improved capture strategies should be focused.

7. With consideration to the volume of gas that can be managed by high ventilation rates and the volume of gas transported to surface in production coal, the main change that

will be brought about by the need to reduce fugitive emission may be the reduction of working seam gas contents by pre drainage in mines that otherwise would not require it.

8. The review of international gas management practice identifies a number of gas drainage strategies not currently employed in Australia, such as cross measure roof holes adjacent to the face and superjacent drilling galleries. Historically, these methods have not commonly been used due to economic factors, lack of need with respect to ventilation or alternative strategies and because Australian mines are relatively shallow with many having surface access making the underground approach unnecessary.

9. There are some examples of cross measure holes being employed, for example floor seam relief holes at Central Colliery and in Bulli seam mines, but these are only been used in circumstances where very gas emission rates would otherwise occur during periodic floor breaks. They have not been used to improve capture efficiency when surface goaf drainage combined with high ventilation rates is adequate for compliance. In countries where increased capture is beneficial for power generation and or increasing credit for GHG emission mitigation these techniques are employed, for example United Nations CMM projects in Chinese coal mines.

10. The technology is already available for destruction of methane at high and low purity and is employed world wide. High purity (>30%CH_4) drainage streams are most easily used for power generation using internal combustion gas engines or, when available at high concentrations, delivered to combined trans continental gas reticulation systems.

11. Oxidation VAM units are understood to work best with steady methane concentration and with the fastest rate of return when at limiting concentrations (circa 0.9%). Consideration should therefore be given to supply of high purity pre drainage gas streams, when available, to these units as a means of stabilising methane purity in the exhaust air supply.

12. The technology for VAM destruction is available although still undergoing a period of further research and development together with assessment of longer term project economics, maintenance costs in particular.

    In this respect, decision making is dependent on confidence in longer term planning values for both CO_2-e charges and cost of electrical power in the current and post 2012 (Kyoto protocol review date) economic climates.

13. In most coal mines it is the continuous and accurate measurement of air velocity for calculation of air quantity in exhaust shafts that is problematic for the purposes of compliance with determination requirements. Monitoring of gas concentrations, pressure and temperature is or should be relatively straightforward using available instrumentation.

8.2 Recommendations

The main recommendations along with comments on future research requirements and opportunities for improvements in gas capture and mitigation strategies are outlined below:

1. The gas reservoir characteristics of working seams are relatively well described by current exploration data in the public domain. However, data for non working sections and porous interburden are less well documented and often lacking in the exploration programmes of current and future underground coal mine projects. This can be improved by increasing the intensity of gas content testing in these non working sections, including strata with up to 85% ash content.

   It is also likely that some current estimates of gas available in the zone of influence of mining underestimate that actually present and, for this reason, apparently large peak emission factors have to be applied for design purposes. Collation and interpretation of mine gas emission data should not only be reconciled with production phases, as they have been in the past, but also with the total gas reservoir in place.

2. It is identified that the residual gas contents of coal in it's virgin or native state may be significantly higher that that indicated by isotherms obtained from crushed samples. If this were the case in some Australian coal seams, as it in other locations worldwide, then it could influence decision making with respect to the need for pre drainage in working and non working seams.

3. There is an identified need to improve understanding of gas reservoir response to underground coal mining in Australian conditions. In particular, actual post mining fluid pressure and residual gas content profiles in the roof and floor of working sections of active and sealed goaves. This would allow the variety of models available to be applied with increased confidence as production rates, blocks sizes and depth of mining are taken beyond historical norms. These are also essential to assess alternative pre drainage strategies targeted at non working seam gas sources which are identified as the major contributors to mine gas emissions other than during the development phase.

4. As underground coal mines proceed to increase production rates in larger blocks and at increasing depths it is inevitable that gas emission rates and gas emission volumes will increase. In many coal seams, the gas emissions will increase well beyond the management capacity of ventilation and current pre and post drainage systems. Improvements are therefore required in these aspects for both operational and GHG emission mitigation reasons.

5. An opportunity for improvement in Australian mines is to consider gas drainage strategies targeting close face gas emissions in order to reduce the fraction of total gas emission reporting to ventilation. Recognising that this gas would otherwise report to the ventilation system, the preferred option will depend on relative costs and site specific factors such as also using roof seam pre drainage holes for goaf drainage.

   These strategies will also be required in gassy mines at higher production rates when gas emission to ventilation, net of that to conventional goaf drainage systems, exceeds the dilution capacity of practicable ventilation rates.

6. Many of the techniques employed in the US for pre and post drainage would be applicable in Australian conditions. In particular, the use of hydro-fracturing with

conventional underground drill rigs has had limited application, for example success at Dartbrook, and would benefit other hard to drain seams. In addition, the use of long directional goaf drainage holes would be a particular advantage in targeting close face emissions. The main issue with here is to support introduction of pilot schemes in Australian mines so that the technique becomes accepted. As with other alternative strategies it is simply the lack of historical need that has resulted in hydro-fracturing not being widely employed.

7. The main opportunities for improving pre drainage in Australia are as follows;
   - Improve understanding of post mining gas desorption from adjacent seams and strata by direct measurement, in particular actual rather than theoretical residual gas contents. Use this information to improve pre drainage hole pattern design and identification of non working seam targets.
   - Use MRD from surface or underground directional holes to pre drain working and non working seams well ahead (3 to 5 years) of production. In mines where surface access is limited, this may require additional development which, in some respect, is similar to the suprjacent approach taken in some mining locations elsewhere.
   - Consider the use of hydrofracture for improving connectivity between multiple thin or closely spaced non working seams in addition to increasing permeability of hard to drain seams. This technology is readily available and does not require significant research and development to apply.
   - Consider use of increased gas drainage effectiveness provided by nitrogen injection but balance this against the actual residual gas content on surface and how CO_2-e charges will be applied for gas emission from stockpiles and surface transport.
   - Overall, the driver for increased pre drainage effectiveness will be financial or operational justification through reduced CO2- charges, reduced VAM costs, increased production rates, reduced downtime and, with rising energy prices, the benefits of power generation. However, in many lower gas emission mines it may well be the case that a VAM only solution is appropriate for short or long term plans.

8. The main opportunities for improving post drainage are as follows;
   - Increase capture in close proximity to the face line using directional MRD roof holes from surface or underground, possibly serving both a pre and post drainage function.
   - Apply goaf drainage to sealed areas with automatic control loops for gas concentration and barometric pressure. Consider the need for positive pressure balance chambers with or without nitrogen subject to sponcom propensity.
   - Use proactive inertisation as a control for spontaneous combustion and explosive gas mixtures in order to maximise goaf drainage capture efficiencies. Nitrogen supply systems could also be used for pre drainage improvements if necessary.

9. The application of VAM units to parts of a mine ventilation circuit would be advantageous in many gassy operations where longwall gas emission is largest contributor to fugitive emissions. The main objective would be to maximise the methane concentration (0.9%) at all times in the least quantity of air with the balance of mine exhaust reporting to surface at less than 0.2 to 0.3% CH4.

10. There is a need for further work on the continuous measurement of air quantity in coal mines in general both for GHG emission and operational reasons. This should focus on

applying standard techniques (pitot tubes and differential pressures) together with introduction of electronic devices currently available to the metalliferous industry.

11. There are a number of strategies that should be investigated further, in particular the use of automated reverse flushing to clean pressure differential devices and measurement of intake airway velocities in order to determine return airway quantities.

12. There is a need to improve methodologies and technologies to estimate or directly measure emissions from coal stockpiles on the surface. The current post-mining estimation factors provided may significantly underestimate emissions that are arising in gassier coal mines with short underground residence times. The post-mining emissions also depend on long term residual gas contents and the rate of desorption on surface.

Ideal GHG friendly mine scenario

With consideration to all socio-economic factors affecting the underground coal mining industry, the aim should be to develop and operate “near zero emission” mines. In this context, the ideal future GHG friendly underground coal mine is one in which;

- All sources of seam gas emission are properly accounted for, monitored and quantified.

- There is good reconciliation between gas reservoir in place, effect of mining on all sources, including roof and floor seams, and therefore a defined reliable relationship between gas emission and production rates together with prioritisation of pre drainage targets.

- The volume of gas captured by pre and post drainage systems is maximised with due consideration to the cost of capture compared to discharge. This includes gas emission from production coal on surface.

- All captured gas streams are at least flared with power generation or direct gas sales provided for when economically viable.

- Consequently, the volume of gas reporting to ventilation systems is minimised allowing less ventilation to be employed (lower fan power consumption), less development required for distribution, improved safety and reduced gas constrained production. Where appropriate VAM oxidation units are applied to some or all VAM streams with increasing use of reject heat (VAM units and IC engines) to reduce mine or local area power consumption.

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APPENDIX A – GHG EMISSION MITIGATION IN MINES WITH VERY HIGH GAS EMISSIONS (CASE STUDY A)

Case study A is concerned with a very high gas emission mine (110< GRS m^3/m^2 and WS > outburst limit) located in the Bowen Basin.

A.1 Gas Reservoir Conditions

A series of longwall blocks is located in a 2.8m high seam with gas contents ranging 8 to $14m^3/t$ and with a composition of about 98% methane. Depth of cover is 250m to 500m with surface access generally unconstrained by geographical or other features, Figure A.1.

The outburst limit for this seam is $7.5m^3/t$ and the self imposed limit for frictional ignition control is $5.75m^3/t$. There is a single floor seam and eight roof seams containing 10 to 15m of coal within the nominal cave zone (50m below to 200m above the working section). All these seams are known to have a similar gas content – depth relationship as the working section.

Longwall blocks are 300m wide and up to 3.6km in length with a planned production rate of 110,000tpw. These blocks are not overlain by old workings but there are a three other mines in the locality from which gas emission data has been obtained. Prior to mining being undertaken, this data indicated a specific gas emission rate of 10 to $15m^3/t$.

With consideration to longwall block geometry, it was the high potential gas emission values that led the mine to develop three heading gate roads from the outset in order to provide a high volumetric capacity ventilation system for dilution together with provision for a tailgate intake airway. This is currently the only mine in Australia to employ three heading gate roads.

A.2 Gas Emission Values

Further assessment of the gas content and thickness of roof seams indicated that the gas reservoir ranged up to 120 m^3/m^2 or 15 to $30m^3/t$ SGE gas emitted per tonne mined. At plan production rates this would equate to gas emission rates of 3,500 - 7,000 l/s $CH_{4,}$ generally increasing with depth. However, previous studies at an adjacent mine demonstrated that there is also a significant fraction of the gas reservoir present as free gas in porous interburden for which gas content data is unknown and accurate quantification is difficult. Gas emission from this source is superimposed on that contained in coal seams.

Although development rib emission is managed by reducing gas contents to below the frictional ignition limit prior to mining, longwall emissions (ventilation plus goaf drainage) reached 2,000l/s in longwall 1, 4,500l/s in longwall 2 and 6,000l/s in longwalls 3 and 4, Figure A.2. This equates to an increase in specific gas emission of 15 to $32m^3/t$, Figure A.3.

Although these values were within the predicted life of mine emission envelope, they were occurring at shallower depths indicating that deeper block emission rates would exceed feasibility stage predictions, possibly reaching 9,500 l/s in the longer term. A further significant feature of longwall gas emission in this mine is that day maximum emission rates are some 1.5 to 2.0 times the day average i.e. requiring a gas management system that can deal with short term peaks.

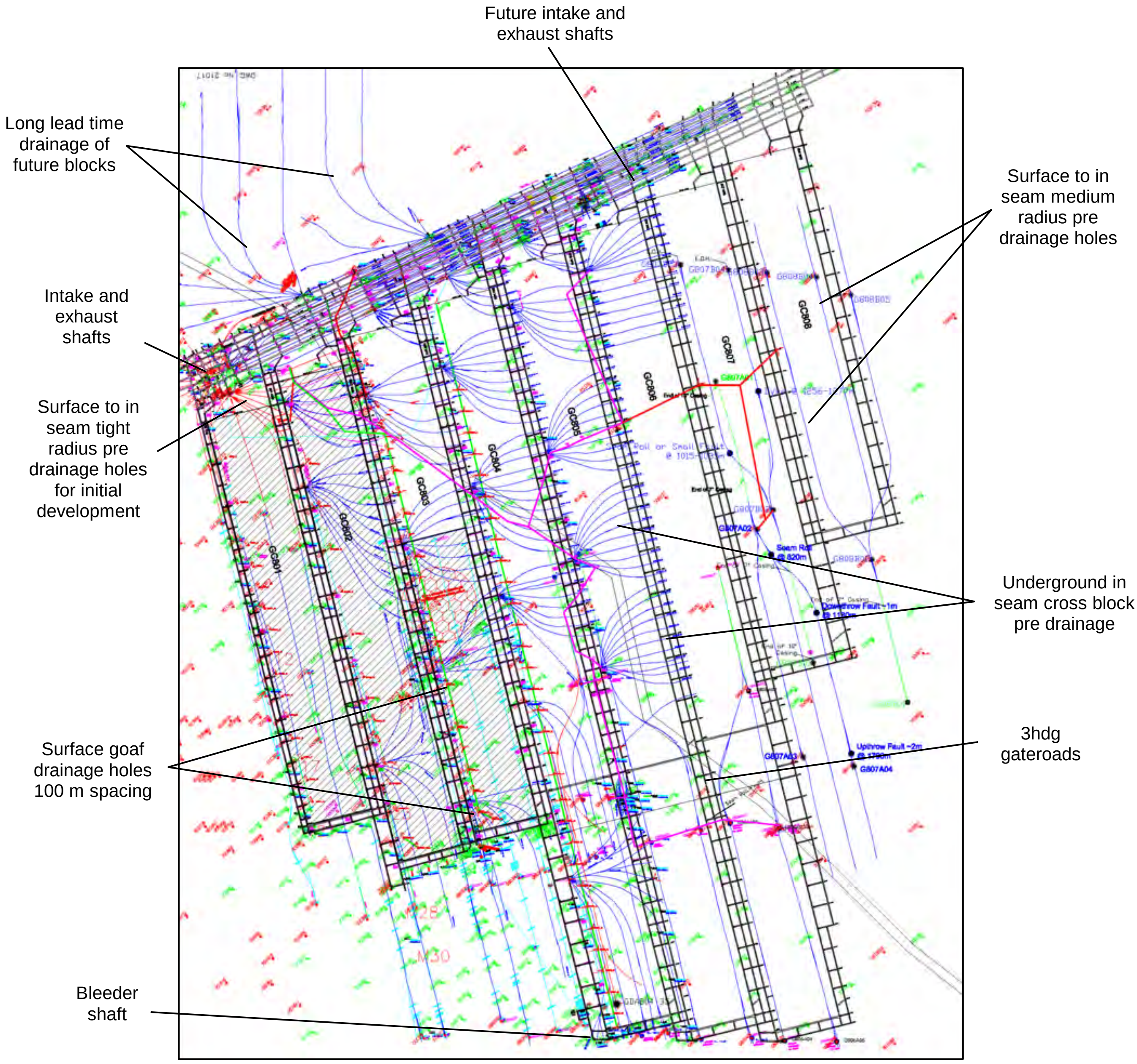


Figure A.1 Mine Plan and Gas Management Systems

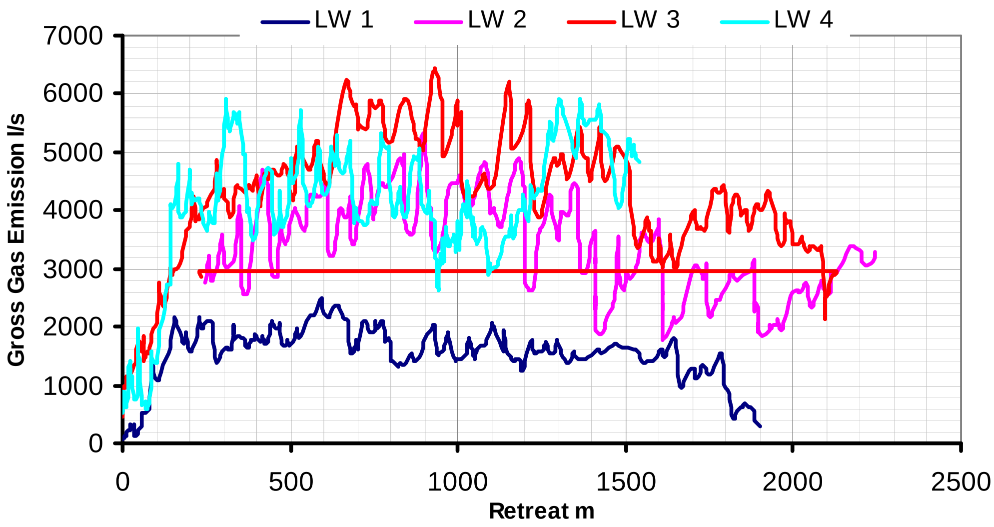


Figure A.2 Gross Longwall Gas Emission (Ventilation plus goaf drainage)

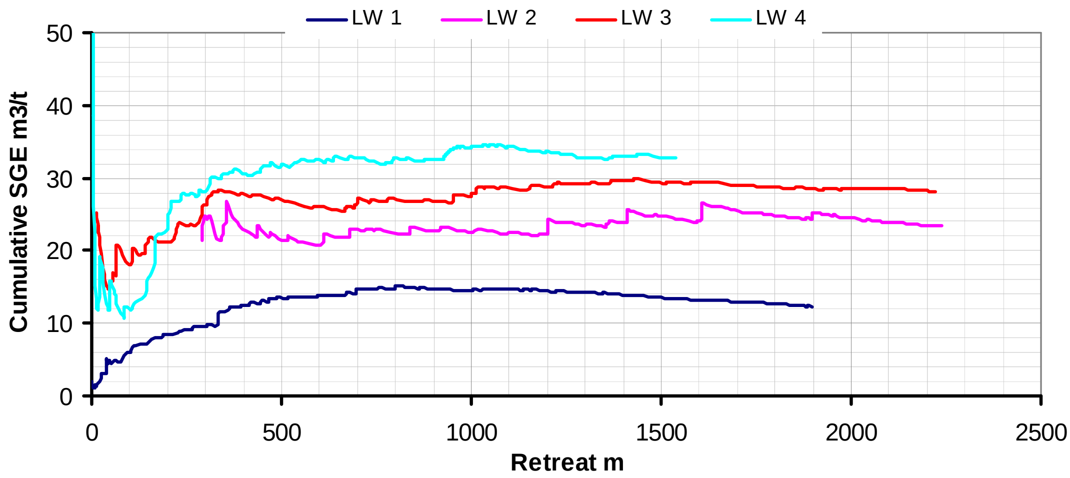


Figure A.3 Longwall Specific Gas Emission Rates

A.3 Gas Management Strategy

Development phase outburst and frictional ignition limits are reached using a combination of surface to inseam MRD holes supplemented with underground directional holes and compliance holes that are cored for gas content testing. The initial pit bottom area was pre drained with TRD holes.

The original plan to employ three heading gate roads was correct in providing a longwall ventilation circuit capacity of 100 to 120m^3/s (2,000 to 2,400 l/s CH_4 at the return limit of 2.0%), if and only if the gas can be distributed evenly. It is important to note that, following the Moura disaster of 1994, coal mine regulations, guidelines and custom and practice in Queensland prevent mine's employing a full US or South Coast Bulli seam style flood ventilation bleeder system. However, controlled bleed with due consideration to the location of potentially explosive mixtures and control of spontaneous combustion is possible.

In any event, the realistic dilution capacity of a bleeder system in these blocks is well below total longwall gas emission rates and alternative strategies are required. To date, the mine has successfully employed conventional surface to goaf drainage holes (250mm diameter at 100m spacing located on the tailgate return side) to reduce the gas emission load on the ventilation system. This strategy has achieved an average 65% capture, (goaf drainage / (goaf drainage plus ventilation)) with peaks of about 80% at high gas stream purity (>>90% CH_4) in longwall 3 and 4, Figure A.4.

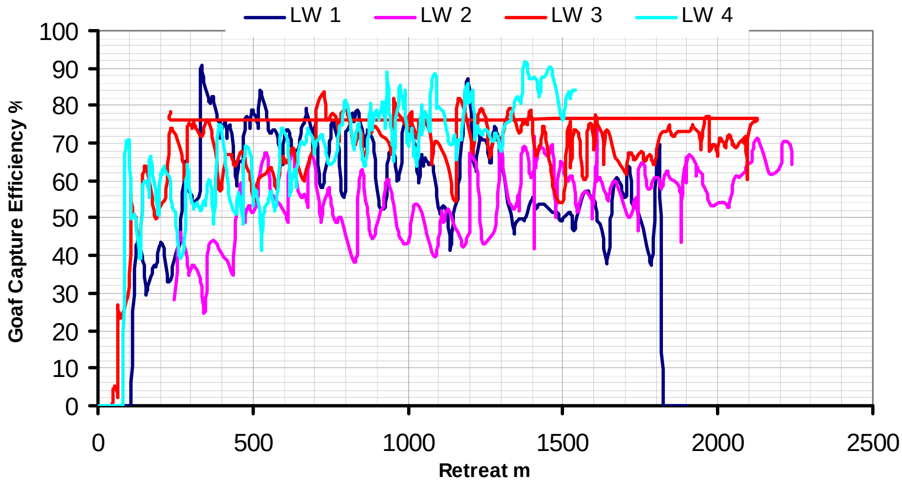
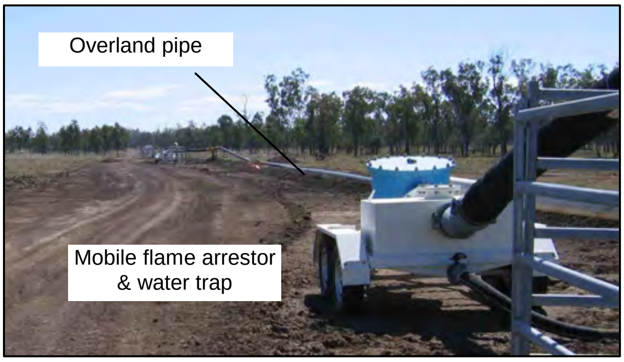


Figure A.4 Goaf Drainage Capture Efficiency

Due to unconstrained surface access, all gas is reticulated through surface 450mm diameter pipes, including that from vertical connections to underground directional holes i.e. an underground gas reticulation system is not employed. All surface gas streams from underground pre drainage, surface MRD pre drainage and goaf holes report to a central pump station from where about 2,200l/s of gas is discharged to 16 of 2.0MW gas engines with the balance reporting to flares, Figure A.5. The site policy is to avoid direct discharge of captured gas if at all possible.

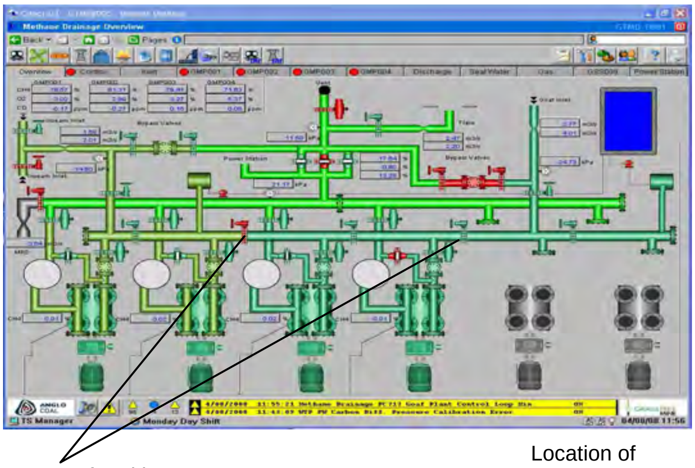
Recognising that, in future blocks, gas emission to ventilation net of goaf capture will still prove problematic for the ventilation system, the mine is now attempting to also pre drain thicker roof target seams using 2.0km long MRD holes drilled on the axis of blocks. These holes will serve two purposes. Firstly, to reduce the gas reservoir present in non working roof seams and secondly as goaf drainage holes targeting close face gas emission. The main limitation of this strategy is that the roof gas reservoir is contained in multiple seams and porous interburden which suggests that multiple completion CBM type hydrofracture wells may also be appropriate above future deeper workings.



Mobile Water Traps and Flame Arrestors

Remote MRD Holes

Pump Station



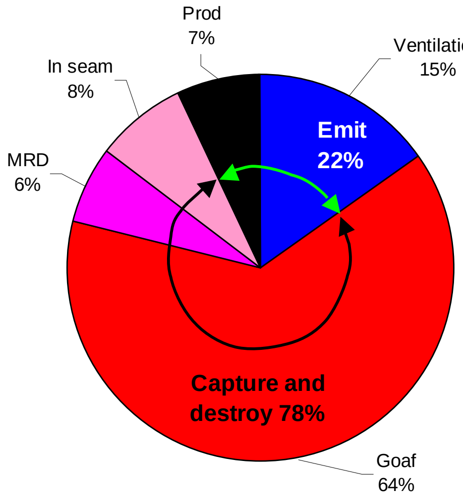
Pump Station Schematic

Power Generation

Figure A.5 Gas Management Infrastructure

A.4 GHG Emission Mitigation Opportunities

This mine is already capturing a high fraction of potential gas emission, including pre drainage of the working section to reduce surface release from stock piles. Most of the captured gas streams are passed through flares or engines to minimise net CO_2-e emissions. Figure A.6.



Item	l/s
Ventilation	1,200
Goaf	5,000
MRD	500
In seam	600
Prod	550
Total	7,850
Capture	6,100
Emit	1,750
Power	2,200
Flare	3,900

Figure A.6 Overall Methane Stream Distribution (Nominal Values)

The main opportunity for improvement, other than capturing more gas by means of MRD goaf drainage, would be to destroy VAM by means of thermal or catalytic oxidation units. However, the current ventilation circuit does not maximise the use of rear bleeder shafts with a significant fraction of longwall gas emission being mixed with return air at much lower (<<0.4%CH_4) methane concentrations. Figure A.7.

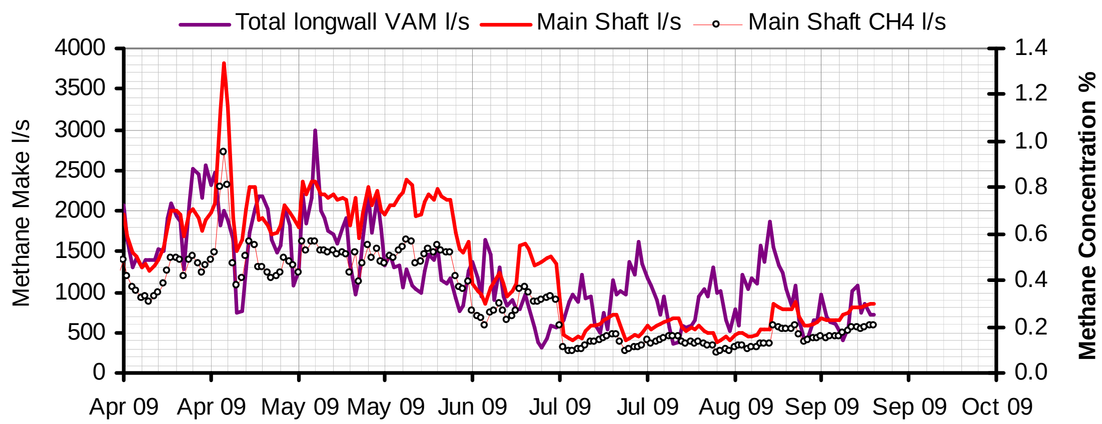


Figure A.7 Longwall and Main Shaft VAM

A conventional bleeder shaft is, by design, an ideal split ventilation system for destruction of VAM and has the potential to provide a very efficient method of destroying most VAM emissions at this mine. The comparison between the sum of methane make reporting to the longwall circuit and that reporting to the main exhaust shaft shown in Figure A.7 indicates that the vast majority of mine VAM is that from the longwall.

A number of issues arise from this analysis;

1. The total VAM from the longwall is about the same as that reporting to the main shaft for much of the time but there are clearly times where the total calculated longwall VAM exceeds the calculated VAM reporting to the main shaft. This is not possible.

2. If all longwall VAM were to be discharged to the bleeder shaft (say $100m^3/s$ at 1.0 to 1.5%CH_4) then there would be very low emissions from the main shaft with methane concentrations below VAM limits.

3. The precision of this balance is not accurate enough due to location of sensor points and use of monthly ventilation values to calculate gas makes.

4. Current VAM emissions would incur a substantial CO_2-e charge in millions of dollars (if applicable) and, if included, gas emitted from stockpiles would also incur a significant charge. This total could be reduced by application of VAM to bleeder shafts and increasing pre drain times.

APPENDIX B – GHG EMISSION MITIGATION IN MINES WITH MEDIUM GAS EMISSIONS (CASE STUDY B)

Case study B is concerned with a medium gas emission mine (50< GRS < 80m^3/m^2 and WS < 8m^3/t and < outburst limit m^3/t) located in the Hunter Valley. Gas data values used in here are those from 2000 to 2004, prior to the need for gas capture for GHG emission mitigation purposes becoming a planning issue.

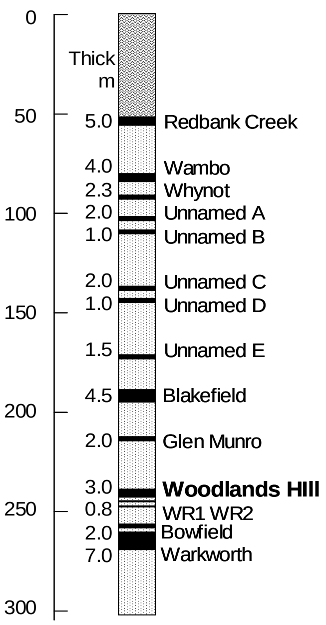
While case study A is an example of intense high efficiency gas drainage, case study B represents the potential distribution of gas streams in a mine that only goaf drains in gas contents below outburst limits. The issue to address now is the CO_2-e emissions/cost of such a strategy even if it would otherwise be acceptable.

B.1 Gas Reservoir Conditions

A series of longwall blocks is located in a 3.1m high seam with gas contents ranging 3 to 8m^3/t and with a composition of about 70% methane and 30% carbon dioxide. Depth of cover is 150m to 275m with surface access partly constrained by surface features, Figure B.3.

The outburst limit for this seam is about 9.0m^3/t and the risk of frictional ignitions is low. There are multiple floor and roof seams containing 15m to 20m of coal within the nominal cave zone, Figure B.1. All these seams are known to have a similar gas content – depth relationship as the working section, Figure B.2. This provided a gas reservoir size of 45 to 60m^3/m^2.

As a consequence of these various reservoir characteristics, the mine did not employ pre drainage but relied on increased ventilation rates and goaf drainage, where surface access permitted.



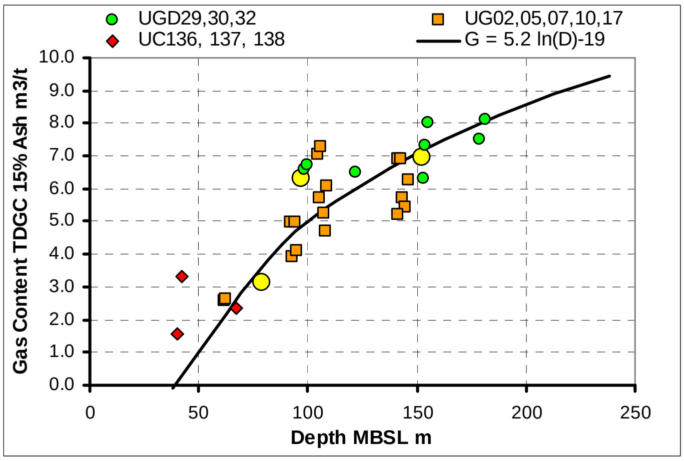


Figure B.2 Gas Content – Depth Relationship

Figure B.1 Stratigraphic Column

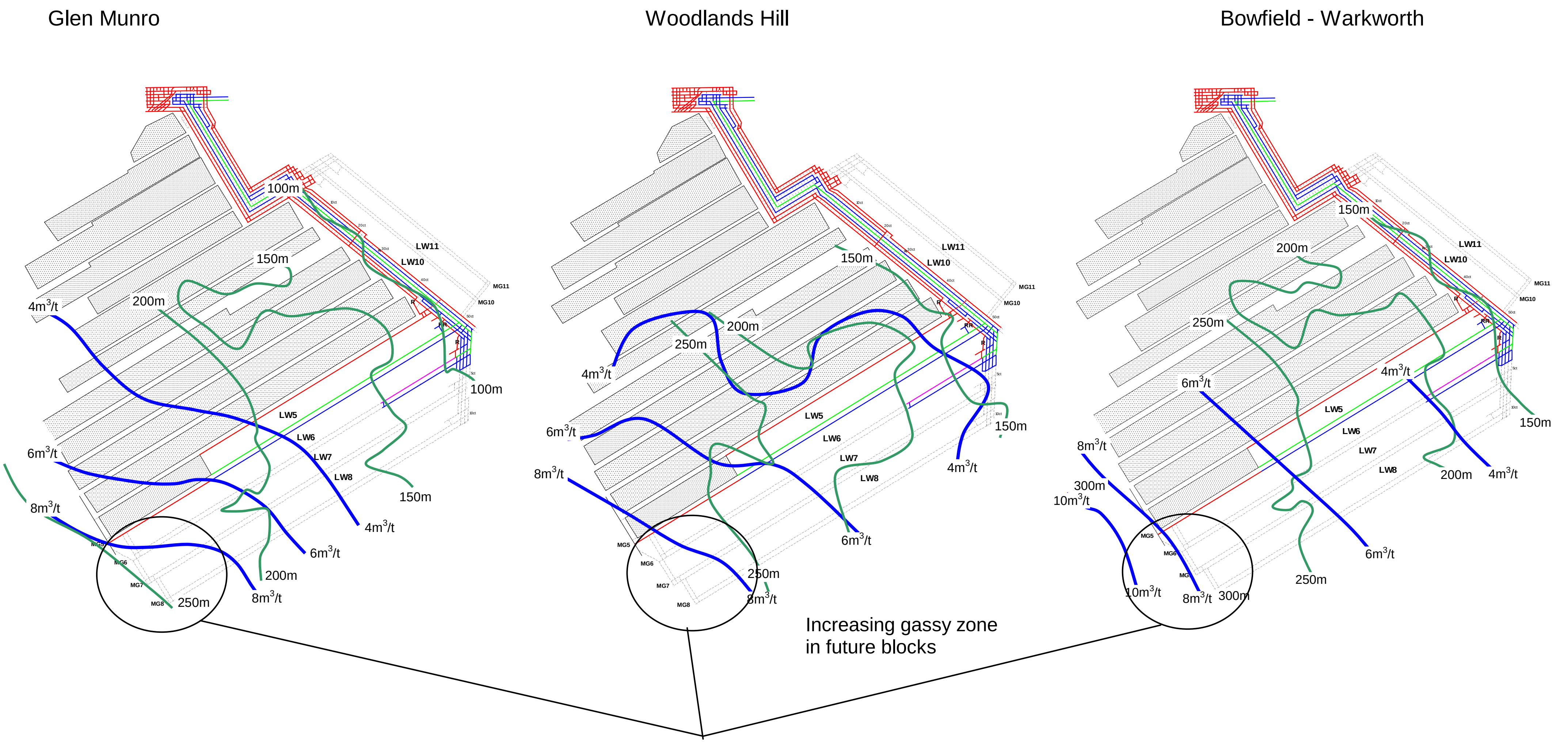


Figure B.3 Seam Gas Contents And Depth Of Cover

B.2 Gas Emission Values

Gas emission from undrained development panels reached 300l/s, Figure B.4, which started to cause problems complying with the NSW hazardous zone limit of 0.25% CH_4. An exemption was sought and granted to raise the limit to 0.5% CH_4 at the panel transformer. This allowed development to continue without pre drainage. Overall development emissions for gate road and mains was about 400l/s.

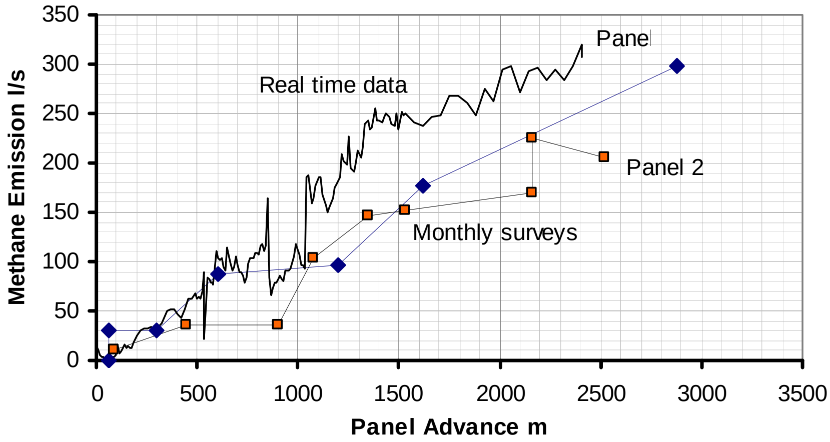


Figure B.4 Undrained Gate Road Rib Emission

The total balance of VAM streams for longwall 2 is shown in Figure B.5. Notably, emission from development and sealed areas amounted to about 1,300l/s prior to commencement of longwall operations. During longwall operations, goaf connectivity meant that a significant fraction of active and sealed goaf gas was captured by surface goaf drainage holes with the balance of about 1,500l/s reporting to the longwall circuit. Quite clearly, both floor and roof seams were significant contributors to longwall gas emission.

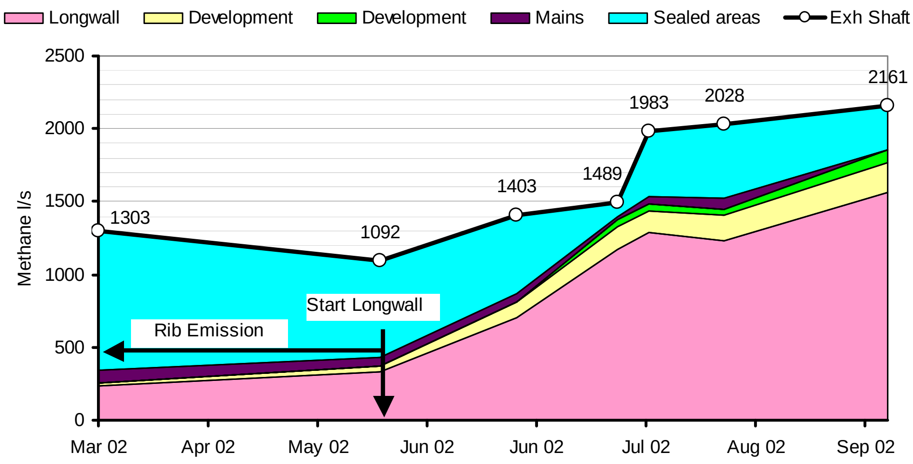


Figure B.5 Total Mine VAM Balance

Various issues associated with goaf drainage are shown in Figure B.6. This mine is prone to spontaneous combustion which meant that the intensity of goaf drainage was limited by oxygen ingress with an overall capture efficiency of about 40% achieved.

The top graph shows methane reporting to each of the plants and the specific hole location.

The lower graph shows average and maximum methane make reporting to tailgate ventilation. The axes have been aligned so that features from both graphs can be compared.

Although a rather complex situation, there are some points of note.

A. Conventional ramp up without plants operating. peak at 1,500l/s.

B. No significant change underground when total plant flow reduces.

C. With plant 3 operating on hole 2C, a reduction in underground methane make is observed even though production rates remain high.

D. Underground methane make remains stable when plants off and slow production.

E. Underground methane make drops when 2 plants remain on during period of low production.

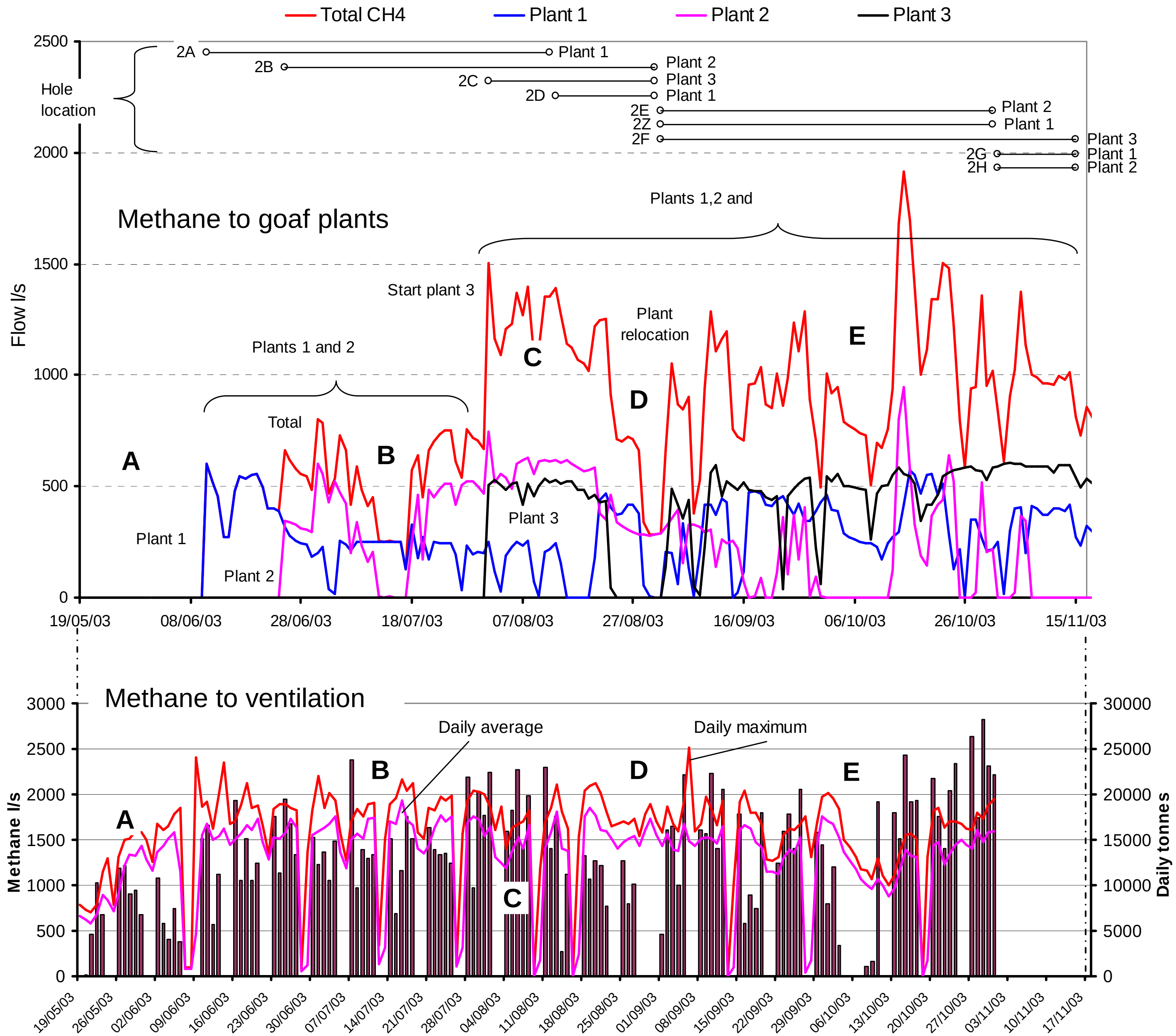


Figure 8.6 Methane Reporting to Ventilation And Goaf Plants

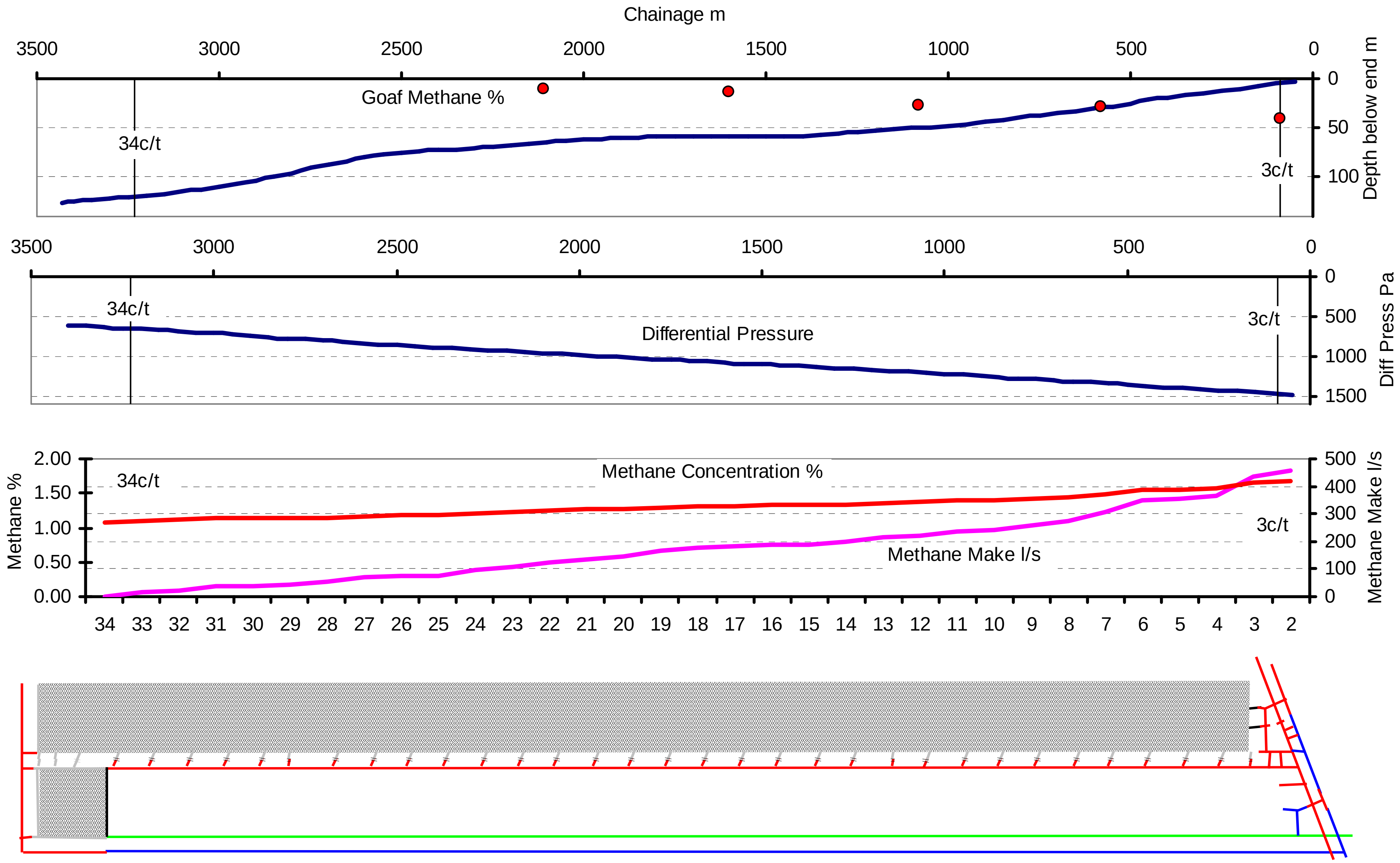


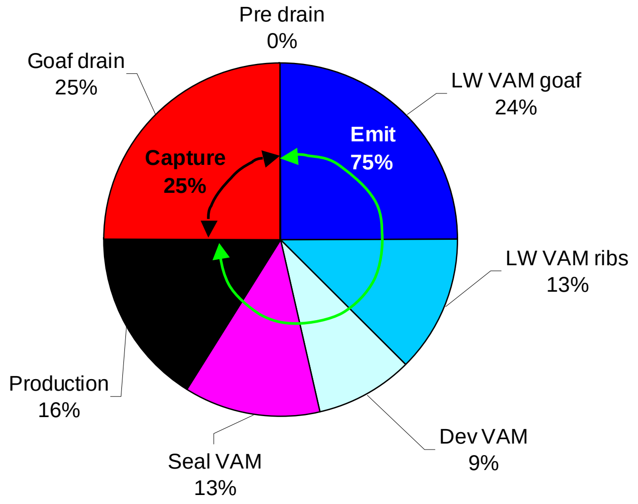
Figure B.7 Leakage from Sealed Goaf to Longwall Tailgate Through Seals

A further significant source of VAM in this mine was sealed area leakage to tailgate ventilation due to buoyancy pressures in adjacent goaves, Figure B.7. The recovery road was about 125m above the face start line that gave rise to outbye seal pressure differentials of 1.0 to 1.5kPa (breathing out). Notably, the sealed area methane concentration increased from 10% inbye to 50% outbye indicating a tendency to draw in air to the rear of the goaf, the “chimney “ effect.

The main consequence of this situation was that the methane make to the longwall due to leakage through seals varied with barometric pressure but was normally about 450l/s of a total 1,500l/s to 2,000l/s. From an operational point of view, this was at times sufficient to stop longwall operations by putting the outbye return methane concentration above 2.0% CH_4. Other than the magnitude of the gas source, the main leaning point here is that it could only be quantified by having tube monitoring points inside and outside the goaf along the length of the tailgate. A single outbye monitoring point would not have been sufficient.

B.3 GHG Emission Mitigation Opportunities

The distribution of gas streams at the time of these surveys (2000 to 2004) is shown in Figure B.8. The mine was directly emitting about 75% of all methane and capturing about 25%. However, initially all of the captured gas was emitted from goaf drainage plants, Figure B.9. Since this time, the mine increased the use of ground level enclosed flares, of the type shown in Figure B.10 to destroy captured methane.



Item	Actual l/s	Fraction %
LW VAM goaf	1,000	25.0
LW VAM ribs	500	12.5
Dev VAM	350	8.8
Seal VAM	500	12.5
Production	645	16.1
Goaf drain	1,000	25.0
Pre drain	-	0.0
Total	3,995	100
Capture	1,000	25.0
Total emit	3,995	100.0
Power & flare	-	0.0

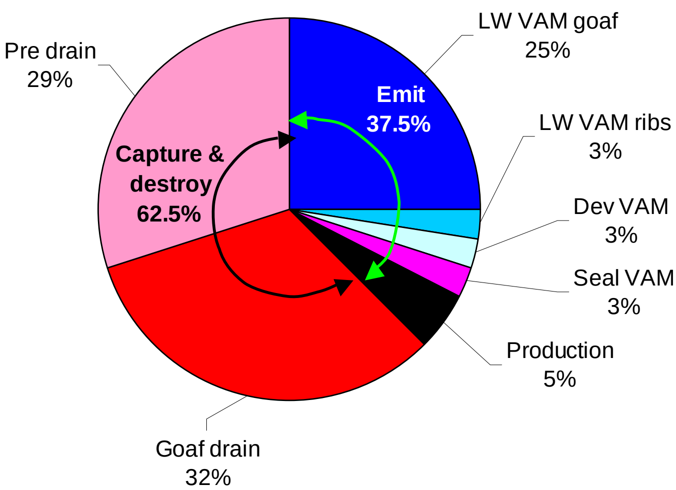
Figure B.8 Overall Methane Stream Distribution (Nominal Values)

Figure B.9 Goaf Drainage Pumps & Discharge

Figure B.10 Ground Level Flare (of the type used)

This is an example of a mine that avoided pre drainage by means of increased ventilation rates and re location of the hazardous zone boundary. Historically, this was an acceptable and common sense strategy when considering the cost, operational issues and health & safety risks associated with pre drainage systems.

However, with pre drainage and additional sealed area drainage the potential distribution of methane streams is shown in Figure B.11. It would be a reasonable expectation to increase overall capture from 25 to 60%. Methane destruction by flares would be the lowest risk strategy until gas flow rate profiles could be established with confidence.



Item	Potential l/s	Fraction %
LW VAM goaf	1,000	25.0
LW VAM ribs	100	2.5
Dev VAM	100	2.5
Seal VAM	100	2.5
Production	200	5.0
Goaf drain	1,300	32.5
Pre drain	1,195	29.9
Total	3,995	100
Capture	2,495	62.5
Total emit	1,500	37.5
Power & flare	2,495	62.5

Figure B.11 Potential Overall Methane Stream Distribution (Nominal Values)

Without pre drainage and assuming flares were applied to existing goaf drainage streams, then emission to atmosphere would have been about 3.0m^3/s CH_4 , with substantial carbon charge, if applicable. With pre drainage and some additional sealed area capture, the potential net emission could be reduced to about 1,500l/s CH_4 if no further improvement in longwall gas capture were to be made due to the risk of spontaneous combustion. The annual CO_2-e charge would then be reduced by 50% of that without pre drainage.

At 30 m hole spacing, the annual cost of pre drainage would be between $3.0 M and $4.0M. However, it is to be noted here that even that additional cost of pre drainage, the annual net savings would be in millions of dollars due to substantial reduction in fugitive emissions.

With respect to the balance of VAM reporting to the exhaust shaft, the concentration would be 1,500 l/s in 200m^3/s = 0.75% CH_4 which would be acceptable and subject to capital cost, economically justifiable.

The overall conclusion is that the introduction of pre drainage and VAM oxidation systems at this mine would be justified for operational and economic reasons. In addition, the following strategies could be considered;

- Use proactive nitrogen injection to increase goaf drainage from active and sealed goaves.
- Increase the frequency of goaf hole installation to improve capture of close face emissions.
- Use high purity pre drainage streams to blend with variable goaf drainage purity for power generation.

APPENDIX C – GHG EMISSION MITIGATION IN MINES WITH LOW GAS EMISSIONS (CASE STUDY C)

Case study C is concerned with a low gas emission mine located in an environmentally sensitive area with restricted surface access. Planned production rates are 4.0Mtpy in 3.0 to 5.0km long 300m wide blocks with an extraction height of 3.1m.

C.1 Gas Reservoir Conditions

As with most Australian coal mines, there are multiple seams in the roof and floor of the working section with a total of 30 to 40m of coal in the nominal cave zone, Figure C.1. The difference in the coal thickness profile for boreholes 1 and 2 is due to a 30m thick sandstone/conglomerate member in the outbye end of blocks.

Gas contents of all seams are low ranging from less than the residual $0.75m^3/t$ to about $2.0m^3/t$ at 400m depth of cover, Figure C.2. This represents one of the lowest gas content regimes in Australia.

However, the combined effect of coal thickness and net gas emission provides a gas reservoir size of approximately $30 \times 1.4 \times 1 = 42m^3/m^2$ or a specific gas emission rate of 2 to $6m^3/t$ depending on the nature of roof interburden.

C.2 Gas Emission Values

At the low gas contents present, rib emission rates are very low (<20 l/s per km) and a ventilation rate of $30m^3/s$ is sufficient for gate lengths of 3 to 5km. Similarly gas emission prior to longwall start up is low to negligible with methane concentrations at commencement of the hazardous zone below 0.1% CH_4.

During longwall extraction, caving of the multiple roof seams results in specific gas emission rates of 2 to $6m^3/t$ with resultant gas emission to ventilation of 400 to 800l/s, Figures 3 and 4. The mine was able to operate without gas constrained production, although above 1.0% CH_4 in the tailgate, with face ventilation rates of 40 to $45m^3/s$.

In addition to active longwall gas emission there was an additional 200 to 300l/s CH_4 emitted from sealed areas (particularly through tailgate seals) during periods of falling barometric pressure.

Total mine gas emission ranged 600 to 1,000l/s CH_4 with a total mine ventilation rate of $165m^3/s$. The relatively low ventilation rate was sufficient to support the single longwall and two CM units. However, unlike the previous two case studies, there is no opportunity to reduce ventilation rates if gas was to be captured.

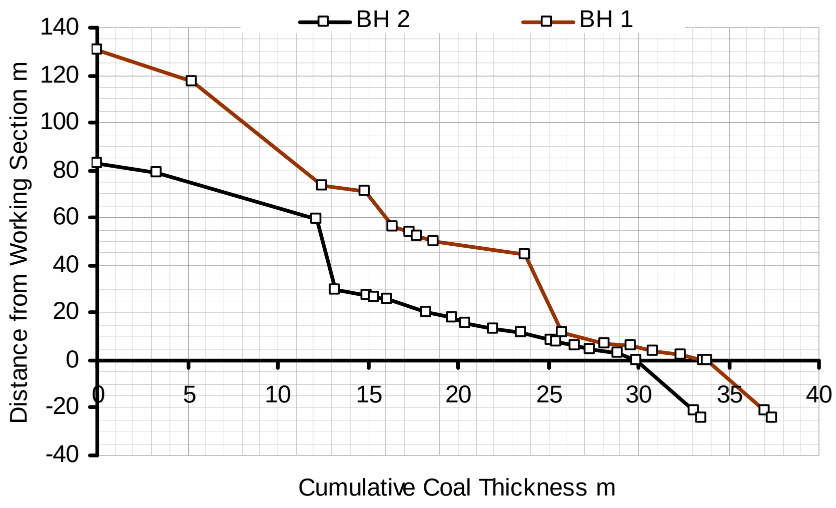


Figure C.1 Coal Thickness in Cave Zone

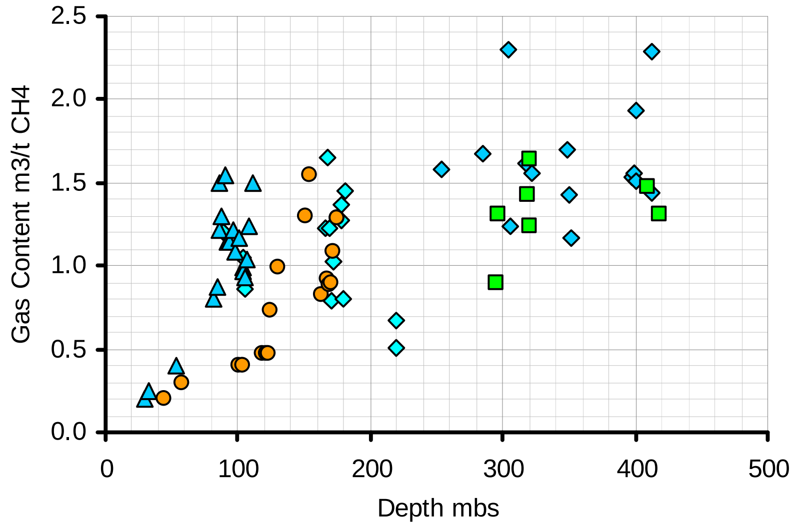


Figure C.2 Gas Content – Depth Relationship

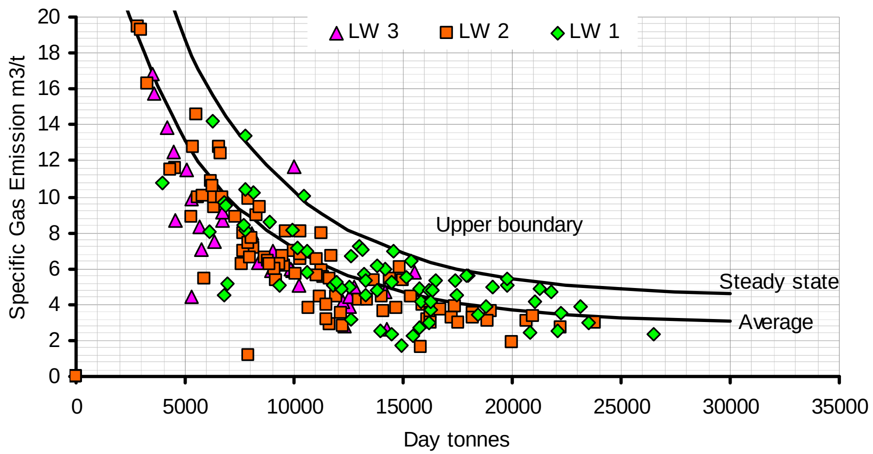


Figure C.3 Observed Specific Gas Emission

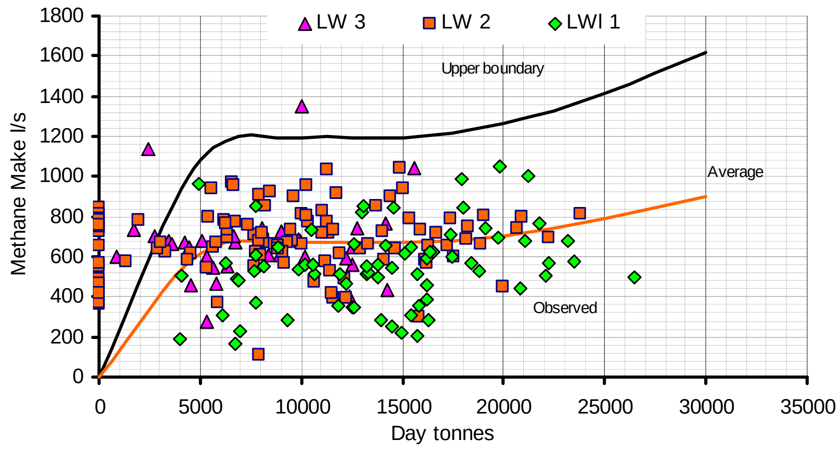
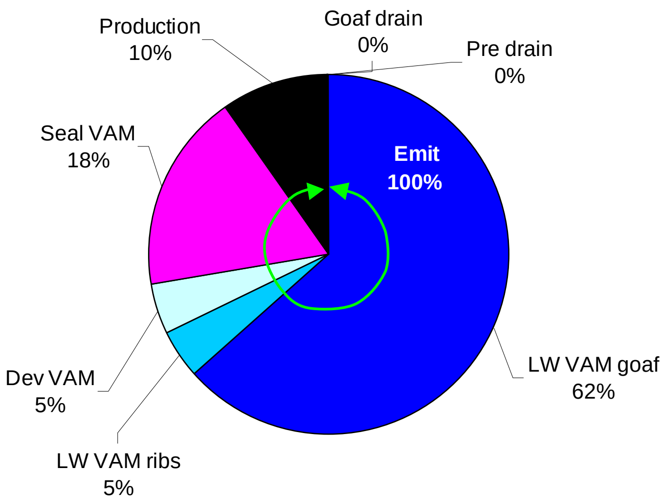


Figure C.4 Observed Gas Emission Rates

C.3 GHG Emission Mitigation Opportunities

The average distribution of methane streams is that shown in Figure C.5.



Item	Average l/s	Fraction %
LW VAM goaf	700	63.2
LW VAM ribs	50	4.5
Dev VAM	50	4.5
Seal VAM	200	18.1
Production	107	9.7
Goaf drain	-	0.0
Pre drain	-	0.0
Total	1,107	100
Capture	-	0.0
Total emit	1,107	100.0
Power & flare	-	0.0

Figure C.5 Overall Methane Stream Distribution (Nominal Values)

The main issues to consider at this mine are as follows;

- The low seam gas content means that pre drainage is not required for control of outbursts or rib emission. In addition, gas contained in production coal amounts to about 107l/s including residual gas contents.
- Even if goaf drainage were to be employed and a capture efficiency of 50% achieved, there would not be significant opportunity to reduce ventilation rates in the mine and therefore the capacity of VAM oxidation units would remain the same.
- Destroying methane by VAM oxidation alone would require 10 units. In this context, it could be considered that the ventilation system has in fact captured 90% of the mines emission (net that in production coal) and VAM oxidation provides the most efficient solution.
- Assuming that goaf drainage is not required for operational reasons, then it should be avoided so as to maximise the effect of VAM oxidation units i.e. without goaf drainage the return shaft methane concentration is about 0.63% but would fall about 0.3% with a goaf capture efficiency of 50% were to be achieved. The mine would then have to manage both gas streams.

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