GUIDELINES

MDG 1006-Technical Reference

Technical Reference for Spontaneous Combustion Management Guideline

Produced by Mine Safety Operations Branch
Industry and Investment NSW
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DISCLAIMER

TABLE OF CONTENTS

1 SCOPE

The content of this document applies to all underground coal mines in Australia.

2 PURPOSE

The purpose of this document is to provide historical and technical information to assist operators in the development of a Spontaneous Combustion Management plan that complies with MDG 1006.

The technical reference is not intended to be a complete reference work on the subject of spontaneous combustion but rather focus on some issues of importance. References are provided for other information on spontaneous combustion.

3 FUNDAMENTALS OF SPONTANEOUS COMBUSTION

Spontaneous combustion describes the process of self-heating of coal by oxidation. After exposure by mining, coal undergoes a continuous exothermic oxidation reaction when exposed to air.

A hazard exists when, in confined areas, the rate of heat accumulation due to oxidation exceeds the rate of cooling by ventilation or environment. The coal can then increase in temperature until combustion takes place leading to the emission of toxic and explosive gases together with propagation to open fire. The self-heating will then become a potential ignition source for an explosion if exposed to a flammable mixture of gas.

Spontaneous combustion of coal occurs by the following steps:

- Oxygen (from airflow and ventilation) reacts with coal. This is called oxidation.
- Oxidation produces heat. This is called an exothermic reaction.
- If this heat is lost to the surroundings (mine environment), then the coal mass will cool. However, if the mine environment favours the heat being retained, the coal mass will increase in temperature and the oxidation rate will increase leading to spontaneous combustion. Significant amounts of heat can also be generated when the coal absorbs moisture.

Heat generated is lost by some or all of the following mechanisms, depending upon the temperature and physical conditions of the mine:

- Conduction through the solid coal mass.
- Conduction and radiation to the ventilating air.
- Evaporation of moisture.
- Convection through the solid coal mass and ventilating air.

The oxidation process is complicated, and not fully understood, but the following stages occur as the temperature of the coal increases:

- The absorption of oxygen by the coal and formation of oxycoal without the production of carbon monoxide. This is a reversible process.
- As the temperature increases through the range 30^{o}-$40^{o}C$ the coal/oxygen complexes break down and produce carbon monoxide and carbon dioxide. This reaction occurs irrespective of the presence of atmospheric oxygen.
- Further increases in temperature are associated with increased rates of oxidation and the production of increased quantities of carbon monoxide and carbon dioxide.

There are two conditions of oxidation equilibrium that can occur:

- If the quantity of air flowing over a coal surface is very small, then the rate of oxidation is low. This is the condition that occurs in high resistance air paths, such as through goafs and in sealed areas.
- If the air quantity is large, the heat due to oxidation is lost as quickly as it is generated and this cooling effect may be enough to prevent any significant rise in temperature. This is probably the condition that occurs in almost all the low resistance intake and return airways.

Should this equilibrium condition be destroyed by either an increasing airflow in the first case, or a decreased airflow in the second case, then the temperature will rise and spontaneous combustion may result.

All coals are liable to spontaneous combustion if the conditions are right.

4 PREDICTION

4.1 HISTORY

The history of spontaneous combustion events at a mine, adjacent underground and open cut mines and the same coal measures in other areas, is invaluable in providing guidance on the propensity for heating, location of heatings and behaviour of the coal (gas evolution) as it self-heats.

There may be considerable information where the coal seam has been mined extensively. This may indicate a high or low propensity for spontaneous combustion. Testing coal for propensity for spontaneous combustion is useful although there are limitations in its validity. Information from operating experience in the seam is of great value.

4.2 DEVELOPMENT OF HEATINGS

Conditions for the development of heatings typically exist in out of the way places such as goaves where they cannot be seen. Where they occur in ventilated and accessible roadways, they are hidden below the surface of coal stowage, or within the rib side of a pillar. Coal that heats on the surface of stowage etc. is cooled by the ventilation flow. Ideal conditions exist deeper in the coal mass.

Heatings are difficult to discover and are often not detected until well advanced. They may commence as small football size shapes, giving off low volumes of gas in a goaf, which is difficult to detect.

Gaseous products of heatings in goaves may not be easily detected in adjacent ventilated roadways because of the irregular and intermittent ventilation flow from the source, barometric changes, temperature variations and the passage of air through the goaf where absorption or dilution may take place.

A heating in a surface coal or refuse stockpile provides an opportunity to observe behaviour. Coal stockpiles are readily accessible although the heating sites cannot be seen in early stages because they develop below the surface within the coal mass.

Coal on the surface of a stockpile where it can be seen, does not self-heat. There is sufficient oxygen on the surface for oxidation to take place but not conditions that favour the retention of heat. Again, heatings tend to commence as small football sized shapes within the mass of the stockpile. The temperature in such a shape may be higher than normal (60^o to 80^o) and in the adjacent area, normal.

Unless there are attempts to monitor temperature changes within the heaps, spontaneous combustion is more likely to be detected in an advanced stage by smell, visual observation of shimmering (heating) of the air above the heap, or smoke and flames when the coal is loaded out.

4.3 LOCATION OF HEATINGS

An appreciation of the characteristics of spontaneous combustion and an understanding of the places in the mine where heatings may develop is critical to the development of an effective Spontaneous Combustion Management Plan (SCMP). Prevention, early detection, and control of the spontaneous combustion risk will not be effective unless the potential hazards and locations are correctly identified.

Places in the mine where heatings may develop include:

4.3.1 Longwall Extraction Area

The conditions required to initiate a heating are more likely to exist in a goaf than in other parts of the mine. The risk of heating in an active longwall goaf is greater as there are a number of flow paths available into areas that cannot be sealed by strata consolidation. The major factor preventing heatings is the exclusion of oxygen by accumulation of seam gases.

Spontaneous combustion in the active longwall goaf may be caused by air drawn behind the roof support line or leakage through a goaf edge seal. The area of greatest risk is the edge of the goaf where there is rib spall, voids, incomplete caving and close proximity to ventilated roadways. Air permeability is higher and high ventilation pressure and poor containment can allow air to enter the goaf.

Heatings are unlikely to develop in a fully caved area because the fallen rock buries the potential heating location and air permeability is low. (Consolidated area) Replenishment of oxygen in the fully caved area is unlikely to be adequate to sustain spontaneous combustion.

In mines where longwall gate roads have to be heavily supported, goaf formation alongside chain pillars may be delayed or incomplete resulting in cavities extending a considerable distance into the goaf. Air may flow into the goaf due to pressure differences around the panel (where bleeders are used) or into and across the goaf behind the longwall face. This was believed to be an issue at Moranbah North in Qld and at Dartbrook in NSW.

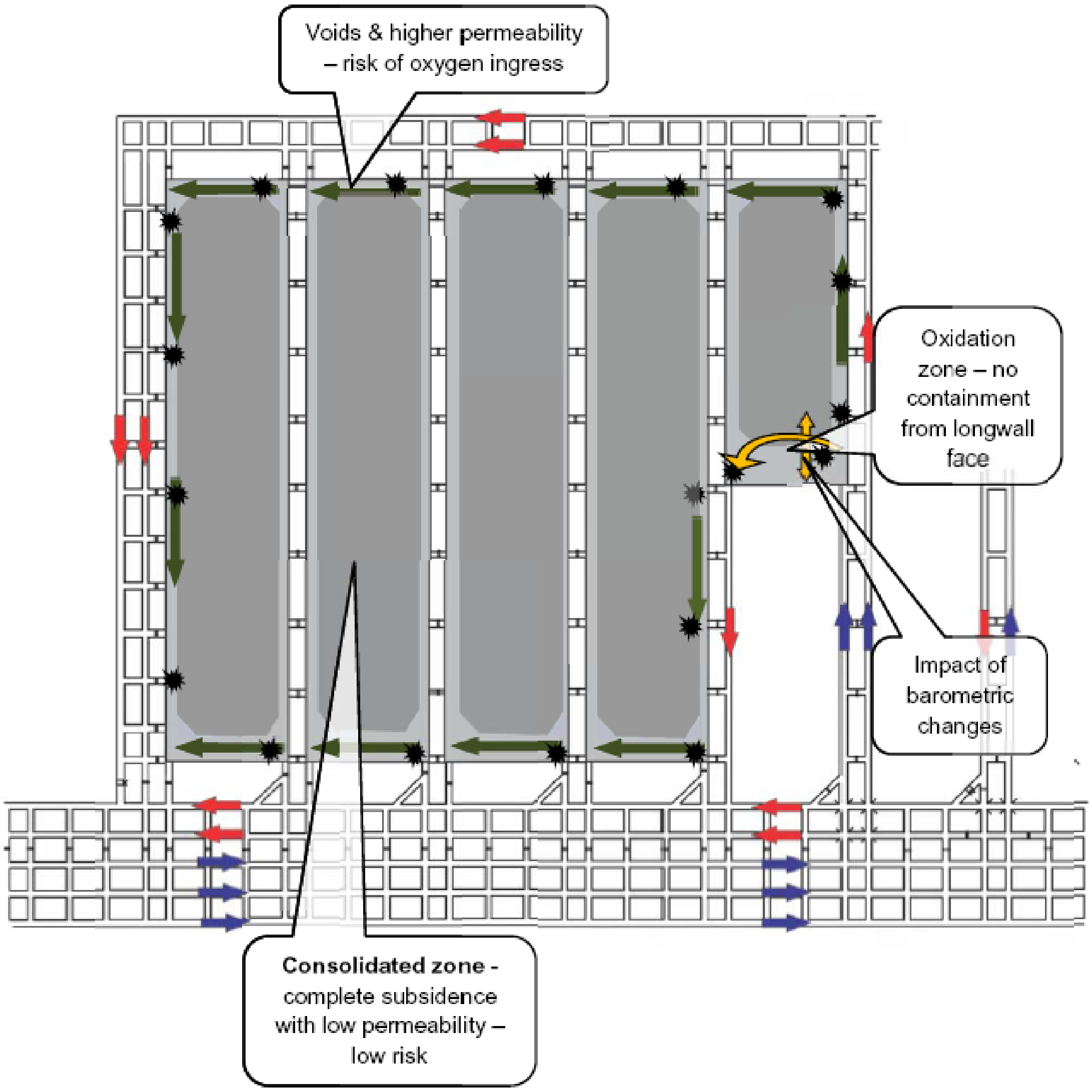
Sub critical extraction systems, (associated with limited surface subsidence) designed with stable chain pillars, may result in voids above the caved area permitting increased airflow paths across the goaf. Rider seams in the area where there are voids pose a risk of spontaneous combustion.

Active long wall panels have an unsealed side adjacent to the goaf where the longwall face equipment is located. Oxygen can enter the extracted area due to face ventilation airflow and barometric changes. Barometric variations exceed the ventilation pressure difference across the face and can have a significant effect by moving air, in and out of the goaf, by expansion and contraction.

In an active longwall panel, there may be sufficient oxygen to allow oxidation to take place approximately 150m to 400m into the goaf from the long wall face. The distance will vary according to the frequency and severity of barometric changes, dip of the seam and direction, natural inertisation processes and the standard of containment structures.
If operating conditions result in a protracted delay in long wall face retreat, there is a risk of spontaneous combustion developing in the goaf.

Figure 1 shows areas in a longwall extraction area where spontaneous combustion may develop if preventative measures such as the standards for stoppings and seals and ventilation pressure difference are inadequate.

Figure 1: Longwall goaf - hazards



Legend

	Consolidate zone - extraction completed, caving and full subsidence has taken place - low permeability, effective inertisation & low risk
	Area of voids & higher permeability alongside the fully caved goaf –risk of heating if containment poor and the impacts of high ventilation pressure result in ingress of oxygen – higher risk.
	Poor containment & high ventilating pressure may result in air ingress and air movement into the goaf in this direction.
	Possible heating sites

4.3.2 Bord & Pillar Extraction Area

The system of continuous miner extraction requires stooks to be left to protect operators in the working area. This ensures broken coal in the goaf, and may result in delayed caving.

Similar to a longwall, spontaneous combustion is controlled by:

- The consolidate zone where complete caving takes place,
- Inertisation as a result of containment, seam gases and oxidation,
- A regular and progressive extraction rate,
- Minimisation of pressure differences across & alongside sealed areas,
- Inspection and maintenance of seals and seal sites to control leakage, and
- Sampling and analysis of sealed area atmospheres.

Deficiencies in any of these control measures will increase the risk of spontaneous combustion.

Spontaneous combustion in sealed areas may be caused by air leaking into or through seals or the sealed area having an oxygen rich atmosphere.

Heavy weighting, seam structures and roof control problems may result in additional coal being left in the goaf. Incomplete extraction may delay caving, encourage greater air movement in the goaf and cause coal to be exposed to the risk of heating.

The rate of extraction with continuous miners is normally less than that of a longwall system.

Where discrete panels are developed with each panel having a barrier on 3 sides, containment and inertisation is effective except for the working area adjacent to the goaf.

Partial extraction systems are more often being adopted. Where such systems are used, caving may be incomplete and irregular, leading to voids and potential air paths in the extracted area. Stable pillars may be left and spans reduced to sub-critical. This results in voids in the goaf where air can flow and may increase the risk of spontaneous combustion developing within the goaf.

Figure 2 shows a system of panels isolated by barriers and typical locations where heatings can occur. With low depth of cover, or other seam workings within close proximity, there is the risk of interconnectivity through cracks and those areas of risk could extend to all goaf edges.

Heatings will only develop if inertisation is ineffective.

Figure 2 – Continuous miner isolated panels – hazards

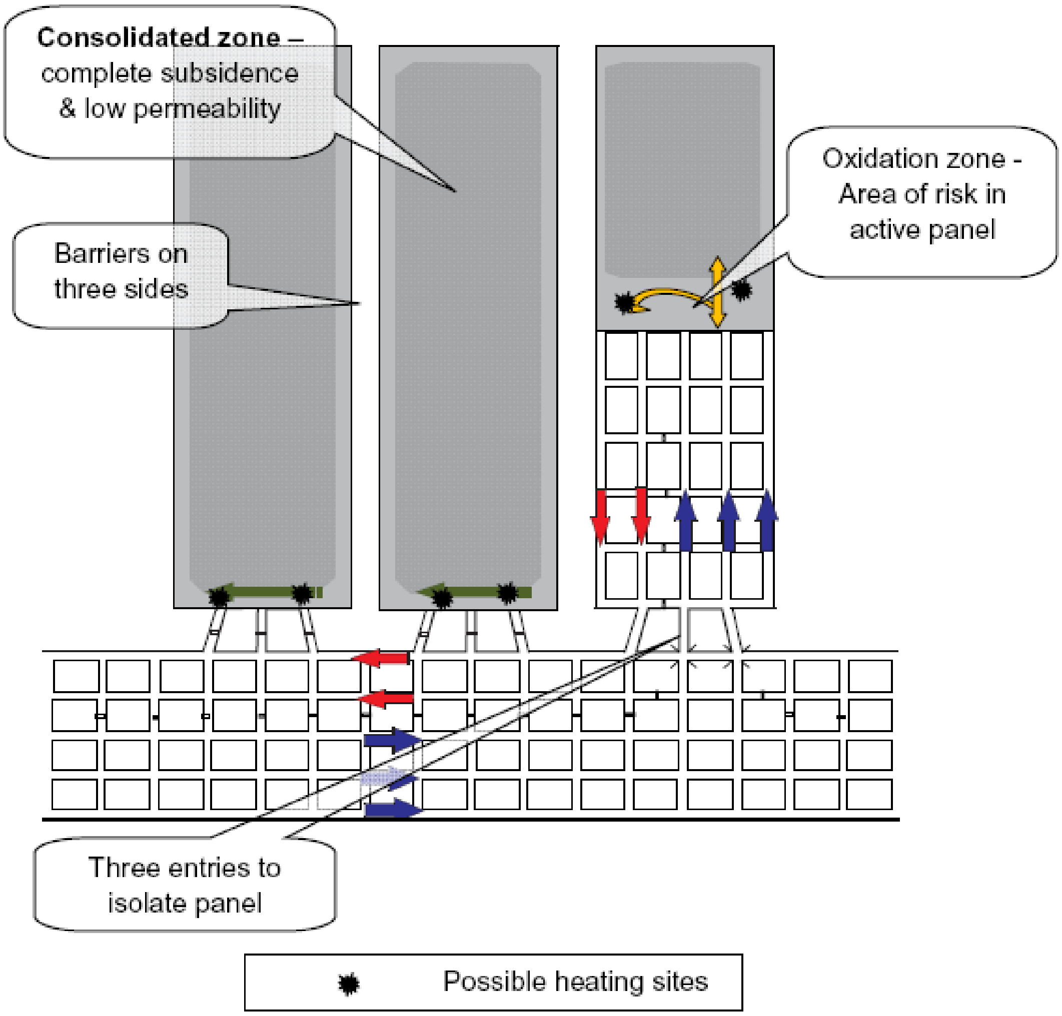
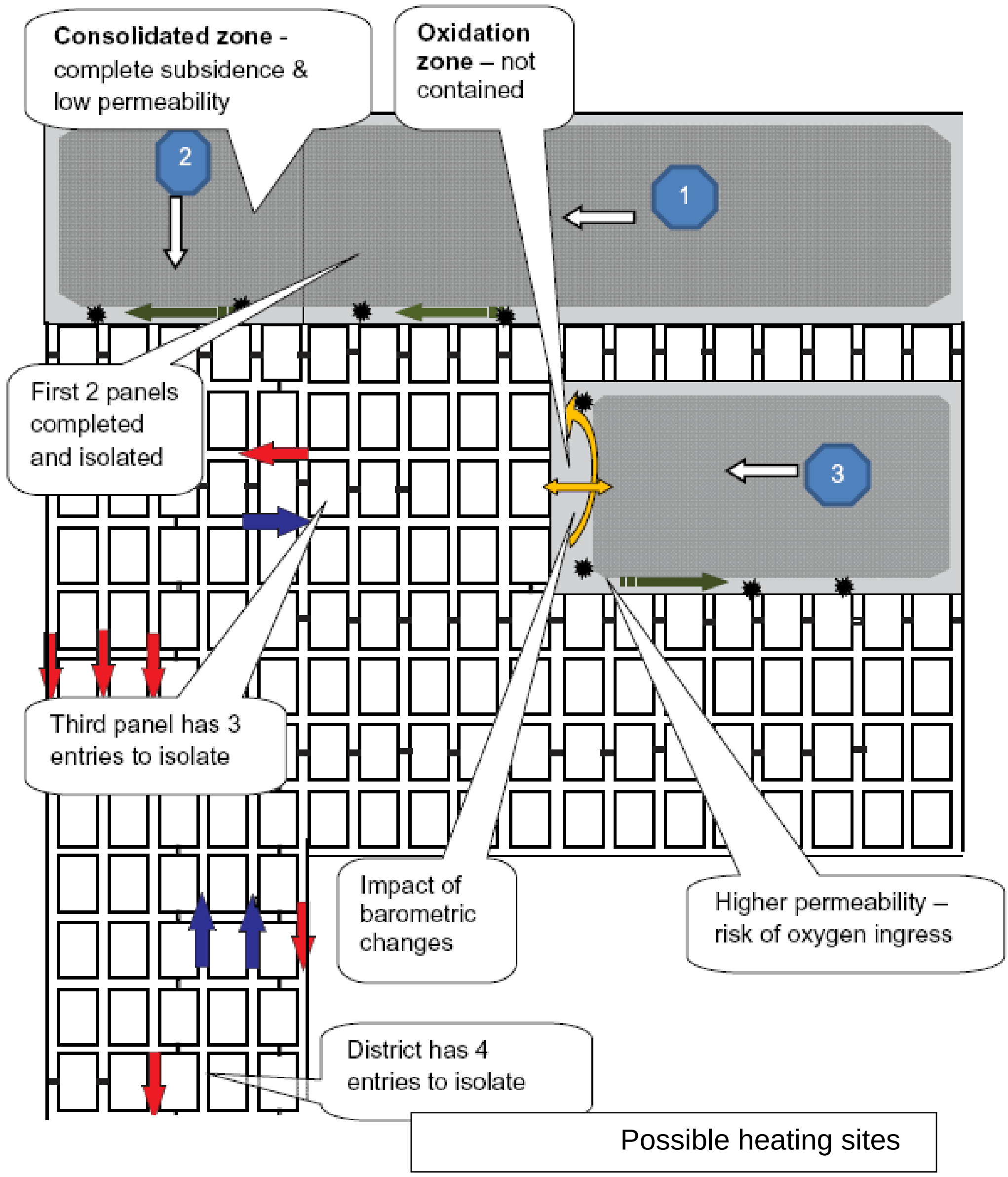


Figure 3 shows an arrangement where panels are not isolated by barriers and instead have reduced to manageable size by a line of stoppings. Areas of risk where heatings may occur in the goaf are shown. This assumes no interconnectivity from the surface, boreholes, or workings in another seam.

Heatings will only develop if inertisation is ineffective.

Figure 3 – Continuous miner interconnected pillars – hazards



4.3.3 Stowage (fallen tops & dumped coal)

A heating may develop in coal stowage or fallen top coal. Stowage can be likened to a surface stockpile where a heating develops. Conditions that favour the development of a heating in stowage include:

- Limited ventilation flow across the stowage or fall
- Height and mass of the stowage
- Ingress of moisture
- Ineffective inspection.

Long term storage of coal in a bin may self heat given the right conditions.

4.3.4 Rib Side Pillar

Pillar heatings, particularly where adjacent to ventilation stoppings, are generally caused by:

- High-pressure differential between intake and return airways and along a length of roadway.
- Fracturing in the rib side.
- Crushing of pillars.
- Presence of broken coal as accumulations or behind lagging.
- Flow of air to underlying or overlying workings.
- Air crossings with high differential pressures.
- Coals with high propensity.
- Leakage paths associated with cracks, cleat, fractures, faults, joints, friable seam bands, and unsealed boreholes.
- Box cut entries where the mine fan is located in the box cut and near the intake roadways result in high ventilating pressure in ground that may be damaged by blasting during construction of the box cut.

Shotcreting or equivalent sealing material is sometimes applied to control rib and roof stability and reduce leakage paths, particularly around return airway entries from box cuts, highwalls, drifts and shafts intersecting seams. The shotcrete is sometimes a contributing factor to either inhibiting the discovery of heatings by masking the heat present behind it, or reducing air leakage to a degree were oxygen is supplied but heat is not removed during oxidation of the coal. Cracks in shotcrete allow egress of air into the return airways from the intakes. These cracks in shotcrete require regular inspection for indication of changing gas emissions or radiation of heat.

Heating sites tend to be near and on the intake side of the stopping in the highest pressure difference area, i.e. closest to the mine entries. Such heatings may be difficult to detect until well advanced because of their relatively small size and the dilution of gaseous products by high volume airflows.

4.3.5 In-situ Coal

Heatings may occur in roof or floor coal that has been cracked or broken by convergence. Top coal or floor coal left in the mining process may be subject to heating under favourable conditions.

A heating has been known to take place in top coal in-situ in the roof. The coal was a few metres thick and subject to convergence and cracking. An adjacent area may have fallen, exposing one face of the tops to air ingress. The top coal fell and burst into flame. Edges of the roof fall from where the fall originated were hot enough to turn water from fire hoses into steam (Aberdare North Tunnel 1970).

4.4 HAZARD IDENTIFICATION

Matters that need to be addressed from the information collected and evaluated and in the identification of hazards that may lead to spontaneous combustion include the following:

4.4.1 Propensity to Spontaneous Combustion

Some coal seams have a higher propensity to spontaneous combustion than others. An evaluation of the liability to spontaneous combustion and an assessment of the hazards in a seam and mine environment should commence with tests on coal expected to be mined or affected by the mining operation.

Examples of scientific tests to determine coal properties relevant to spontaneous combustion include:

- Small scale tests such as the R70 and moist coal test (Beamish)
- Bulk testing of coal to simulate seam and goaf conditions
- Column tests to simulate gas evolution from coal as it heats

Refer to Table 1 for comparison of spontaneous combustion propensity.

Whilst tests are useful in determining propensity for self-heating, properties of the coal in the seam will vary in different parts of the mine and cannot be precisely simulated in the laboratory. A limited number of tests may not be indicative of the propensity for spontaneous combustion in all locations and conditions.

Figure 4: Adiabatic self heating curve

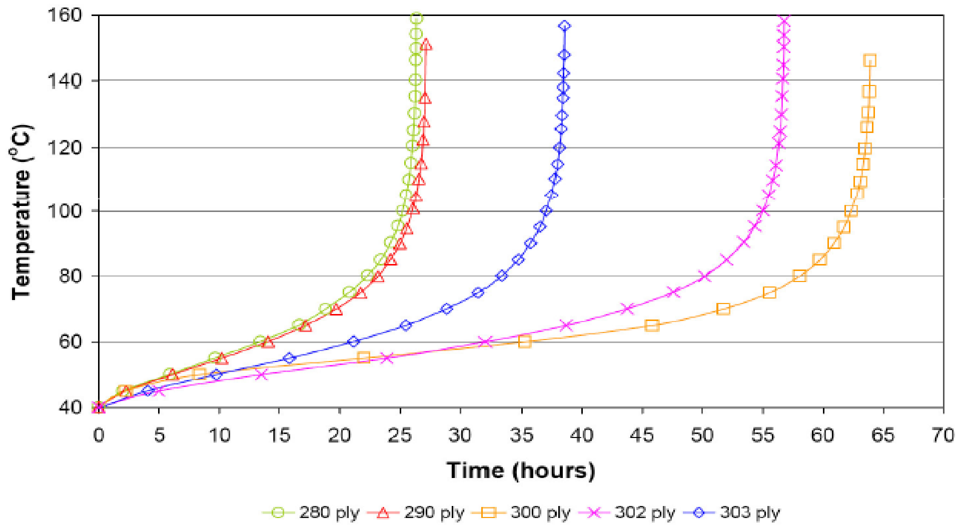


Figure 5: Self heating relationship with ash content and coal rank for Australian coals, showing intrinsic spontaneous combustion classes (chart courtesy of Bulga Coal)

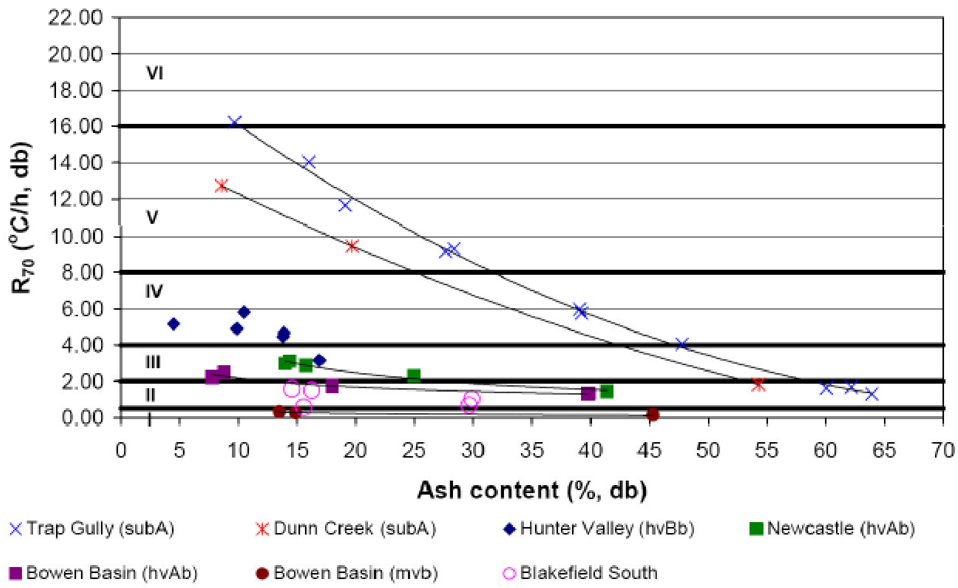


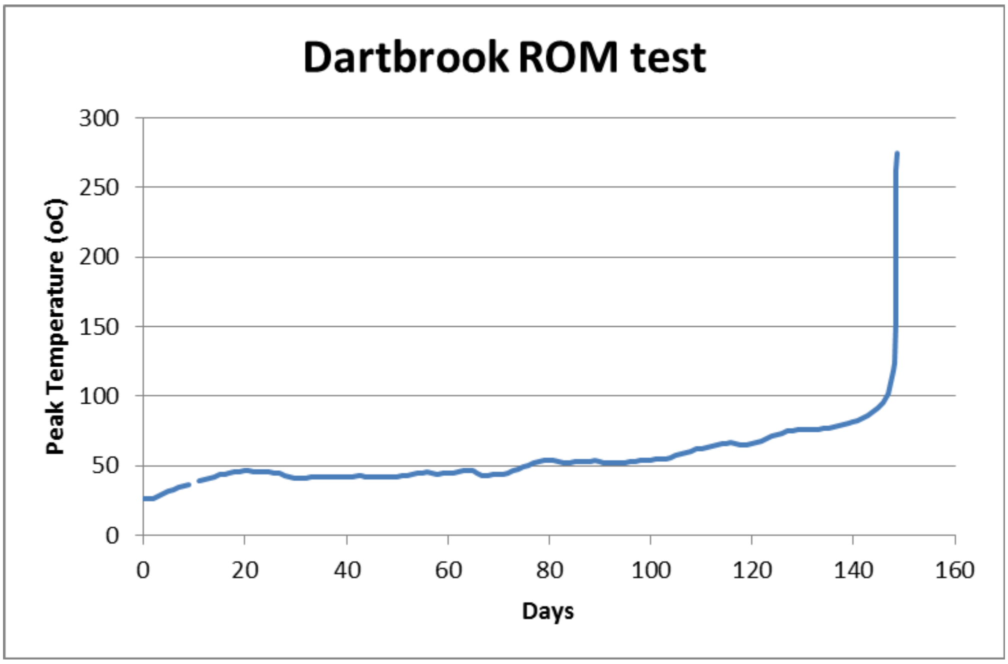
Table 1: Spontaneous Combustion Propensity Classification

Intrinsic spontaneous combustion propensity classification (ISCP) (based on Queensland and New South Wales coal conditions)			
ISCP Class	**Propensity rating**	**Queensland R_{70} value (°C/h)**	**New South Wales R_{70} value (°C/h)**
I	low (L)	$R_{70} < 0.5$	$R_{70} < 1$
II	low-medium (LM)	$0.5 \leq R_{70} < 1$	$1 \leq R_{70} < 2$
III	medium (M)	$1 \leq R_{70} < 2$	$2 \leq R_{70} < 4$
IV	high (H)	$2 \leq R_{70} < 4$	$4 \leq R_{70} < 8$
V	very high (VH)	$4 \leq R_{70} < 8$	$8 \leq R_{70} < 16$
VI	ultra high (UH)	$8 \leq R_{70} < 16$	$16 \leq R_{70} < 32$
VII	extremely high (EH)	$R_{70} \geq 16$	$R_{70} \geq 32$

There is a different rating scheme used for New South Wales conditions and Queensland conditions. The two schemes are required to take into consideration the different start temperature conditions that exist in both settings.

Figure 6 is an example of a bulk sample test of coal in a large scale reactor. It shows a (very) rapid rise in temperature after being apparently dormant for several months. This test demonstrates that development and progression of spontaneous combustion is sometimes erratic and unpredictable.

Figure 6: Results of bulk heating test for Dartbrook coal



4.4.2 Coal Rank

Generally, as the rank decreases, the moisture and oxygen levels and volatile matter of the coal increases, and the carbon content decreases. It is generally accepted that the lower the rank, the faster the rate of oxidation and the greater the tendency to spontaneously combust.

4.4.3 Pyrites

Sulphur minerals, iron pyrite (FeS_2) and marcasite may be present in coal seams as veins of highly crystalline mineral or in a finely divided state throughout a seam. When present as veins, the surface area exposed to oxygen is relatively small and contributes little to any heating.

A significantly larger surface exists when these minerals are in a finely divided state, and are able to react with oxygen to produce heat and a product that has a larger volume. The heat produced from oxidation of the pyrite increases the temperature of the coal and the rate of oxidation, and the increase in volume causes fracturing of the coal that exposes a greater surface area for further oxidation. Generally, pyrite must be present in concentrations > 2% before it has a significant affect.

4.4.4 Ash Content

A lower value of incombustible matter generally means a lower propensity to spontaneous combustion. For a given coal the higher the ash content, the lower the R70 value.
However, if it is a reactive ash (mineral matter) such as pyritic or carbonate it can actually enhance the reactivity.

4.4.5 Coal Particle Size

During the mining process, coal is broken into fragments. As the coal breaks, the surface area is greatly increased; more coal surfaces are exposed to oxygen for oxidation to occur, and therefore an increased risk of spontaneous combustion. Common areas where broken coal may be found include:

- Around crushed ribs or pillars.
- Around seals and stoppings.
- In stowage.
- Goaves.
- Around conveyors.
- Floor heave.
- Faults and intrusions

4.4.6 Permeability

Highly permeable coals introduce other potential spontaneous combustion risks such as leakage paths through coal around gas drainage boreholes, seals, stoppings and even through pillars. Permeability may have a direction bias, i.e. be higher in some directions rather than others.

4.4.7 Effects of Moisture & Water

All coal has inherent moisture, the amount depending upon its rank. Coal seam moisture content also varies with the permeability of the seam and the degree of saturation. If moisture drains from a seam vacant space will be filled with gases.

Moisture can either remove heat or add heat, depending on the mine conditions. Evaporation of water from the surface requires heat – the latent heat of vaporisation. This heat is taken from the solid surface, cooling results, and causes a drop in temperature on the coal surface. This is the same principle as the cooling of a water bag.

On the other hand, if water vapour condenses on the coal surface, the heat of condensation (the opposite of vaporisation) causes a rise in coal surface temperature.

In stockpiles, the effect of rainfall may be to wash fines out of the coal. Pumping water into a goaf area, or other area where there may be broken coal has a similar effect. Ceasing to pump the water, and subsequently drying out, creates air leakage paths. Spontaneous combustion may be further facilitated by the heat released during the absorption of water into the coal. (This is thought to have been a significant factor in the development of a heating in the Great Northern seam at Wallarah Colliery)

A substantial make of water in a mine also serves to prevent and control heatings by covering broken coal in the goaf, and by reducing the gas volume in sealed areas, and thus reducing the ingress of air as a result of 'breathing' of the seals in response to barometric pressure changes.

Water build-up on seals may lead to increased pressure on the seals and the deterioration of the seals in the worst case, a breach of a seal may result in inrush of water or gas into the workings of the mine.

The effect of water accumulation in roadways is to increase ventilation resistance. This increase in resistance along the planned ventilation path facilitates the formation of new leakage paths in unwanted areas.

4.4.8 Seam Gas

Spontaneous combustion has occurred in mines with a seam gas and also mines without a seam gas. The consequences of spontaneous combustion in a mine with a flammable seam gas may be catastrophic and it is vital to prevent any fire in any mine.

Oxidation of the internal surfaces of coal will normally be delayed by desorption of seam gases into the mine atmosphere. This situation will tend to prevail as long as the seam gas pressure exceeds the mine atmospheric pressure. Desorbed seam gases may be used to develop inert atmospheres in sealed-off areas in the mine.

A major factor in reducing the risk of spontaneous combustion is the presence of moderate to high makes of seam gases. As the early stages of development of spontaneous combustion are highly sensitive to the availability of oxygen, any significant make of methane or blackdamp will reduce the chance of oxidation developing into a heating.

Gases will act to exclude oxygen from the area immediately adjacent to the face, and will restrict percolation of air into fractures and cavities. However, where a ventilation circuit exists across a goaf, through a length of fractured roof, or through a failed pillar, airflow will occur, seam gases will be diluted and heatings may occur. Where the gas make is inadequate to fill the waste, a particularly difficult situation may arise wherein a heating develops in broken coal on the floor in the goaf, while the upper section of the goaf is filled with methane, possibly in explosive concentrations.

The type of atmosphere that develops in a goaf and the explosive ranges of the various gases should be considered. Explosive gases including carbon monoxide, hydrogen and methane are produced by spontaneous combustion. The gaseous products of oxidation can create an explosive mixture that may be ignited by a heating.

4.4.9 Gas Drainage

Generally it appears that pre-drainage of gas from the coal may increase the likelihood of spontaneous combustion occurring. Gas drainage using negative pressures may contribute to the development of a heating by promoting a flow of air into a permeable coal seam or a mined out area. Ideally the rate of gas extraction should not exceed the desorption rate.

The hazard exists when the oxygen level in the drained mixture rises above 8%. Sampling of the drained gas should be practised to reveal any entry of oxygen and to determine the levels of carbon monoxide within the system.

Pre-drainage of the coal seam also removes much of the free water. The water is drawn out of the drainage holes with the gas flow. Drying the coal makes it more powdery which increases the dust generated during mining. This produces dry, more finely divided coal dust that may settle in the goaf. This increases the reactivity of the coal and consequently the likelihood of heatings. The removal of the water from the seam also increases the permeability of the goaf and increases the ingress of air into the seam.

4.4.10 Seam Thickness & Coal Recovery

The thicker the coal seam, the greater the area of coal surface exposed to oxidation and the more liable it is to spontaneous combustion.

Where the coal seam is too thick to be mined in one lift, top or bottom coal may be left in the goaf. This broken coal is prone to oxidation and heating.

For a given production rate, the thicker the extracted section, the slower the rate of face retreat and the greater the time available for oxidation of coal left in the goaf.

The volume of spalled and fractured coal along the sides of roadways and coal ribs increases with seam thickness which increases the potential for spontaneous combustion.

No mining system can guarantee total recovery of coal. Some remnant coal will be left in a goaf and may be liable to heating. The risk of heating in a goaf can be reduced by full seam extraction. Where this is not practical, mining the upper part of the seam, can reduce the amount of broken coal. If the bottom part of the seam is left in the goaf, heave and cracking of bottoms can still occur.

A risk may arise due to the crushing of longwall chain pillars. Airflow along the pillar edge may create conditions for a heating to develop. In continuous miner extraction areas, the extraction edge is less regular and ventilating pressure differentials in the face are usually lower. These factors result in a decreased risk of a heating. Should a heating be detected equipment can be more easily removed and the area sealed.

4.4.11 Multiple Seams

A seam split or another seam (above or below) may be exposed by the mining process and broken coal may be exposed to oxygen. These seams are often of poorer quality or not thick enough to be commercially mined, and yet may be the source of gas and broken coal in the goaf.

Where there are a number of overlying seams within a lease, there may be a risk of interconnectivity between workings and air movement between seams. Air movement will depend upon permeability and pressure difference.

Seams that are worked in close proximity are most at risk. The interval between seams that constitutes a risk is dependent upon the extraction thickness and other geotechnical factors.

Adjacent seams may be a hazard due to their propensity for spontaneous combustion and location etc.

4.4.12 Structures & Geological Anomalies

Structures such as faults, dykes, and open joints may be associated with zones of weakness that require extra support and reduce the rate of extraction. The slower the rate of extraction, the longer a particular area of coal is exposed to air. This increases the potential for spontaneous combustion. The probability of roof bed separation and cavities is increased with accompanying low airflow through the fractured strata. Roadside and rib spalling tend to increase in structure zones and may further increase the likelihood of spontaneous combustion.

Where a structure zone passes through a roadside or barrier pillar that is subjected to the differential ventilation pressure between intake and return, extra care must be taken to ensure that air leakage paths do not develop and increase the risk of spontaneous combustion.

Faults and intrusions become focal zones of increased stress and may require special attention.

4.4.13 Depth of Cover

The effect of seam depth is somewhat contradictory. Increasing depth will tend to reduce seam permeability due to increased pressure of the strata. Increased depth will mean that higher loads are redistributed to pillars and coal ribs, tending to increase fracturing and spalling of coal and therefore increasing the likelihood of spontaneous combustion.

Where extraction takes places at relatively shallow depths, interconnection between the surface cracks and above seam caving cracks or voids is possible with sufficient permeability to permit air circulation from the surface into the extracted area and mine roadways. Extraction thickness and strata types have an influence on interconnection.

There is a recorded case of a heating in a panel extracted by longwall where the depth of cover was 110m and the extracted seam thickness 3.8m. Steps should be taken to ensure closure of these cracks and to control these leakage paths.

4.4.14 Direction of Mining

The direction of mining and seam dip may affect the ability to efficiently inertise a goaf on the basis that ventilation problems are compounded (or created) by buoyancy effects in the goaf or along the roof in workings. If methane is the predominant gas given off in the goaf, mining to the dip will result in the buoyancy of methane causing it to flow to the upper end (start) providing efficient inertisation.

If the direction of mining is to the rise, methane may migrate towards the face and result in poor inertisation of the deeper areas of the goaf and an unwanted concentration of methane in the working area. On the other hand it may be considered beneficial to bring the gas fringe nearer to the face and into the location of maximum risk of oxidation and heating.

If carbon dioxide is the predominant gas given off in the goaf, this migrates to the dip side, providing effective inertisation if mining advances to the rise, and less effective inertisation if mining advances to the dip.

4.4.15 Extraction Systems

Full extraction and super critical systems leave less coal behind in the goaf, resulting in complete caving and consolidated areas where there is a low risk of spontaneous combustion. Longwall mining results in more complete extraction than continuous miner extraction.

Partial extraction may result in more coal and voids in the extracted area with a risk of spontaneous combustion in what would have been a consolidated zone if full extraction was practised.

Subcritical extraction systems designed to control surface subsidence may result in voids in the strata above the seam. Coal at this horizon may be at risk of spontaneous combustion. Tables 2 and 3 detail the relative risk of heating for various extraction systems.

Ventilation flow and inertisation of the extracted area are important factors.

The relative risk of spontaneous combustion for various mining systems is shown in the following tables:

Table 2: Relative Risk of Heatings - Continuous Mining Methods

Solid development (full seam thickness in one pass)	LESS RISK
Solid development (tops of thick seam)	
Solid development (middle or bottom of thick seam)	↓
Shortwall extraction (full seam thickness in one pass)	
Shortwall extraction (tops of thick seam)	
Shortwall extraction (middle or bottom of thick seam)	
Wongawilli extraction (full seam thickness in one pass)	
Wongawilli extraction (tops of thick seam)	
Wongawilli extraction (middle or bottom of thick seam)	
Pillar extraction by split and lift (full seam thickness in one pass)	
Pillar extraction by split and lift (tops of thick seam)	
Pillar extraction by split and lift (middle or bottom of thick seam)	
Wongawilli extraction in descending lifts (thick seam)	
Wongawilli system (top coal from bottom development)	
Partial extraction of pillars (full seam thickness in one pass)	
Partial extraction of pillars (middle or bottom of thick seam)	
Loading of top coal in thick seam after continuous miner development	
Random pillar extraction (full seam thickness in one pass)	MORE RISK
Random pillar extraction (middle or bottom of thick seam)	

Table 3: Relative Risk of Heatings - Longwall Methods *

Retreat Mining (full seam thickness in one pass)	LESS RISK
Retreat Mining (in tops of thick seam)	
Retreat Mining (in middle of thick seam)	↓
Retreat Mining (in bottom of thick seam)	
Retreat Short Longwall (full seam thickness in one pass)	
Retreat Short Longwall (in tops of thick seam)	
Retreat Short Longwall (in middle of thick seam)	MORE RISK
Retreat Short Longwall (in bottom of thick seam)	

* Risk will increase with wider longwalls due to relative rates of retreat

In longwall workings there are voids along the edges of the goaf adjacent to ventilated roadways where fresh air can intrude and flow if the goaf is not efficiently contained and inertised.

Areas where voids in the goaf exist and the likelihood of the development of heatings increase are:

- Face Start Line - air may percolate into the original face start line. This may be due to the high standard of support in this installation roadway.
- Face Finish Line - This is controlled by rapid salvage of face equipment and by design of adequate final barrier pillars. The risk may arise if face recovery is delayed by equipment failures, industrial action or mining conditions.

4.4.16 Highwalls & Box Cuts

Punch or highwall mining may increase the risks of spontaneous combustion developing due to the effects of open cut blasting, pre-splitting and endwall stress effects, and subsequent highwall slumping, causing mining induced fracturing.

These effects may exacerbate any pre-existing geological anomalies in the near vicinity of the highwall, such as faults, joints, and cleat. The results are potential air leakage paths in the near highwall area from either highwall face, or surface, or intake to return, airways.

4.4.17 Ventilation Pressure Difference

A pressure difference between two areas in a mine will cause air to flow from the higher to the lower pressure area. The amount of air that flows along each path depends upon the resistance to flow. This can result in unplanned ventilation flows and air leakage.

High ventilation quantities and pressure differential may result in air leakage into or from a sealed area or through or around pillars which will increase the risk of spontaneous combustion. A good example of this is the pressure difference between an active longwall face and the ventilated gate road alongside the goaf of the active longwall, especially if it is a return.

4.4.18 Abutment Load & Pillar Crush

Excessive pillar yield may result in air being able to be drawn through the pillar by ventilation pressure differential.

Yielding pillars or 'sacrificial roadways' may be designed in order to prevent heave in the maingate and to improve conditions at the face end. These should not be used in areas with a moderate to high propensity for spontaneous combustion unless adequate investigation and design work is carried out on the ventilation aspects of the design.

Pillar heatings have been encountered between intake and return airways, often near pit bottom. They are normally associated with crushing of pillars or other mechanisms such as high ventilation pressure difference and open joints that create potential flow paths.

The abutment load from extraction areas will be carried on surrounding pillars. Roadway convergence near stoppings may cause leakage to occur and result in poor inertisation of the goaf.

4.4.19 Reduced Extraction Rate &/or Unplanned Disruption

Continuance of a rapid rate of retreat ensuring that coal in the goaf is sealed or immersed in an inert atmosphere before accelerated oxidation occurs, is an effective means of preventing spontaneous combustion in a goaf.

An unplanned disruption to mining, or significantly reduced extraction rate could result in an increased risk of spontaneous combustion. These events can occur due to geological & geotechnical factors, industrial action and slowness in moving a longwall after panel completion.

Means to increase the rate of retreat include:

- Reschedule planned maintenance
- Operate weekend shifts that may not be planned for production

4.4.20 Extracted Areas

Goaves are a potential source of heating if not sealed. Ventilation may be such that the oxygen supply is adequate to promote oxidation but the cooling effect is inadequate to prevent heating.

Where goaves are sealed, there may be a risk of a heating where there is leakage of air through or around seals, and a high pressure differential exists. There will be no effect in the deep-seated regions but areas near sealing sites may be continually supplied with oxygen from barometric pressure fluctuations.

4.4.21 Barometric Variations

Flow into sealed areas results not only from differences in the mine ventilating pressure, but also from the large volumes of such areas, affected by barometric changes creating inflow and outflows.

Barometric changes may range from about 960 to 1040 millibars and rapid changes in barometric pressure can occur as a result of storm activity. This represents a change in absolute pressure of 8Kpa, which is significantly greater that the pressure differential at which mine fans typically operate.

The rate of change has the greatest influence. This pressure differential acts on the seals in a mine. Well-constructed seals and surrounds can act to restrict the volume flowing and retard the changes in sealed areas by reducing the effects of peaks and troughs in the barometric pressure fluctuation.

4.4.22 Integrity of Stoppings & Seals

Stoppings and seals that allow significant leakage will prevent efficient inertisation of the goaf. Matters to be considered in the design of goaf edge stoppings and seals are detailed in 5.7.

4.4.23 Boreholes & Wells

Unsealed boreholes or wells may provide a conduit for air to flow from the surface into a goaf area, or allow the atmosphere in the goaf to flow to the surface.

Boreholes placed in areas affected by subsidence pose a risk, even if not drilled all the way to the coal seam.

4.4.24 Accessibility of Roadways Adjacent to Goaf for Inspection

If the roadway adjacent to the stopping or seal becomes inaccessible there is the risk of a damaged and leaking stopping and the non-detection of a heating. If a stopping or seal is to be relied upon to contain and inertise a goaf, then its integrity should be able to be confirmed by periodic inspection.

4.4.25 Integrity & Effectiveness of Monitoring System

Detection of increasing levels of oxygen in goaves and early signs of heating will rely upon effective monitoring and inspection systems. The location and number of monitoring points is critical to the effectiveness of the system.

If the monitoring and inspection system is not properly designed, implemented and maintained, there is a risk that spontaneous combustion will not be detected until a serious problem develops.

Sample turnaround time is a factor. If the information is not provided promptly, decisions and corrective action may be delayed.

4.4.26 Reporting & Tracking of Information

Recording of results of inspections and historical data is important as is the review and action as a response to this information. Information recorded should be specific in terms of the time, location, quantities etc. if it is to be of value.

Inspections missed, or inadequate interpretation of information is a hazard.

Integral to a comprehensive and effective reporting system is an audit & review process to provide checks and balances in the mine's stated controls for spontaneous combustion.

4.4.27 Sample Turn-around Time

The time taken to analyse and interpret a sample taken from and heating site has delayed decision making in a number of spontaneous combustion events (refer to "Events" in the Appendices). This is a factor where the analysis has to be carried out remote from the mine site.

4.4.28 Surface access

Access may be required to surface areas of the mine for the purpose of monitoring, or remedial action in the event of a heating. This needs to be considered for an effective and prompt response to an incident.

5 PREVENTION

5.1 MINE PLANNING PROCESSES

5.1.1 Research & Collection of Information

The following information should be collected, evaluated and considered in the development of a spontaneous combustion hazard management plan.

- A comprehensive mine plan showing seam contour, seam dips and water collection areas should be collected as an aid to spontaneous combustion management and gas behaviour.
- A detailed and precise description of the mining method and any “recent” changes to the mining method should be made. The presence and amount of slack coal in the goaf should be determined as this material may alter the spontaneous combustion risk. Rib spall and other signs of broken or failed strata containing coal or carbonaceous shale should be noted. Regional stress and depth of cover should be noted.
- A comprehensive ventilation plan showing all aspects of the ventilation system should be collected. Ventilation quantities should be determined. Pressure differentials should be noted, particularly across and around sealed areas. Gas composition in ventilation currents should be noted and any changes monitored.
- The progressive history of seam gas (and gas from other sources) should be collected and collated. The desorbable gas content of the seam should be determined. Gas sources from the roof and floor following the breaking of surrounding strata should be noted.
- The development and nature of atmospheres within sealed areas of the mine should be determined. These measurements should apply to panels where spontaneous combustion has developed and also to those panels where it did not. Comparison of results may be vital in understanding how, why and where events may occur.
- The nature of gases in existing sealed area atmospheres should be taken, any changes in these should be monitored.
- An assessment of the development of explosive atmospheres within the mine should be made. This would include ventilation currents, goaf areas and sealed areas.
- Trending of all gas measurements should be conducted.
- The condition of all seals to extracted areas should be determined, the method and materials of construction should also be noted. The effectiveness of sealing should be determined or estimated. The location, depth, diameter and condition of Boreholes to the surface including the nature of sealing and/or capping. The incidence of subsidence cracking should be noted.

5.1.2 Impact of Other Mine Site Hazards

The Spontaneous Combustion hazard must be managed in conjunction with other hazards at the mine. Optimum prevention measures may not be able to be implemented because of the need to control other hazards and adapt to other constraints.

Examples of hazards that need to be managed and constraints that require special consideration are:

- Requirement for gas drainage to control outburst or high gas levels
- Working of multiple seams, particularly those within close proximity
- Need to control surface subsidence and limit extraction width
- Shallow workings
- Working of thin seams or seams remote from entries that require higher ventilation pressures
- Presence of seam structures
- Size and shape of coal lease

The process for mine planning should focus on the need for management of all hazards together with the provisions that need to be implemented to control spontaneous combustion in the circumstances at the particular mine.

5.1.3 Measures to Prevent Spontaneous Combustion

Mine planning processes to prevent spontaneous combustion include:

- Type of extraction & percentage extraction for each type
- Extraction thickness to be mined from the seams
- For continuous miner extraction panels, design to provide barriers on both sides.
- For longwall panels, reducing the number of goaf entries to be contained.
- Extraction of as much of the coal seam as possible
- Consideration of the risks of permeable subsidence cracks when mining under shallow depth of cover
- Minimum number of entries to panels with provision for rapid sealing
- Controls on stowage of material in roadways
- Controls on accessibility of roadways alongside goaf areas
- Eliminating restrictions in roadways to reduce airflow resistance and pressure difference
- Maintenance & quality control of ventilation structures and operation
- Ventilation system & pressure difference in various locations
- Systems to balance pressure on seals where required
- Avoidance of main intake and return entries in proximity to a box cut or entries in shallow cover
- Control of mine water removal and placement
- Provision for inertisation of extracted areas

5.2 MINE DESIGN

5.2.1 Behaviour of the Atmosphere in the Longwall Goaf

As the active longwall panel retreats, oxidation takes place in the area behind the face until the goaf is inertised by containment, seam gases and goaf consolidation. The distance into the goaf from the longwall face in which oxidation takes place varies and depends upon these factors and particularly barometric changes. Frequent and significant changes in the barometer can pump fresh air a considerable distance back into the goaf.

In the oxidation area, CO and CO_2 will be produced. At the inbye end of the goaf where inertisation is effective, CO should not be detected.

The ACARP Project report C12020, "Proactive Inertisation Strategies and Technology Development" - Rao Balusu, Ting X Ren and Patrick Humphries Dec. 2005 contained a Computational Fluid Dynamics (CFD) base model of the atmosphere within a longwall goaf prior to any inertisation. A number of scenarios were modelled with varying results.

Figure 7 shows one scenario; the goaf atmospheric oxygen in a longwall with a face ventilation flow of $50m^3/s$, MG intake 20m lower than the TG return and goaf gas emissions of 600 l/s. There does not appear to be any allowance for barometric variations.

Results show that oxygen ingress into the goaf was high with oxygen levels on the maingate side well over 14% at 350 m behind the face and over 10% even at 600 m behind the longwall face.

Analysis showed that intake airflow and ventilation pressures seem to have a major influence on gas distribution up to 50m to 150 m behind the face, and beyond that, goaf gases buoyancy seems to play a major role on goaf gas distribution.

Figure 7: CFD Model of Goaf Atmosphere

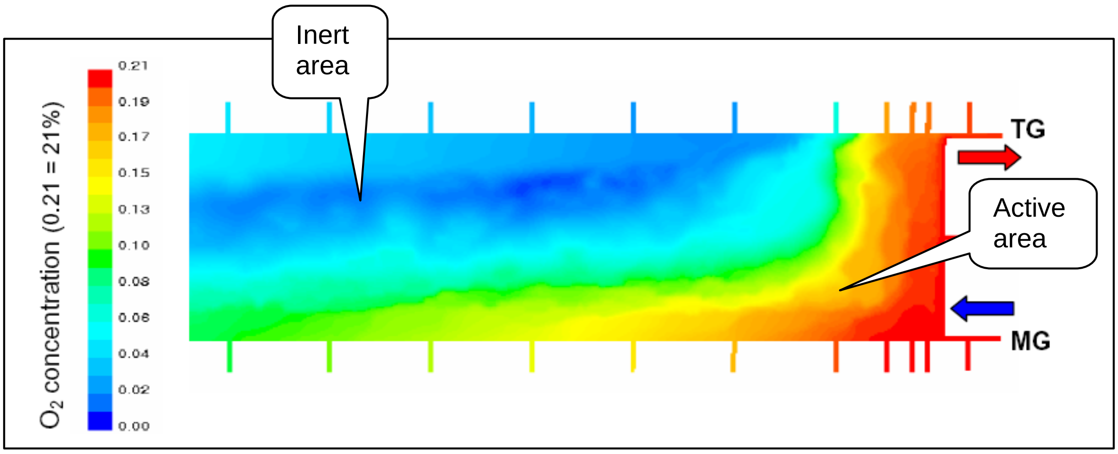
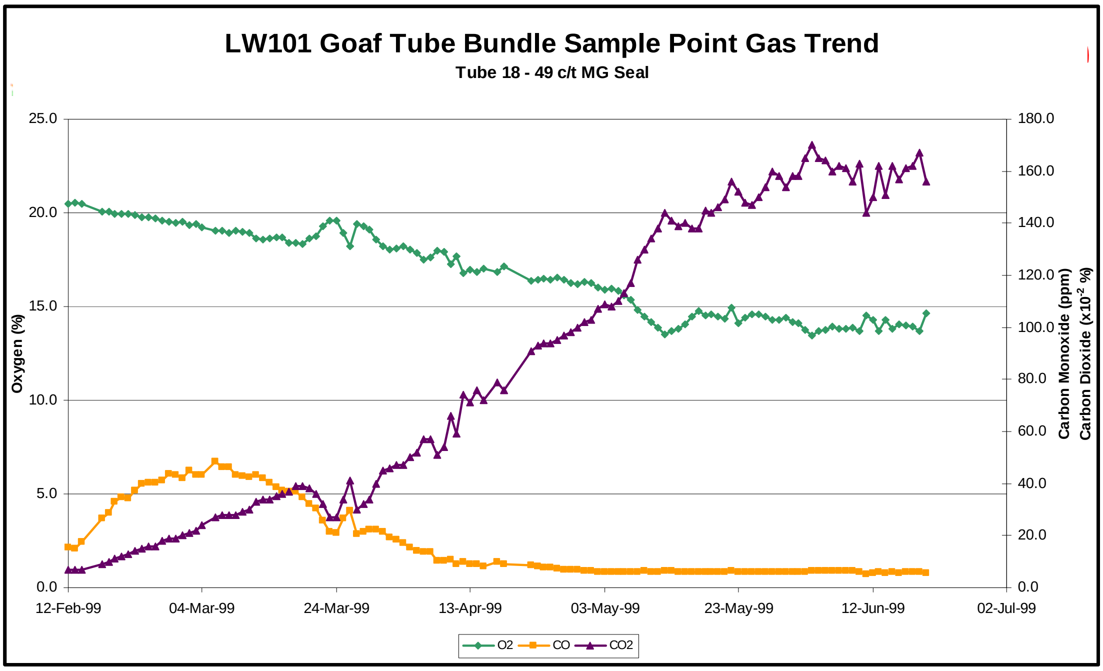


Figure 8 shows the changing goaf atmosphere as the longwall retreats away from a fixed sampling point and demonstrates that the atmosphere eventually becomes inert. Oxygen reduces, carbon monoxide initially increases due to oxidation, and then reduces as the face retreats and oxygen decreases.

Figure 8: Goaf atmosphere analysis as Longwall retreats



5.2.2 Control Measures for Longwall Extraction

Control measures in longwall extraction include:

- Enclosing the goaf as the longwall retreats with effective seals.
- Minimising pressure differential across the goaf.
- Maintaining a constant rate of longwall retreat.
- Prompt recovery of longwall equipment at panel completion.
- Monitoring of longwall tailgate goaf atmosphere as the longwall retreats.
- Monitoring of goaf atmosphere adjacent to the ventilated roadways
- Inspection of seals, longwall return and goaf edges.

In an active longwall, the goaf alongside the working area cannot be enclosed and there is a risk of spontaneous combustion developing should there be a protracted face delay. Reliance on the incubation period and rate of retreat is the normal control.

The time taken for a heating to develop (incubation period) is unpredictable and variable. It depends upon factors such as the properties of the coal and environmental conditions. This requires consideration of provision for inertisation and rapid sealing in the event of a protracted production delay which results in a heating.

Acceleration of the rate of extraction by extending operating time is a control that has been used for many years in both longwall and continuous miner extraction.

The system shown in Figure 1 is most common for Australian longwalls. Bleeder roads have the advantage of ventilating the future tailgate for the successive longwalls and avoiding the drivage of single entry development for gate roads. They do provide a risk of air passing into the goaf from the adjacent bleeder road if containment and inertisation is not effective.

An option for a longwall mine with a high propensity for spontaneous combustion is to drive single entry roadways either side of the block, or to leave barriers between sets of gateroads.

5.2.3 Control Measures for Bord & Pillar extraction

Similar to a longwall, spontaneous combustion is controlled by:

- complete caving,
- effective inertisation,
- a regular and progressive extraction rate,
- minimisation of pressure differentials across sealed areas,
- inspection and maintenance of seals and seal sites to reduce air leakage, and
- sampling and analysis of sealed area atmospheres.

Figure 2 shows a most effective system of containment of the goaf. The panels are of such a size that extraction will proceed reasonably quickly and there are only three entries into the extracted area that will need to be sealed on panel completion. This would be appropriate where there is a high propensity to spontaneous combustion.

In some circumstances it may not be possible to plan continuous miner extraction panels as in figure 3. There may be pillars already formed from earlier workings or there may be other constraints on mine planning. Large areas of pillars can be reduced into manageable panels by placing stoppings such that the panel width is reduced and there are a minimum number of entries to seal off on completion, or prior to completion in the event of a heating.

Figure 3 shows a method of dividing a large area into a number of smaller panels that are capable of being isolated quickly in the event of onset of spontaneous combustion. A number of variations to this theme are possible. The extent to which panels need to be reduced in size and barriers provided for isolation depends upon the propensity for spontaneous combustion and the efficiency of Inertisation.

5.3 MULTIPLE SEAMS

Where overlying seams lie within the influence of mining, migration of air from one seam to another and into sealed areas may cause spontaneous combustion. Balancing of pressure between seams and sealing of strata cracks are controls (5.5.1 & 5.5.2). Flooding of lower seams is most effective.

5.4 VENTILATION SYSTEM

Ventilation design tools to prevent spontaneous combustion include:

- Reducing mine resistance and ventilation pressure
- Providing a high standard of stoppings and seals & roadway support
- Using low flow/ low pressure drop roadways alongside goaf areas
- Using balance chambers to contain sealed areas
- Injecting inert gases into balance chambers

▪ Providing for inertisation or flooding of goaf areas

There should be a process in place for reviewing the potential impact on the spontaneous combustion risk prior to significant ventilation changes being implemented.
Ventilation systems that minimise pressure differentials across goaves or waste workings and along roadways adjacent to a goaf reduce the spontaneous combustion risk.

The simple ‘U’ system of ventilation is considered to be the most effective in preventing spontaneous combustion in longwall workings. However, this has a disadvantage in the return airflow passing alongside the adjacent goaf. Leakage through stoppings in this area will generate an induced airflow through the goaf, which may lead to a heating.

5.4.1 Pressure Difference

Placing values on the pressure difference across a goaf and along a roadway adjacent to a goaf is important in setting a standard for the mine that reduces the risk of spontaneous combustion.

A low pressure difference across a goaf can significantly reduce leakage through seals and stoppings. Setting values is important to establish a standard for the mine.

Values are dependent upon the circumstances in the mine and may include:

- Extracted seam thickness
- Length of longwall block
- Number of access roadways
- Standard of stoppings and seals
- Gas make in the goaf
- Pillar and seal stress environment/regime

Reducing the pressure difference is not the only control adopted for prevention of spontaneous combustion. The impact of barometric variations is significant and values most often exceed the pressure difference across a goaf. Reliance on minimising leakage is dependent upon the standards of seals.

Blakefield South mine in the Hunter Valley area of NSW has adopted an innovate approach to the control of spontaneous combustion and the risk of air leakage from the surface to seams above and within the currently mined seam.

The mine’s primary ventilation system incorporates a pressure balancing (force-exhaust) ventilation system. Refer to Figure 9. The “neutral point’ and pressure difference between seams and the surface are controlled by varying the duty of the forcing and exhausting fans.

Figure 9: Pressure drop and neutral point for force-exhaust system

5.4.2 Balancing Pressure

Balancing the pressure across a group of seals enclosing a goaf is an effective technique to minimise air movement in and out of the goaf. Techniques include balance roadways and balance chambers.

The need to balance pressure can arise where there are a large number of seals enclosing the goaf and/or there is high airflow and significant pressure differences in the ventilated roadway adjacent to the seals.

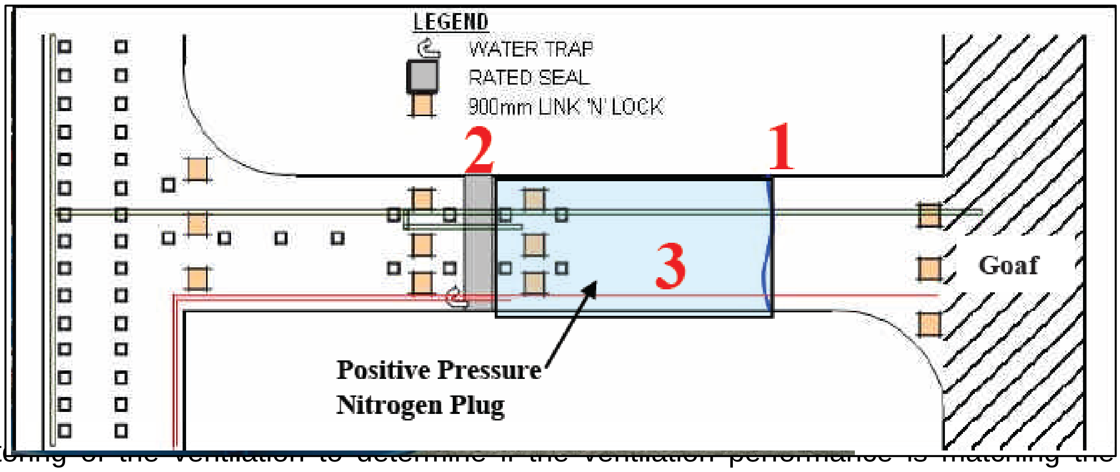
A dedicated roadway alongside the goaf with a low airflow and low pressure differential will assist to balance pressures across a number of goaf seals.

Balance chambers can be constructed by placing two seals in the roadway alongside the goaf. Chambers can be balanced with one another by means of the following:

- A pipe line connected into each chamber
- Surface to chamber boreholes
- Pressurising each chamber with a vent fan
- Pressurising each chamber with an inert gas.

Austar Coal mine in NSW makes use of a technique that effectively prevents leakage of air into the goaf as detailed in Figure 10. The goaf is contained by constructing two stoppings in each access roadway. The area between the two stoppings is pressured by piping inert gas from a unit located on the surface.

Figure 10: Goaf seal arrangement with balance chamber



5.4.3

Monitoring of the ventilation to determine if the ventilation performance is matching the design intent is good practice.

Monitoring of pressure differentials is particularly important in detecting changes in roadway resistance, goaf containment and air flow into goaves. Routine monitoring of pressure differentials is recommended.

Readings should be recorded and trended to identify issues of concern. Consideration should be given to inclusion of ventilation measures in TARPs.
Real time recording of ventilation quantity in a Control Room is of value in determining CO make.

5.4.4 Seam Gas emission management

High levels of seam gas may require large air quantities to dilute the gas thus generating significant pressure drops across the face and increased air ingress into the goaf.

5.5 INSPECTION & MONITORING

An effective inspection and monitoring system will detect early variations from the planned ventilation and gas management values and assist in the prevention of spontaneous combustion.

5.5.1 Access to Stoppings and Seals

Where there are stoppings and seals containing a goaf they should be inspected regularly in accordance with a plan. The sites may also be required to sample the atmosphere in the sealed area behind the seal.

Roadways adjacent to seals and stoppings and cut-throughs where seals are placed should be kept in an accessible condition at all times Figure 11.

A fall in the access roadway or the dumping of material in the roadway can result in the following consequences:

- Damage to seal or convergence not discovered
- Inability to sample atmosphere behind seal
- Inability to transport materials to site for repair of seal or roadway
- High resistance in roadway causing air movement into goaf
- Development of spontaneous combustion

5.5.2 Monitoring of Seals

A register of seals should be kept with details for each seal of:

- Seal type and date constructed
- Secondary support placed both sides of the seal
- Sample pipe, water traps & injection pipes placed in the seal
- Seal condition on inspection & date
- Roadway condition on inspection & date
- Evidence of abnormal leakage
- Evidence of water build up
- Repairs affected

Seals should be regularly inspected and samples taken of the atmosphere behind the seals on a periodic basis. The interval should be based on past performance and risk.

Figure 11: Stowage against goaf stopping impeding access

5.6 VENTILATION CONTROL DEVICES (VCD's)

All VCD's will leak because of the limitations on seal construction, coal permeability, convergence, and the pressure difference created by the mine ventilation system or fluctuations in the barometric pressure. Installation of ventilation structures to a high standard with measures to minimise roadway convergence will reduce the leakage and provide for effective natural goaf inertisation.

Matters to be considered in the design of goaf edge stoppings and seals include:

- The environment in which stoppings and seals are placed
- The permeability of the coal seam
- Designing chain pillars to be stable and not subject to excessive spall
- Overpressure resistance
- The reduction of gas leakage from within the extracted area
- Stopping or seal construction type
- The location of ventilation structures in a stable area (mid pillar)
- Support of the stopping or seal site to minimise convergence due to abutment loads
- Provision for sample pipes and inertisation
- Protection against water build up behind the seal
- Inspection to confirm stopping & seal integrity.

A number of stopping types are available for use. Some types are stiff and can resist strata movement but are susceptible to cracking. This results in air leakage through the stopping. Other stopping types will yield without cracking but allow strata movement with resulting air leakage through the strata.

A common issue with some stopping types is poor sealing and adhesion between the stopping and the roof, sides or floor because of the thickness of the stopping and irregularities in the roadway profile. An example of a stopping type that deals with this issue is the "Micon". These stoppings have a polyurethane core between two layers of cement blocks which penetrate fissures and the brick joints which provides an excellent bond between roof, ribs and floor. Micon stoppings are able to withstand considerable convergence without cracking, but are not "stiff" enough to control roadway convergence. Figure 12 shows convergence in a gateroad. Secondary support may be required. Active support is better than passive support.

If goaf stoppings and seals are constructed to a high standard, leakage of gas into the adjacent bleeder roadway will be reduced even with significant barometric change. This allows the ventilation quantity required to dilute the gas in the adjacent roadway to be reduced. A reduction in air quantity results in a reduction in ventilating pressure along the roadway and across the goaf and the risk of spontaneous combustion. It is good practice when constructing a seal to first inject the surrounding strata with a quality sealant.

In many cases it is not leakage through the stopping or seal that is the problem but the leakage around the sides, under and over the VCD. The coal in these zones is usually fractured and when a pressure differential is applied across the cracks, cleat planes, floor heave or roof convergence air will pass into or out of the sealed area or between intake and return.

If the rate of leakage is unacceptable roadway convergence and fracturing can be corrected by:

- Additional support
- Strata injection or
- Strata sealing

Strata injection is usually more effective than strata sealing.

Figure 12: Roadway convergence

The above photograph shows a roadway that has undergone convergence, been re-supported and is now stable. There will obviously be voids in the roof and ribs, resulting from the convergence, that allow the movement of a significant quantity of air across a stopping or seal placed in the roadway. Strata injection or sealing of the strata is necessary in these circumstances to provide effective containment.

Because seals (other than water seals) cannot prevent air flowing into a sealed area certain precautions must be taken when siting and constructing seals. Seals should be sited only in areas of unbroken coal where limited surface area will be presented to the airflow. As the rate of oxidation is related to surface area this will decrease the risk of heating.

Where coal is broken and no other suitable site is available, strata around the stoppings should be grouted to seal cracks and fractures. Seals should also be sited in large pillars or in solid coal to ensure that airflow cannot occur through a pillar. Coal seams with high permeability and roadways with convergence require special consideration for the design of seals and seal sites.

5.7 CONTAINMENT & INERTISATION

Natural inertisation of the atmosphere within the goaf takes place through oxidation of carbonaceous material and displacement by seam and strata gases. The oxygen content should be 2% or less for oxidation to cease.

In seams where the coal is highly reactive and/or there is liberation of significant quantities of gas into the goaf from the strata, remnants of seam mined or seams above or below, inertisation can occur quickly. Where the coal is not very reactive and there is little or no seam gas, this process may take some time.

The natural inertisation process can be assisted by adding inert gas such as nitrogen or carbon dioxide, or by making use of methane that is rendered inert because of its concentration as further discussed in Section 7.4.

Inertisation of an extracted area requires stoppings and seals to be placed in access roadways to contain the inert gases and to prevent other sources of air ingress from above and below the seam. Sources where air ingress may take place include:

- Workings above and below the seam, particularly extracted areas where strata may be disturbed and cracked due to subsidence effects.
- Uncapped surface to seam boreholes
- Exploration or service boreholes that may be capped but are within the subsidence affected zone
- Water bores
- Shallow workings and subsidence cracks

In an active extraction panel, containment of the goaf alongside the working area is not possible. Reliance is placed upon the incubation period to prevent the risk of spontaneous combustion. The rate of advance of the extraction unit and the development of caving is usually enough to prevent the development of heatings. Additional controls may include monitoring and provision for inertisation.

Extraction systems that use partial ventilation through the goaf should be avoided. Ventilation of all parts of the goaf is difficult to achieve and can't be verified.

Where gas make in a seam is very high and attempts at containment result in an unacceptable increase in gas in surrounding roadways, consideration may be given to the release of the gas to reduce the pressure within the goaf. Gas wells placed near the edges of the extracted area are a viable option. The risk of spontaneous combustion must be considered in conjunction with other major hazards at the mine and a total systems approach adopted.

The best form of inertisation of the extracted area is flooding. This excludes oxygen and cools any incipient heating. Other inertisation methods reduce oxygen but do not cool the heating site.

An option to reduce goaf void space is the introduction of water, inertisation gases, and fly ash slurries or washery slimes.

5.8 SEGREGATION OF PARTS OF THE MINE

In a mine that is prone to spontaneous combustion, extracted areas should be segregated so that they are of manageable size. Consideration should be given to limiting the length of longwall panels, the number of openings into the goaf and the number of successive long wall panels to avoid large numbers of stoppings being required to contain the goaf. The number of successive extraction panels could be reduced by leaving barriers periodically.

Even with a high standard of stoppings and seals, there is a limit to the size of the containment area. If a large number of stoppings and seals are relied upon to contain the area and allow it to inertise, there may be difficulty in lowering the percentage of oxygen in the sealed area to safe levels. Even seals constructed to a high standard will leak. The combined leakage from a large number of seals may not be offset by the natural inertisation processes. Another reason for segregation is to rapidly seal parts of a mine where a heating may develop.

Mines should assess the need for segregation that is supported by a risk management approach.

5.9 CONTROLS ON STOWAGE

Accumulations of carbonaceous materials in roadways should be avoided. Such accumulations may come from fallen top coal, dumped material or convergence in top or bottom coal.

This material is best cleaned up and removed from the mine. Where this is not a practical option, the material can be stowed underground in a manner that controls the risk of spontaneous combustion, i.e. by sealing in specially driven roadways, or by spreading in thin layers along the roadway and compacting.

The dumping of stowage material, carbonaceous or not, against stoppings and seals enclosing the goaf is be avoided. Stowed material in these areas will impede inspection, sampling and repair of stoppings and roadway surrounds.

If stowage must be placed underground and the roadway is not to be sealed, it should be placed in such a manner that it is ventilated and can be inspected easily without a person having to crawl over spoil heaps.

5.10 PILLAR DESIGN

Pillars where stoppings and seals are to be placed to contain a goaf should be designed so that they are stable when subject to abutment loading. Rib spall and convergence make it very difficult for stoppings and seals to provide effective containment. Properly supported roadways and stable pillars are important.

The design of pillars involves the selection of dimensions and geometries which will ensure that when the pillar is fully loaded, yielded zones on the sides of the pillar remain separated by a competent core of confined, high strength material.

The extent of the yield zone can be controlled by sound pillar design and by the installation of adequate rib support to ensure the yielded material remains partly confined.

Rib side pillar heating risk can be minimised by maximising the separation of intake and return airways near pit bottom, minimising the number of cut-throughs in this area, and by driving multiple roadways for both intake and return airways to minimise the pressure drop along the length of a pillar.

5.11 BOREHOLES & WELLS

Boreholes, if not required for further use, should be filled using a method that ensures the hole is completely filled without air gaps. Details of the driller, process used to fill the hole and company supervision etc. should be recorded.

A register of boreholes, wells, or gas wells placed anywhere in the lease should be kept with the following information:

- Location
- Depth
- Diameter
- Condition as at date
- Purpose
- Capped/ uncapped/ filled
- Driller & drillers log

Boreholes placed within the area of the subsidence effects of a goaf should be monitored to ensure that air is not passing into the goaf. The flow from gas wells may reverse at some stage when gas pressure in the goaf reduces.

6 DETECTION

6.1 IMPORTANCE OF EARLY DETECTION

Early indication of the onset of spontaneous combustion will most often provide time for action to be taken to control the heating before the need for people to be withdrawn from the mine.

There has been debate about the incubation period for spontaneous combustion and its value as a control method. Reliance on a specific incubation period is problematic.

Detection of a heating in the early stage of the incubation period is very difficult. The length of the incubation period will depend upon environmental conditions as well as the properties of the coal. Panels at Moura No.2 were designed to be completed within 6 months as a control against spontaneous combustion. The time taken for development of the heating that led to the explosion was less than this period.

While the oxidation process occurs at relatively low temperatures, a heating may not be detected until the temperature reaches 2 or 3 times the ambient temperature.

While gaseous indicators of spontaneous combustion such as CO and CO_2 are commonly not given off until about 30-45^0 C from some coals, reactive coals may produce large quantities of these gases at similar temperatures.

Experimentally H_2 has been shown to be given off at temperatures below 100^{0}C and C_2H_4 at 100-120^{0}C. These temperatures are approximate and will vary for different coals. The use of modern detection techniques now allows traces of these gases to be detected sooner in the self-heating process.

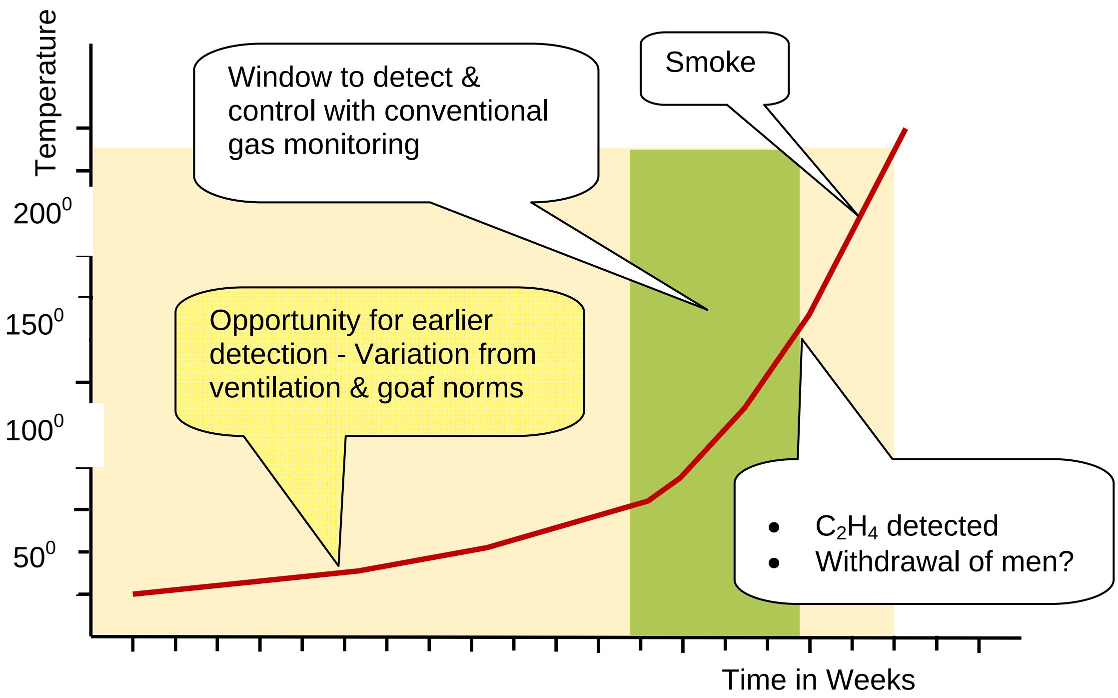
As temperatures exceed those required for the evolution of gases such as ethylene, the situation is rapidly approaching thermal runaway and would be at the stage where the mine goaf spontaneous combustion TARPS will require withdrawal of people.

It is important to realise that all the time taken for a heating to develop to a dangerous stage is not available because there is no way to be certain when the heating has started. Time will elapse before the heating is first detected. Only the time from when the heating is first detected to when men must be withdrawn is available for effective action. This may be weeks or even days depending upon the coal type and environmental conditions.

Early warning techniques include detection of a rise in oxygen in a sealed area and unplanned changes in ventilation pressure and flow. These techniques may provide an opportunity for early warning of spontaneous combustion rather than waiting for the detection of products of combustion.

Figure 13 shows the relatively small window of opportunity to detect and control a heating if reliance is solely upon detection and interpretation of gaseous products of spontaneous combustion.

Figure 13 – Incubation period of a Heating



6.2 GAS EVOLUTION TESTS

Gas evolution tests are useful in determining the behaviour of the coal as it heats and the development of gaseous indicators for early detection and use in TARPS. These tests may be performed using small scale or bulk scale techniques.

The following figures 14, 15 and 16, show the "fire ladder" or hierarchy of development of gases during the development of a heating for several different coals.

Figure 14: Fire ladder for Moura coal

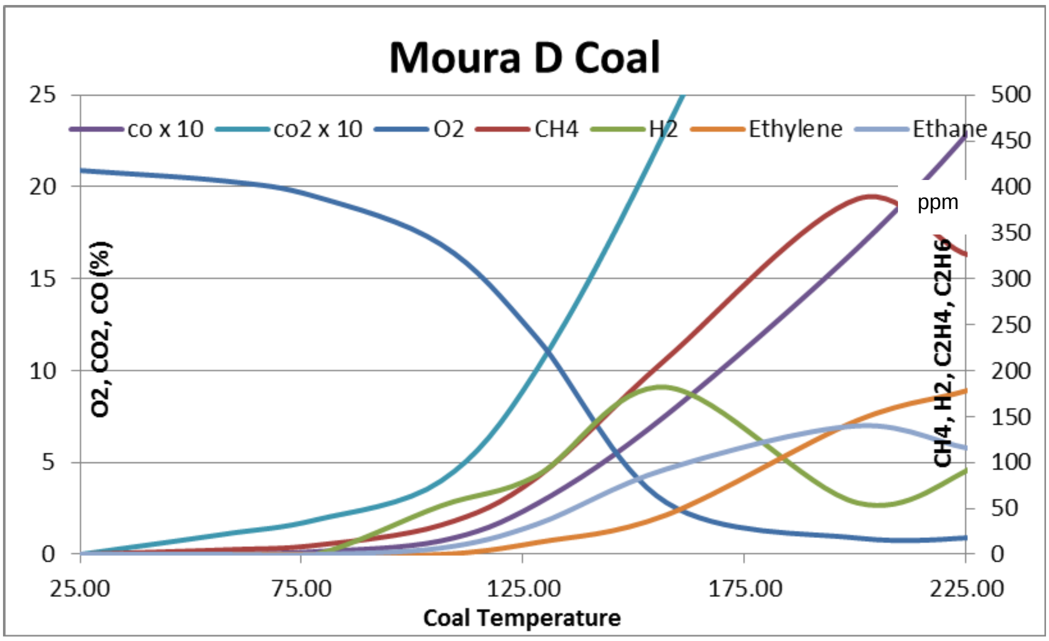


Figure 15: Fire ladder for New Zealand coal

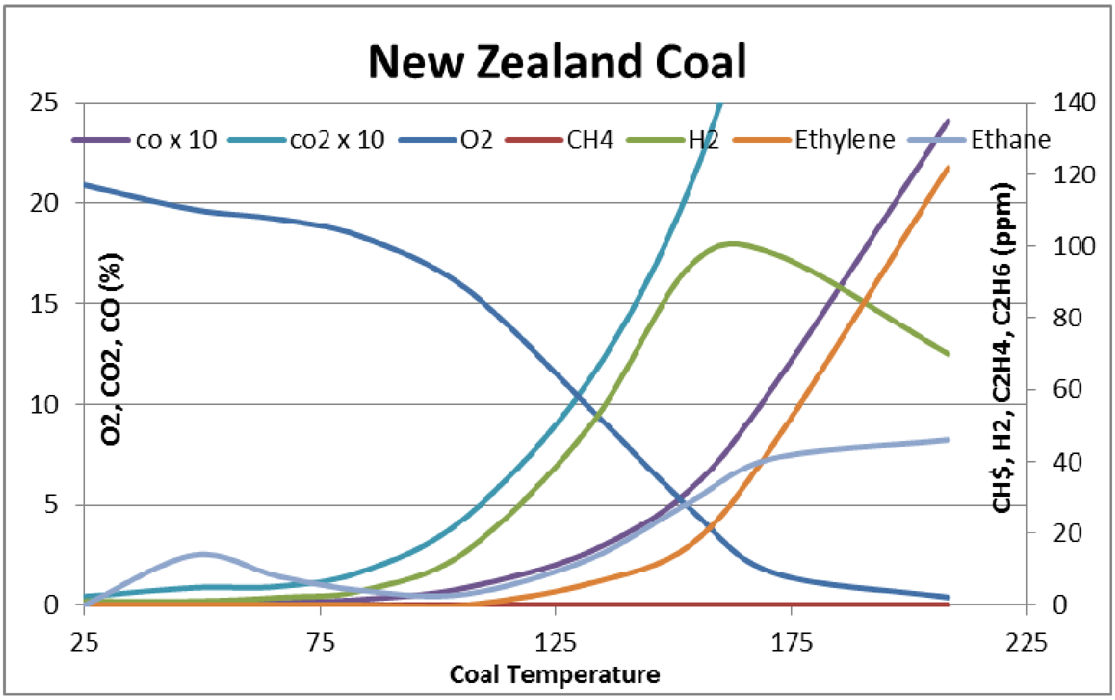
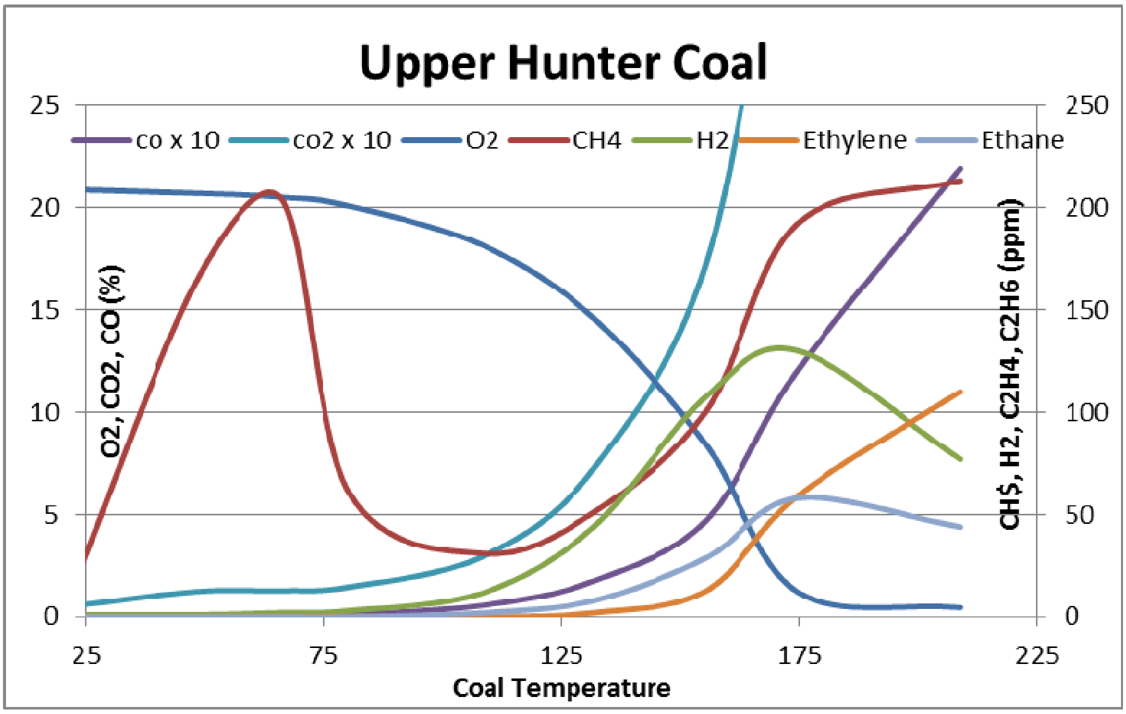


Figure 16: Fire ladder for Upper Hunter coal



The following figures 17, 18, 19 and 20, show a comparison for several different types of coal of the evolution of gases produced by heating coal. Absolute temperature values vary but behaviour is similar.

Figure 17: Gas evolution behaviour for various coals – CO

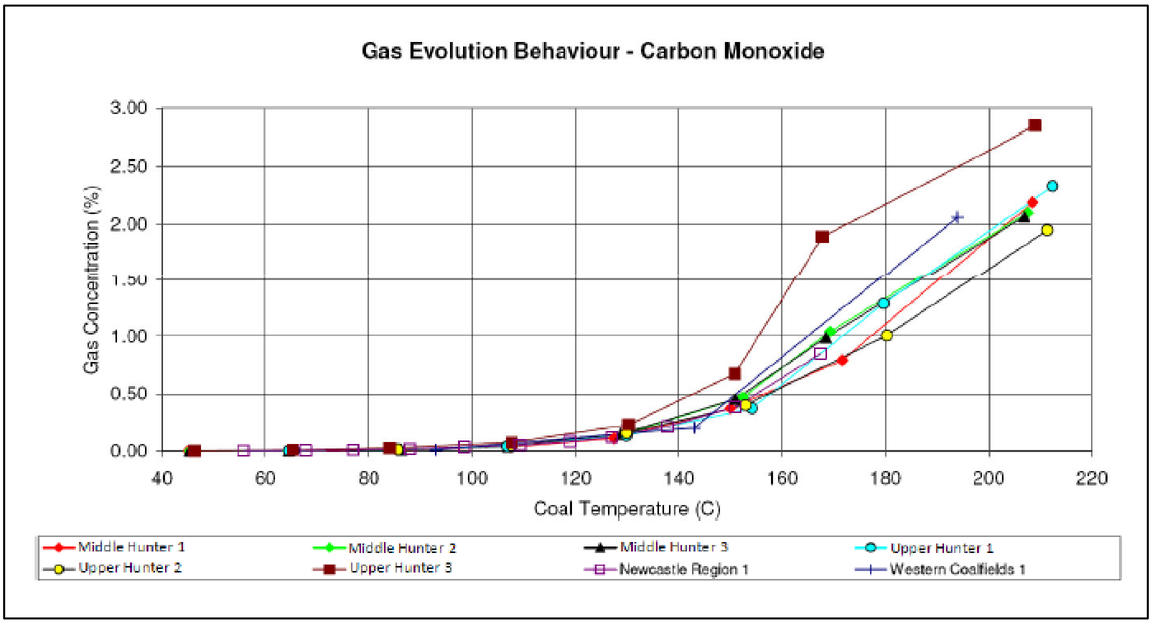


Figure 18: Gas evolution behaviour for various coals – CO2

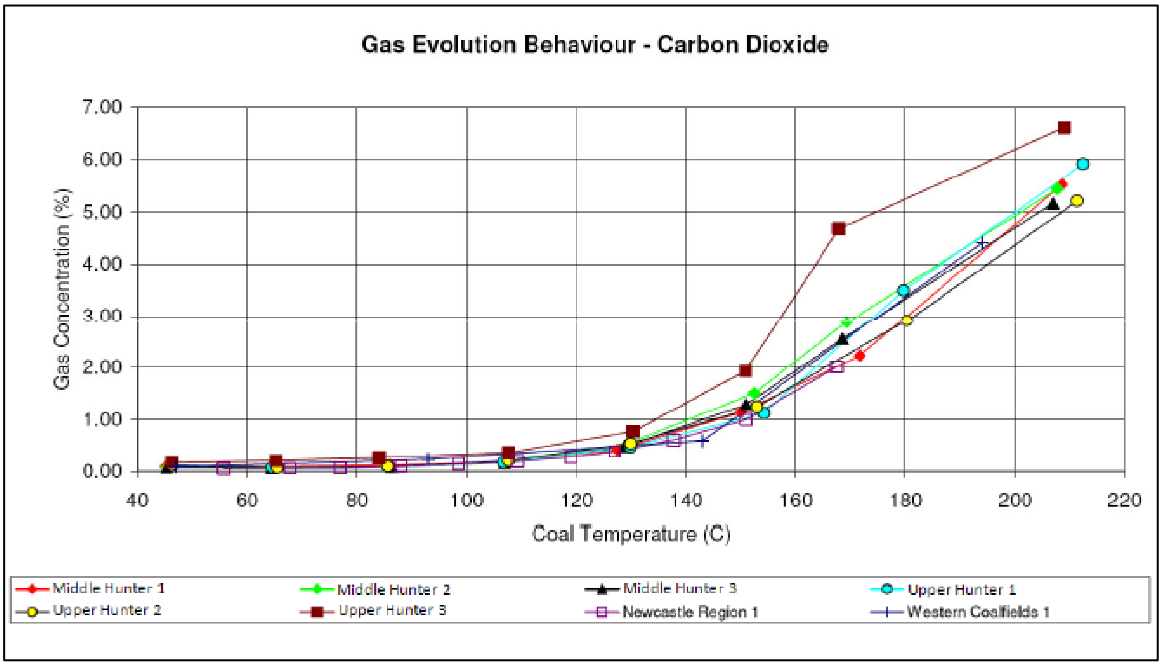


Figure 19: Gas evolution behaviour for various coals – H2

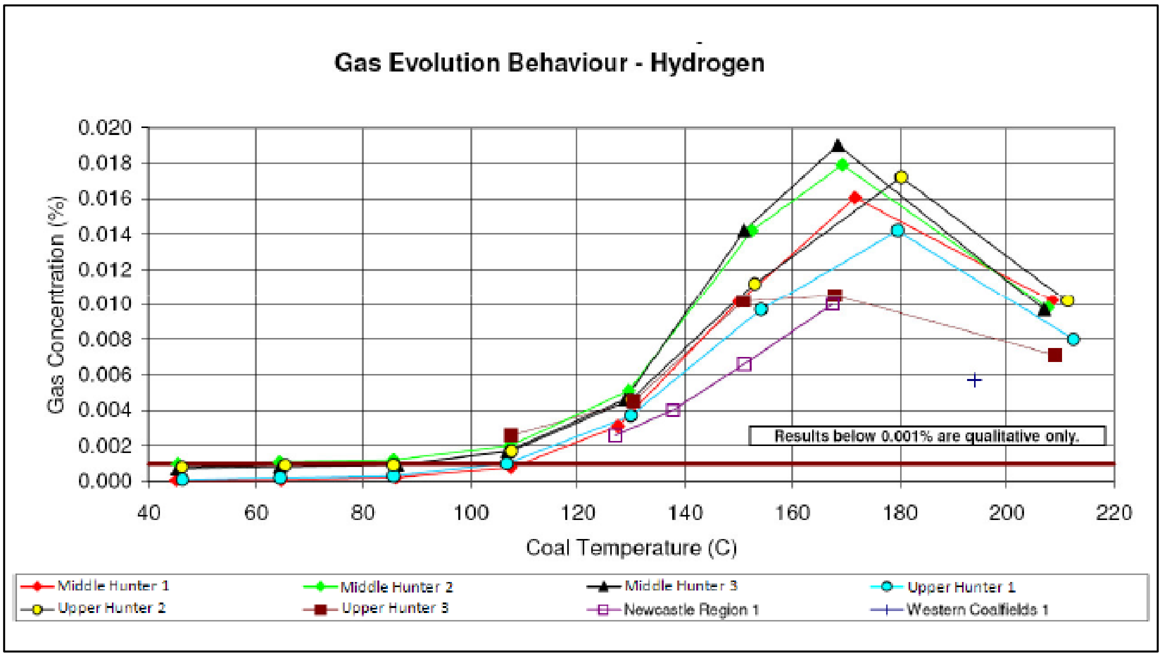
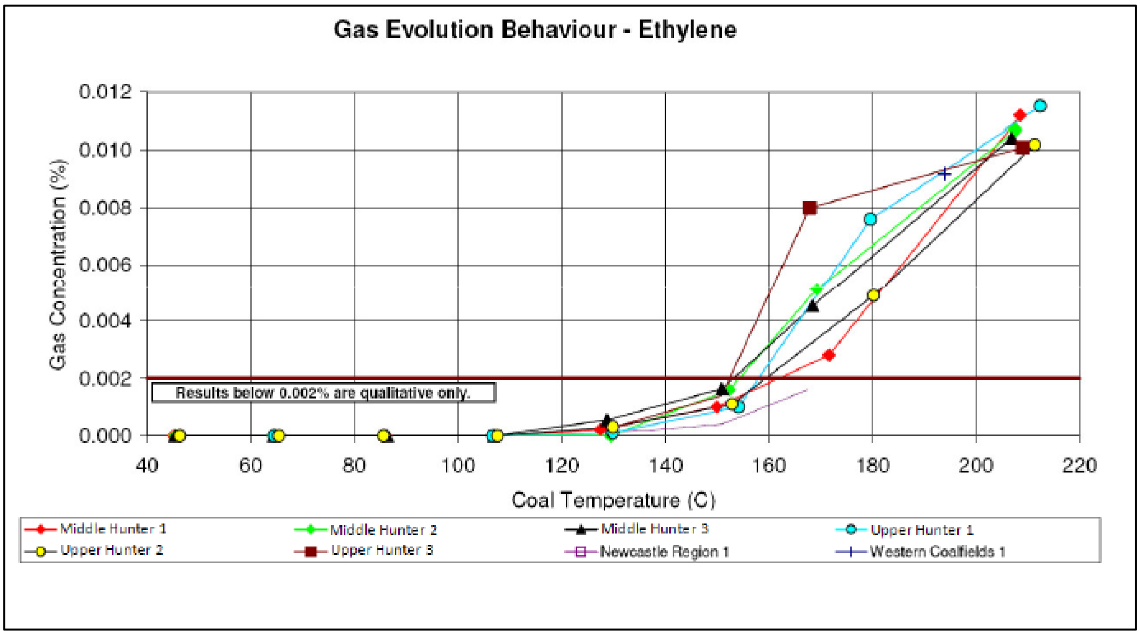


Figure 20: Gas evolution behaviour for various coals – C2H4



Indicators of spontaneous combustion risk should include both gas analysis based and other sensory or observation based indicators. Heatings can only be detected in the early stage if monitoring takes place in or near where heatings can occur.

6.3 METHODS OF DETECTION

Methods of detection and control of spontaneous combustion include:

- Physical inspection
- Monitoring of mine atmospheres in roadways
- Collection of atmospheric samples from goaves
- Thermography

Detection methods need to have these objectives:

- Determine when there are significant variations to the planned design of the ventilation system
- Ensure that containment of goaves is of a high standard
- Confirm the goaf areas are inert
- Confirm that stowage has been removed or otherwise contained
- Ensure standards for stoppings and seals and roadway surrounds are met
- Recognise early signs of spontaneous combustion activity

6.3.1 Physical Inspection

The observation of physical signs is an important method of detection. Physical signs are:

- Rise in temperature
- Sweating (approx. 100oC)
- Smell (approx. 110oC)
- Haze
- Smoke (approx. 300oC)

In some coal seams, very little CO is given off before the heating can be detected by smell or other physical signs. (Greta seam) In other coal seams, gaseous indicators may provide the best early warning. (Liddell seam)

Small quantities of CO may be missed (location and placement of sampling points etc.) or be considered erroneous because the readings were not repeated on a regular basis. Even a fleeting smell is distinctive to an experienced and trained person.

The key factor in the detection of spontaneous combustion is a change from normal conditions. Changes can be in airflow and direction, smell, temperature, gas levels etc. Any change should be fully investigated.

Inspection of stoppings and seals is important to ensure they are effectively containing the goaf. Matters to note when inspecting stoppings and seals include:

- Water trap if fitted is filled with water and not leaking

- Signs of water build up on the inbye side of the stopping or seal
- Sample pipe is in good order with valve turned off
- Presence of stowage in the roadway adjacent to the stopping
- Access to stoppings & seals are accessible and ability to inspect, repair and sample
- Abnormal changes in gas readings in the adjacent roadway when the barometer is falling, indicating stopping damage
- Condition of roof, ribs and floor
- Atmosphere adjacent to the seal on a falling barometer

Stoppings and seals that contain and inertise the goaf should be examined on a regular basis to confirm the integrity of the structures and roadway surrounds.

Roadways should be safely accessible to all stoppings and seals. If these sites become inaccessible due to falls in roadways or other reasons, there is a risk that damage to the structure or roadway surrounds will not be detected. If so the risk of undetected spontaneous combustion will increase. Access to seal sites is important to confirm integrity and allow samples to be taken from behind the seal as required.

A useful technique for examining stoppings or seals containing a goaf is to walk the roadway adjacent to the sealed area with a gas detector and take note of the change in gas levels when passing each cut-through. This should be done when the barometer is falling. Smoke tubes may be useful in detecting leakage.

A disproportional and significant change in gas levels indicates a badly leaking seal or stopping. Physical examination of the stopping or seal then can confirm if the problem is damage to the stopping or seal, roadway convergence, or both. Recording of results of changes in gas levels and the barometer is useful in determining trends in subsequent inspections.

In mine roadways, heatings in stowed or fallen coal and rib side pillars may be difficult to detect in the early stage. Reliance on monitoring changes in mine atmosphere should be not be relied upon as the principal method of detection. The observation of physical signs is important. Thermography may be useful.

6.3.2 Monitoring of Atmospheres in Roadways

The atmosphere in areas may be sampled by means of real time sensors, tube bundle lines and sampling bags. Surface analysers with tubes to various parts of the mine are most often used for sampling the mine atmosphere in ventilated roadways adjacent to goaves for signs of spontaneous combustion.

The time delay for gathering samples through the tubes is normally not critical for the detection of spontaneous combustion activity. The analysers are able to more accurately detect small percentages of gas than real time roadway sensors.

If power is lost in the mine, or access denied, the tube bundle system may continue to operate unless lines are damaged. The range of operation of the analyser may be an issue in an emergency situation where high concentrations of gas may be present.

No matter how extensive or sophisticated the monitoring system may be, unless it is sampling in the right areas it will not provide the necessary information.

6.3.3 Monitoring of Atmospheres in Goaves

Sample bags are often used to supplement the system as there is a limit to the number of tubes and sampling points per analyser.

If there is reliance only on the monitoring of the atmosphere in mine roadways adjacent to extracted areas, and not from within the extracted area, then the system of early detection will be deficient. Damaged stoppings may allow goaf air to exit to the adjacent roadway and this may, or may not be detected at the roadway sampling point.

The atmosphere in the goaf adjacent to adjoining ventilated roadways should be sampled on a regular basis commensurate with risk. Sampling interval may be extended when the atmosphere is demonstrated to be inert.

In the active longwall panel where there are usually large numbers of stoppings enclosing the goaf, samples should be taken along the perimeter of the goaf to confirm the goaf atmosphere is inert, and then resampled based on results and risk. Samples should be taken frequently until the atmosphere is determined to be inert.

Detection at intake seals is made more difficult by the inflow of air, which makes sampling by conventional means dependent on the state of the barometer. One useful technique to limit this effect is the “Buffer Zone” where a second stopping is established outside a seal (balance chamber). The distance between the seals is calculated from the volume of the goaf and the normal pattern of changes in barometric pressure, and is adequate to contain a substantial portion of the gas emitted on a falling barometer. An open pipe passes through the outer stopping.

It is important to understand the oxygen distribution in the goaf, the time required for coal to reach the cross over temperature at which oxidation accelerates (approx. 70^0C) and for action plans to be developed and implemented in the event of a face stoppage.

6.3.4 Goaf Sampling

Sample pipes placed in stoppings and seals should be made from materials that are not reactive. Copper is preferred. The action of acid mine water on galvanised iron can produce hydrogen.

The location of the sample pipe in a seal needs to be determined according to the purpose of the sampling and conditions in that part of the mine. Factors that influence the location and design of the seal sample point include:

- What is the purpose of the sample and what is to be sampled? If the purpose is to sample the atmosphere in the goaf, then the sample pipe should extend to that location.
- What are the gases likely to be in the roadway? If the gases are liable to layering, the sample pipe should be located at the required level.
- What is the dip of roadway? This may result in certain gases migrating
- Is there water behind seal? This may cause sample pipes located close to the floor to block up.

Sample pipes should be slightly inclined to eliminate water

For the purpose of the detection of spontaneous combustion, and given no other constraints, sample pipes are best located in the upper part of the roadway, with the inbye end adjacent to the goaf edge.

The Queensland Department of Employment, Economic Development and Innovation recently revised and reissued Recognised Standard 09 – The Monitoring of Sealed Areas. This document addresses in detail:

- The location of sampling points
- Parameters to be monitored
- Sampling frequency
- Maintenance of seals
- Analysis of information and response
- Storage of information
- Record keeping and reporting

The standard has been released to provide guidance on how to predict and adequately define the potential for an explosive atmosphere to occur within a sealed area, as well as monitoring to identify the potential for spontaneous combustion within the sealed area.

6.3.5 Monitoring of Stowage & Pillars

Thermography is an effective means for detecting rib side pillar heatings and stowage and leakage paths around seals.

Thermocouples can be installed into pillars where a high risk of heating has been identified. Gas sampling from boreholes in pillars is another option.

Physical inspection (using gas sampling and senses) remains one of the most effective means of detection.

6.4 GAS MONITORING SYSTEMS

A comprehensive gas monitoring system is an effective tool for the detection and monitoring of spontaneous combustion. Monitoring systems include:

- Tube bundle system with analysers located on the surface
- Real time (telemetric) system
- Routine inspections using portable devices
- Gas bag sampling for analysis at the mine or by a third party provider
- Gas chromatographic systems

Although a combination of all the above would offer the ideal gas monitoring system for a coal mine, not all mines utilise, or require all components for their particular site. Telemetry (fixed underground sensors) and tube bundle gas monitoring systems are most commonly utilised for monitoring underground atmospheres in Australian coal mines. While both types of systems provide an important and useful means for the routine monitoring of specific gases (i.e. methane, oxygen, carbon monoxide and carbon dioxide), they are generally not suitable for the accurate monitoring and trending of gases produced from an advanced oxidation or spontaneous combustion. In these instances, a gas chromatograph is the preferred analyser for effective decision making as it has the ability to not only accurately determine all the above gases, but also determine key indicators such as hydrogen and ethylene.

While telemetry and tube bundle systems are employed to essentially monitor the same gases, there are significant differences in the type of sensors that are used, the quality and also the range of detection for the individual gases.

6.4.1 Tube Bundle Gas Monitoring Systems

Tube bundle gas monitoring systems utilise sample points that are located for ongoing trending of the mine atmosphere and in areas that do not require immediate warning of contaminants.

The system was developed in Germany in the 1960s to detect and monitor the progression of oxidation and spontaneous combustion events. The fundamental components of the system include a series of plastic tubes extended from the surface to selected locations underground. The tubes are general high grade quality non leaching materials with a variable diameter from 6mm to 20mm (depending on the length) and lengths of up to several kilometres. Air sampling scavenging pumps located on the surface draw the gas from each tube simultaneously via drying, filtration and flame trap systems. Individual tubes are then sequentially diverted into a bank of analysers for analysis.

With minimal restrictions in terms of the intrinsic safety, certifications, approvals, etc. for the analysers used for tube bundle systems (as they are generally located on the mine surface), a broader selection and higher quality of analysers can be used. Non dispersive infra-red (NDIR) analysers are generally used for the monitoring of gases such as methane, carbon monoxide and carbon dioxide, while paramagnetic (or zirconia type sensor) analysers are commonly used for oxygen monitoring. In addition, the broader detection ranges available for these types of analysers facilitate the measurement of the higher gas concentrations found in underground sealed areas.

Although the analysers used for tube bundle systems are generally accepted as being of superior quality to the sensors used for telemetry gas monitoring systems, and do not have some of the cross sensitivity issues associated with some fixed type sensors, moisture in sample tubes can cause significant problems. Most tube bundle systems will have moisture removal devices that remove moisture from the gas sample streams. However, if these devices are not maintained and functioning efficiently, then the result is that any moisture entering the NDIR analysers will affect the accuracy of the readings. Mine sites with tube bundle systems will commonly have switching mechanisms that divert a dry calibration gas into the analysers to confirm the accuracy of the system. A mistaken assumption is often made that the moisture removal devices are working efficiently and, apart from checking individual water traps, no checks are made as to the effectiveness of the main water removal device/s. The result is that when the sample tubes are put back on line, comparative analysis with a gas chromatograph may show a discrepancy between the analysers.

Advantages:

- No explosion proof instruments required when flame traps are incorporated
- Easier maintenance as major components are located on the surface
- No underground power requirements
- A wide range of gases can be analysed
- Analysers can be calibrated on the surface

Disadvantages:

- Results are not in real time
- Leaks in tubes may not be immediately apparent.
- Condensation in tubes can results in blockages and erroneous readings on some types of analysers if moisture removal systems are not adequate
- Faults in tube system may not be immediately apparent.
- Tubes may be damaged by fires/ explosions

6.4.2 Telemetry Gas Monitoring Systems

Fixed sensors are generally located where real time data is required and can therefore provide early warning for the onset of an oxidation.

In relation to the types of sensors used for fixed sensor gas monitoring systems, the selection range is restricted to those certified by the relevant regulatory body. A combination of catalytic combustion (for methane detection), electrochemical (for carbon monoxide and oxygen detection) and simple infra-red detectors (carbon dioxide and methane detection) are typically used for these gas monitoring systems.

However, the sensors used for telemetry systems are often lacking in their range of detection, are generally less stable, can be cross-sensitive with other gases and have a much lower operational life expectancy than the types of analysers that are used for tube bundle systems. Despite these limitations, the real time monitoring capability of this type of system is very important in terms of providing rapid early warning for a mine site.

Advantages:

- Results in real time (rapid indication of potential problems)
- Relatively long distances from surface to sensors are possible
- Sensor failure is generally immediately recognised

Disadvantages:

- Relatively high maintenance
- Limited range sensors for ongoing monitoring of a spontaneous combustion
- Poisoning of methane sensor may occur
- Cross sensitivity for some sensors
- Loss of power when methane limits exceeded
- Limited sensor life

- Unsuitable in oxygen deficient atmospheres (i.e. behind seals).

6.4.3 Gas Chromatograph

Gas chromatographs have been used as an analytical tool for the analysis of underground coal mine atmospheres for decades. They have been useful in providing accurate analysis of components that are not routinely monitored by telemetry or tube bundle gas monitoring systems. These components include hydrogen and hydrocarbons such as ethylene and propylene.

While conventional gas chromatographic systems were initially utilised at some coal mines and other agencies in Australia, their analysis times were too slow for the high volume sampling rates required during mine emergencies. In addition, they required frequent maintenance and a relatively high level of operator expertise. They also had difficulties in analysing parts per million (ppm) levels of carbon monoxide in a balance of high methane and similarly ppm levels of ethylene in a balance of high carbon dioxide. These systems are no longer considered to be an appropriate analytical tool for the monitoring of a spontaneous combustion incident.

The introduction of ultra-fast micro gas chromatographs into the market in the 1980s resulted in a wider acceptance and use of gas chromatographic systems at coal mine sites. They provide analytical run times of between 1-3 minutes for the analysis of key spontaneous combustion gas components. They generally utilise a single detector type (Thermal Conductivity Detector, TCD), require less maintenance than conventional gas chromatographs and are relatively simple to operate.

It may be argued that due to restrictions in their operating systems, some models of ultra-fast micro gas chromatographs have similar limitations to conventional gas chromatographs when analysing low ppm levels of carbon monoxide in a balance of high methane. However, this is not the case for all micro gas chromatographs. There are ultra-fast micro gas chromatographic systems that are able to reliably and accurately determine low ppm levels of carbon monoxide in a balance of high methane. Hence the selection of the correct type of system is very important.

The main advantages in using this type of gas chromatograph for the analysis of coal mine atmospheres include:

- Ability to separate and analyse key spontaneous combustion components, including hydrogen, carbon monoxide, ethylene, ethane and propylene at ppm to percentage levels
- Ability to analyse other general gases found in coal mines including oxygen, nitrogen, methane and carbon dioxide
- Rapid analysis of the above components in typically 1-3 minutes
- Only one type of detector is required for analysis of mine atmospheres
- Relatively simple to operate

6.4.4 Sample Turn-around Time

The “turn around” time required to take the sample and produce a result should be considered. A gas chromatograph located on the mine site can produce results much more quickly than transporting off site to a laboratory that may not be open for business 24 hours per day.

Infra-red analysers located on the mine site will produce results quickly but not provide information on hydrogen and ethylene etc.

Obviously, if a heating is detected in the early stage, time is not critical. As the event develops, time does become critical.

Information on atmospheric conditions is critical to decision making in spontaneous combustion events.

6.4.5 Location of Monitoring Points

To be effective, the monitoring points need to be situated in locations where products of oxidation can be detected. In addition to monitoring the atmosphere in roadways adjacent to the goaf, the atmosphere within the goaf should be determined. (Refer to 6.3)

Alarm levels for monitoring points should be determined and integrated into TARPs.

6.4.6 Gases Sampled

Tube bundle systems should, at the very least, monitor CO, CO2, O2 and CH4. In addition to recording trends of gases produced by heatings, collecting information on all these gases will allow calculation of ratios that are useful for monitoring the development and progress of a heating.

The integrity of the monitoring system should be regularly confirmed.

6.4.7 Production of Gases not Related to Heatings

False alarms may be generated where the above mentioned gases are produced by means other than spontaneous combustion activity. Examples are:

- Use of galvanised iron as sample pipes
- Acid mine water on galvanised iron producing hydrogen and carbonates producing CO_2.
- CO and CO_2 from vehicle emissions
- Unplanned stowage of chemicals and oils
- Oil shales (volatile emissions)

6.5 MONITORING LOCATIONS

The location of monitoring points at strategic locations is of major importance. A single sampling point some distance from the source provides an indication only and can often lead to either an over estimate or under estimate of the seriousness of the hazard.

Points must be sited where heatings are likely to develop. There needs to be little dilution of flows between the heating and the detectors. Consideration must be given to layering of methane and warm combustion gases which may rise up dip in a sealed area.

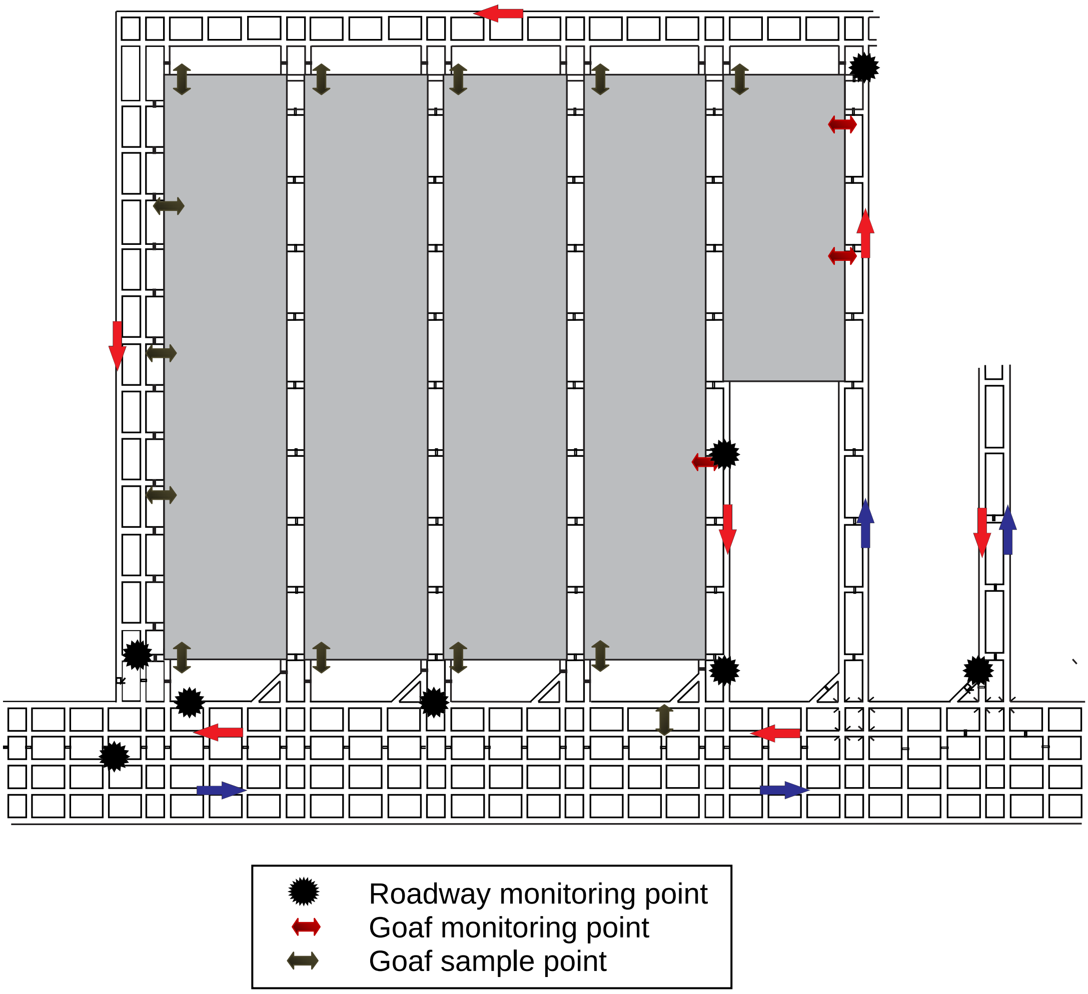
Ideally, sampling points should be in panel returns, behind stoppings and seals and in the main body of the ventilation circuit. All sampling points should be clearly located on the mine ventilation plan.

In identifying the location and number of monitoring points, the ability to determine where the contaminant is originating is important. If the ventilation from a number of possible heating sources passes over a single sampling point, the origin cannot be determined, nor the sample result confirmed

Figure 21 shows the suggested locations for the recommended minimum number of environmental monitoring points for a longwall panel. It includes surrounding roadway and goaf monitoring points for the current and adjacent longwall blocks.

Recommended sites for goaf sampling are shown. Location and frequency of sampling should be based upon results of atmospheric analysis, stability and an assessment of the hazard.

Figure 21 – Location of Monitoring Points



6.6 INTERPRETATION OF RESULTS

The trend in gas levels is important. Are the levels rising, steady or falling? Using gas values alone is problematic. Ratios and CO make are much better indicators.

There are a number of ways certain gases and their presence can be interpreted to determine the presence of a heating and its stage of development.

The ingress of oxygen into a contained goaf provides conditions for the development of a heating. It occurs well before the liberation of products of combustion and is a valuable early indicator for the development of triggers for the mine TARPS.

While there are well documented ratios and indices that are used for monitoring the progression of a heating, the following have shown to be of value:

- Graham's ratio
- CO/CO_2 ratio
- CO make,
- Trickett's ratio,
- Young's ratio,
- H_2/CO ratio
- air free analysis

Three of the most useful indicators for spontaneous combustion management plan TARPS are:

- Grahams ratio (GR) because values steadily increases as the heating progresses and it indicates the intensity or temperature of a heating. (but not the size)
- Similarly CO/CO_2 ratio because it also steadily increases as the heating progresses (not appropriate for mines with a high CO_2 seam gas composition)
- CO make because it compensates for varying air quantity

When setting trigger levels in the spontaneous combustion plan TARPS it is better to use a few important indicators so that people in the mine can be better trained for an effective response.

TARPS should be reviewed based on mine site experience and adjusted accordingly as part of a risk assessment review of the SCMP.

Most ratios are measures of the conversion efficiency of oxygen to products of oxidation and are therefore essentially equivalent. Oxygen consumed can be measured through oxygen deficiency compared with fresh air e.g. Graham's ratio, Young's ratio, etc. Or Excess Nitrogen compared with fresh air – Willett's ratio, Partington's ratio. Therefore there is no need to use a multitude of deficiency ratios as they should all tell the same story. Other ratios can be used to assist investigation but need not be part of TARPS.

Caution should be exercised in setting gas and ratio values based on gas evolution charts. Sampling limitations and goaf conditions dictate that more conservative values be adopted for TARPS. Factors impacting on atmospheric analysis when sampling from behind seals include:

- Where there is no airflow, there is no purging of the sample stream.
- When monitoring over significant time periods, secondary reactions and loss mechanisms may apply.
- There may be a mixture of gas sources.

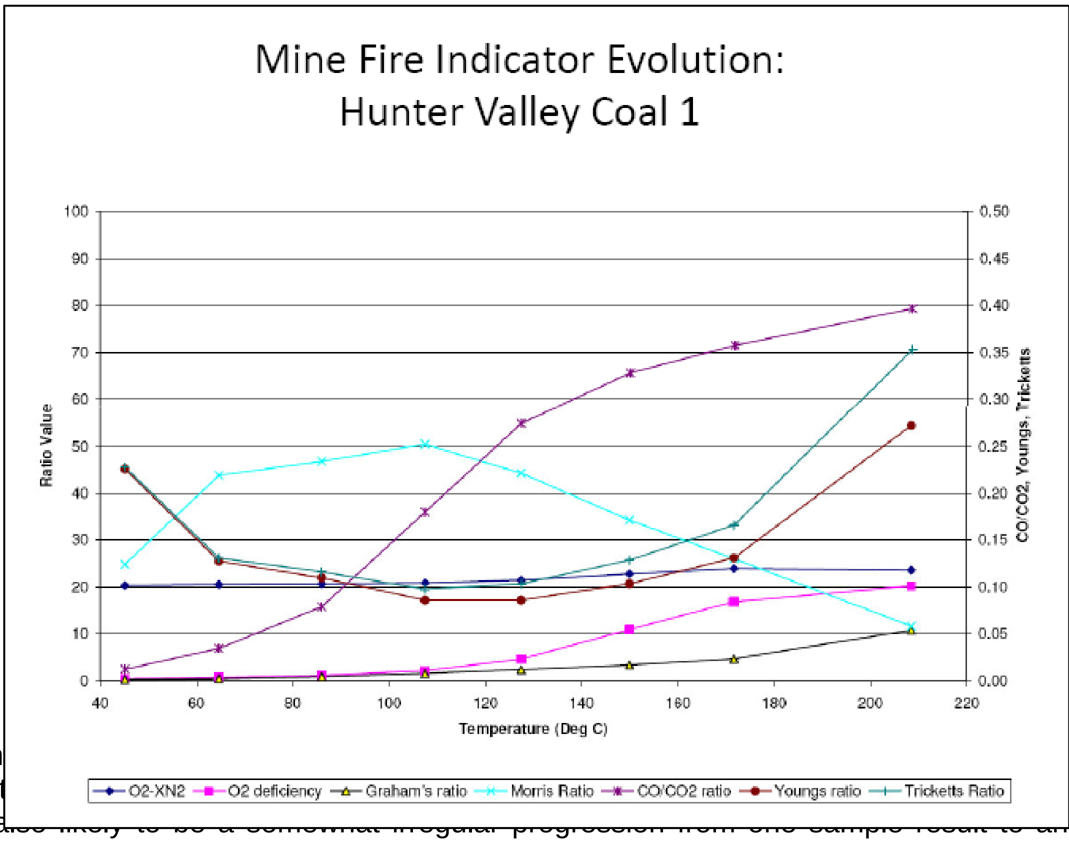
- Lack of oxygen can cause some sensors to read incorrect gas concentrations.
- The atmosphere sampled may be an average over time rather than current conditions

The discrepancy between the ratios and other indicators of spontaneous combustion activity is probably due to the presence of large amounts of broken coal in the goaf between the site of the heating and the monitoring location. This broken coal would be able to absorb oxygen and thus enhance the oxygen deficiency and act as a catalyst to destroy carbon monoxide or convert it to carbon dioxide. Thus the ratios would underestimate the severity of the situation. In general the effects of sealing on ratios underestimate the severity of a situation.

Determinations are only as good as the accuracy of the measurements taken and calibration of monitoring equipment. Concentrations should be adjusted for any background concentrations such as CO_2 in air.

Figure 22 Mine Fire Indicator ratios show a comparison of the behaviour of a heating shown by calculation of various ratios. The results are based upon laboratory tests of coal properties. The chart is useful in showing the progressive movements in the various ratio values. Caution should be used in considering absolute values for adoption in TARPS. Results may vary considerably for other coals.

Figure 22: Mine fire indicator ratios



Wh … s are plot … There is a … other. This occurs because:

- Sampling may not have been taken correctly or been diluted or contaminated
- Barometric changes

- The variable path of products of combustion from the heating site to the sample site and changes in airflow and direction
- Dilution and absorption from areas other than the heating site

The general trend should still be able to be determined. Making decisions based upon individual absolute values is problematic.

6.6.1 Grahams Ratio

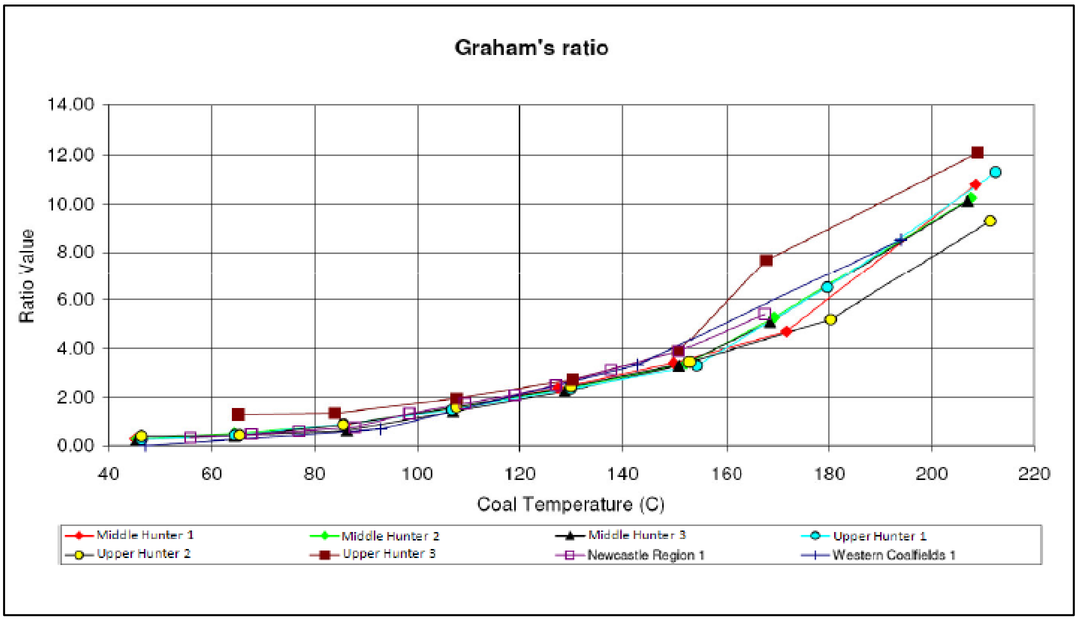
Graham's ratio (GR) is useful in low oxygen environments such as goaves and is also applicable in ventilated roadways.

In ventilated roadways there needs to be a perceptible oxygen deficiency. (The Qld regulations require graham's ratio to be monitored in all panel returns even though with the high flows in most longwall panels there is no perceptible oxygen deficiency)

GR is an indication of heating intensity and temperature and can discriminate between a large mass of coal oxidising and a small intense heating.

Figure 23 shows the laboratory test Grahams Ratio values for a number of coals. It is useful in that it shows the progressive rise in value commensurate with the temperature increase and a similar relationship for the coals tested. Absolute values should not be used for TARPS.

Figure 23: Grahams Ratio values for various coals



Values for Grahams Ratio quoted in a number of technical references are:

< 0.4	Normal
0.4 - 1.0	Investigate
1.0 - 2.0	Heating
> 2.0	Serious Heating or Fire

Operators should determine trigger points for TARPS based upon experience in the seam mined and the determination of risk. This may require lower GR values.

6.6.2 CO/CO_2 Ratio

The CO/CO2 ratio is suitable for both sealed & fresh air heatings. This ratio is independent of oxygen deficiency and so overcomes a lot of the problems associated with other ratios that are dependent on that deficiency. It defines typical coal temperature values. This index can be used only where no carbon dioxide occurs naturally in the strata.

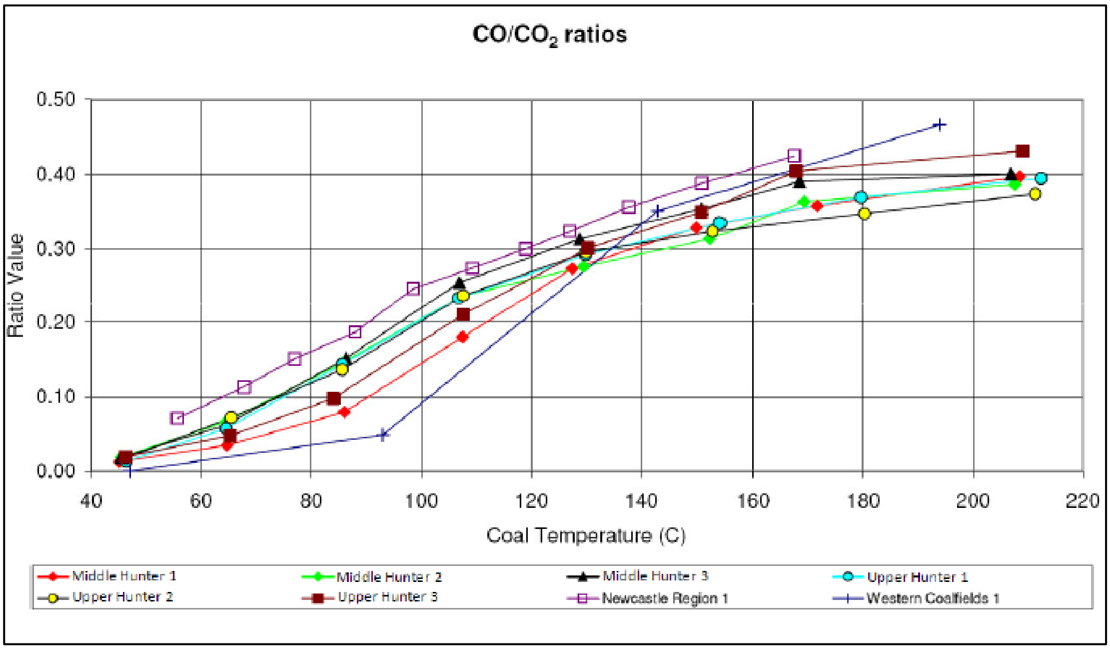
The index increases rapidly during the early stages of a heating, but the rate of increase slows at high temperatures as shown in Figure 24. However, the rate of change at higher temperatures is sufficient to provide a very useful indicator of the progress of a well-established fire. The ratio is independent of dilution with fresh air or seam gas except of course when the seam gas is carbon dioxide.

Typical values of this ratio for Australian coals are:

<0.02	Normal
<0.05	Coal Temperature <60^0C
<0.10	Coal Temperature <80^0C
<0.15	Coal Temperature <100^0C
<0.35	Coal Temperature <150^0C

This ratio is only intended as an early warning for heatings. If an active fire exists the ratio can actually decrease. Further this ratio is invalid if the nitrogen or oxygen deficient atmosphere is passed over any heating or if the oxygen concentration exceeds 20%.

Figure 24: CO/CO2 ratio values for various coals



6.6.3 CO Make

CO make is most useful in panel returns and back bleeder roads. It is the volume of Carbon Monoxide flowing past a fixed point per unit time. This indicator removes the effect of dilution by general body air.

The CO make is dependent upon the amount of coal reacting with air so that if conditions change and a larger goaf is ventilated then the CO make will increase without any actual increase in the intensity of the oxidation (larger volumes of goaf exposed to air are a concern in their own right.

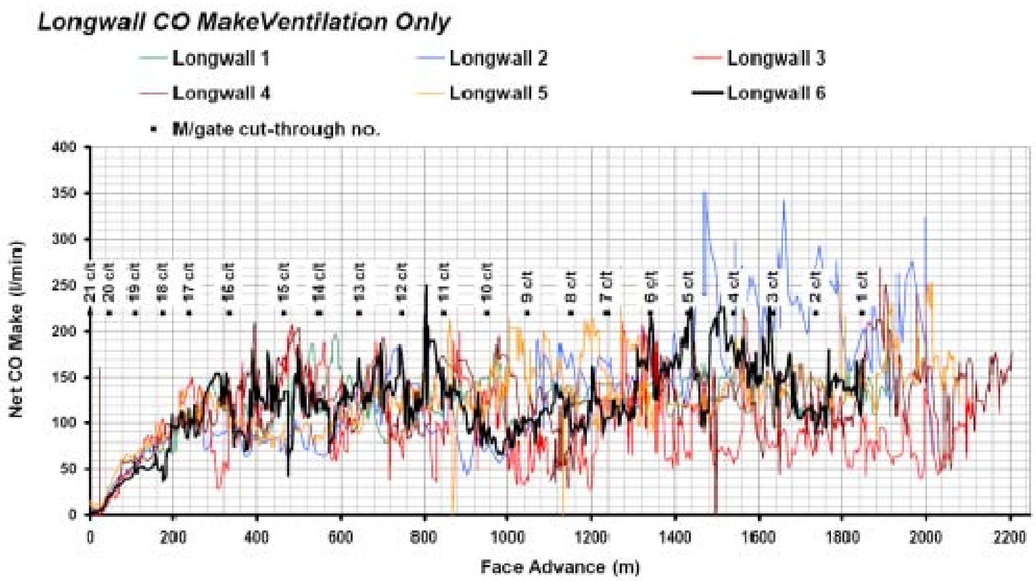
"Textbook" levels quoted are:

- Levels of production > 10 litres per minute require investigation.
- Levels of production > 20 litres per minute indicate that considerable danger exists.
- Levels of production > 30 litres per minute indicate that extreme danger exists.

Values should be set based upon conditions in the mine and these may be higher or lower than the above values.

Figure 24 shows the CO make in the longwall return in the Upper Wynne seam, Dartbrook Colliery NSW. The seam was thick and a high ventilation quantity was required to dilute the CO_2. One maingate seal left open behind the retreating face. Readings are high for most mines but the graph does show the "sawtooth" effect and trends. Several readings triggered the spontaneous combustion TARPS for the mine.

Figure 25: CO make in a longwall return



6.6.4 Air Free Analysis

The following information, in relation to "air", assumes that air contains 20.93% oxygen and 79.07% nitrogen (including inerts).

In many situations the atmosphere of interest is diluted by being mixed with other atmospheres, particularly fresh air. In some cases it is possible to remove the impact of fresh air through adjusting the gas concentrations. This can be achieved in a number of ways.

1. Assuming all the oxygen present is due to the diluent gas

In this case the gas concentrations are adjusted by removing the oxygen concentration and the associated nitrogen concentration (based upon 3.778 times the oxygen concentration. The residual gases are then normalised to 100 %. This is also used to estimate what the ultimate concentrations of gases would be if all the oxygen present was converted using the conversion efficiency of the sample.

For example: Assume the fringe of a sealed area contains only the seam gases methane and carbon dioxide and no air contamination (i.e. 80% CH_4 and 20% CO_2). Assume then that the seal breathes in and that the atmosphere at the fringe behind the seal now contains 50% seam gas (40% CH_4 and 10% CO_2) and 50% air contamination (10.465% O_2 and 39.535% N_2 + inerts). To determine what the seam gas component concentrations are without the air contamination using this method is commonly termed an “air free” calculation. The air free calculation calculates the air contamination based on the normal air/ oxygen ratio of 4.778 and applying the resulting factor to the remaining components. The result would then be an air free concentration of 80% CH_4 and 20% CO_2 (i.e. the original concentrations).

However, if an oxidation/heating occurs behind the seal then the reaction process will consume part of the O_2 in the air concentration. The consumption of the O_2 would produce CO and CO_2 and also result in an excess nitrogen concentration (relative to the original normal N_2/O_2 air ratio). The air free calculation using this method can sometimes be useful in monitoring the level of oxidation products to assist in determining if the process is continuing, or if increased/ decreased levels are purely the result of air moving between the sealed areas as a result of diurnal atmospheric pressure influences.

2. Identifying the degree of dilution through trending or comparison with other gas samples.

For example: it may be possible to use gases such as methane to indicate the degree of dilution. Methane may be expected as a goaf/seam gas to be in a range of values and yet is much lower due to dilution. This dilution is simply worked out based upon the ratio of the expected value to the actual value. The dilution factor is then applied to all gases other than oxygen and nitrogen. The oxygen and nitrogen concentrations are then calculated by difference, scaling them as per the original diluted gas mixture.

This technique is most reliable when the degree of dilution can be confirmed by other gases.

This technique can be used for adjusting the effect of barometric pressure as well. Where concentrations of gases are affected by barometric pressure the variation can be analysed to identify what the diluent gas mixture is. This diluent can then be removed and the residual gases scaled appropriately.

Automatically carrying out air free analysis is not recommended. It is best carried out only when the nature of the diluent atmosphere is confirmed, and by experienced personnel. For example: any attempt to air free an atmosphere that is close to fresh air will lead to wildly inaccurate estimates of the residual gases, due in part to the limitations on accuracy/reproducibility of the gas analysis.

6.6.5 Ethylene (C_2H_4)

Ethylene (C2H4) is a useful signature gas that results from spontaneous combustion and no other known cause. It does not appear until the temperature reaches approx. 1500C. It is not an early indicator but a very useful indicator as the heating develops.

6.6.6 Hydrogen

The presence of hydrogen in abnormal quantities is another indicator. Unlike ethylene, hydrogen has been discovered in circumstances at some mines where there was clearly no incidence of spontaneous combustion.

Hydrogen has been commonly determined at low ppm levels in longwall goaves and borehole drilling at regular intervals.

Hydrogen has also been identified during sampling as a product from acid water reaction with galvanised steel, and use of non-reactive sampling tubes is required to avoid this problem.

Care in the analysis needs to be taken to avoid mistaking helium for hydrogen as they have similar retention times in gas chromatography. Helium is commonly found as a seam gas and in goafs.

7 RESPONSE

7.1 TRIGGER ACTION RESPONSE PLANS (TARPS)

TARPS are a means of providing clear and concise triggers for mine personnel to react to abnormal conditions that may cause risk to property or persons. They provide a graduated response with each stage, if changing conditions are not corrected, becoming more serious. The lowest level response is intended to recognise change and provide time for corrective action before people are placed at risk.

TARPS will have graduated levels of response dependent upon the severity of the situation and risk. A spontaneous combustion management plan TARP system should contain at least 3 levels. Additional levels may be advisable after consideration of the risk and circumstances at the mine.

The three basic graduated levels of response are:

- A change from the normal conditions requiring investigation
- Evidence of a loss of control requiring action to correct
- A risk of harm to people requiring withdrawal of persons from the area

7.1.1 TARP Triggers

Trigger points for response should be clearly identifiable values or observations of change from normal conditions that are ideally not dependent upon a particular persons experience or judgement and not subject to misinterpretation.

The requirement for mine personnel to respond to spontaneous combustion TARPS will be irregular and infrequent. TARPS should be summarised simply in one or two pages so that persons required to action deviations or abnormalities can reference the information they need quickly and without misinterpretation.

TARP triggers will vary for different parts of the mine because of the atmospheric conditions in the monitoring point.

- Fully sealed goaf
- Active goaf
- Bleeder or perimeter roadway
- Extraction panel return
- Intake to return pillars

Circumstances of a heating in a goaf where there is no positive airflow and low oxygen will obviously differ from that of a heating in stowage in a normally ventilated roadway.

Triggers useful for the development of TARPS include:

- Loss of access to seal sites
- Damage to seals
- Unplanned significant increase or decrease in ventilating pressure

- Increase in gas levels in the roadway adjacent to seals indicating abnormal leakage
- Abnormal levels of CO
- A rise in the value of oxygen in an inert goaf
- A progressive increase in CO make in a longwall or continuous miner extraction panel return
- A progressive increase in CO make in a bleeder return
- A progressive increase in the grahams ratio, or other indicator ratios adopted for the mine.
- A change in smell or other physical conditions.

7.1.2 Early Stage Responses

There may be one or a number of stages in a mine's TARP system that can be considered an early stage response.

The first step in an early stage response is to confirm the condition. The number of erroneous readings from environmental monitoring systems may be significant and require readings to be verified. The reason for spurious readings can include:

- Ventilation monitoring system fault
- Environmental monitoring (EM) tubes being damaged with normal mine air entering a tube connected to the goaf.
- EM Filters and water traps not being serviced correctly
- EM analysers not cycling correctly so that residual air from the previous sample contaminates the current sample
- Incorrect location of tube sampling points in the roadway
- Analysers not calibrated correctly
- Analyser calibration drifting
- Flow failure

Some (spurious) readings may be "one off". A repeat sample may be normal. Readings may be confirmed by:

- Waiting for a second reading
- Sending a mine official to the site to inspect the area and confirm the condition by inspection

If the system of environmental monitoring is a tube bundle system with analysers located on the surface, the cycling of the sampling and analysis through multiple points can be altered to sample through one or two points and produce more rapid results from the problem area.

The response depends on the perceived severity of the alarm. If it is has the potential to cause harm to people then action should be taken to withdraw people from the area before confirming monitoring results. If it does not appear to constitute an immediate risk to people in the mine, it may be best to confirm the result before action is taken, assuming that this can be done quickly.

An example of a useful control for long wall panels with back bleeder roadways is to stopping off the back bleeder roadway which effectively reduces the ventilation pressure across the goaf and removes the necessity for a large number of goaf edge stoppings to contain the goaf. It is a short term solution that may cause a problem with holing out the adjacent longwall face and making use of the roadway for the next longwall tail gate.

Once abnormality has been confirmed control actions should be initiated and preparations made for more severe action whilst there is still time. Actions that should be considered are:

- Preparation for inertisation or sealing of areas.
- Preparation for withdrawal of key equipment should evacuation be considered.
- Dewatering arrangements

Time is of the essence so waiting some time to confirm a condition is not sensible.

7.1.3 Withdrawal of Personnel

When a spontaneous combustion heating develops to a stage where there is risk of fire or explosion, TARPS will require people to be withdrawn from the mine, a late stage response. The limitations on discovering a heating site and accurately determining the stage of the heating require a conservative approach, so that people are withdrawn well before exposure to the risk.

Use of inertisation equipment could allow people to remain in the mine to treat or isolate the heating site.

Re-entry arrangements should be considered via a risk assessment process.

7.1.4 Re-entry Provisions

Consideration should be given to the means of re-entry after having withdrawn personnel from the mine. Re-entry will normally be within the scope of the mine safety management plan and not be conducted under the provisions of the mine emergency system. TARPS may be determined for re-entry.

Issues are:

- Is the spontaneous event under control and safe for re-entry
- What are the atmospheric conditions in various parts of the mine and is it safe for re-entry
- Options for re-ventilation and method

Re-entry may require a mine rescue team to explore parts of the mine by entry through an air lock before the mine or part of the mine is re-ventilated.

Even after waiting several months before re-opening, it is wise to plan to make provision for rapid sealing of the part of the mine affected by the heating after re-entry. There are several cases of heatings re-activating within a few days after re-entry.

7.2 MANAGEMENT OF AN INCIDENT

7.2.1 Location of Surface Activities

The surface environmental monitoring analysers, the surface control room, muster room and the main fan controls should be located away from the mine entries where they are not at risk from an underground explosion or products of combustion. A 60 degree angle on both sides of the direct line of the seam entry is generally considered to be the area at risk of effect from an underground explosion. Noxious and explosive gases may accumulate in or near facilities located alongside mine entries.

7.2.2 Incident Management Team

An incident management team is the current convention for managing serious events. There should be provision to bring people in from other mining operations to participate and assist in the incident management team.

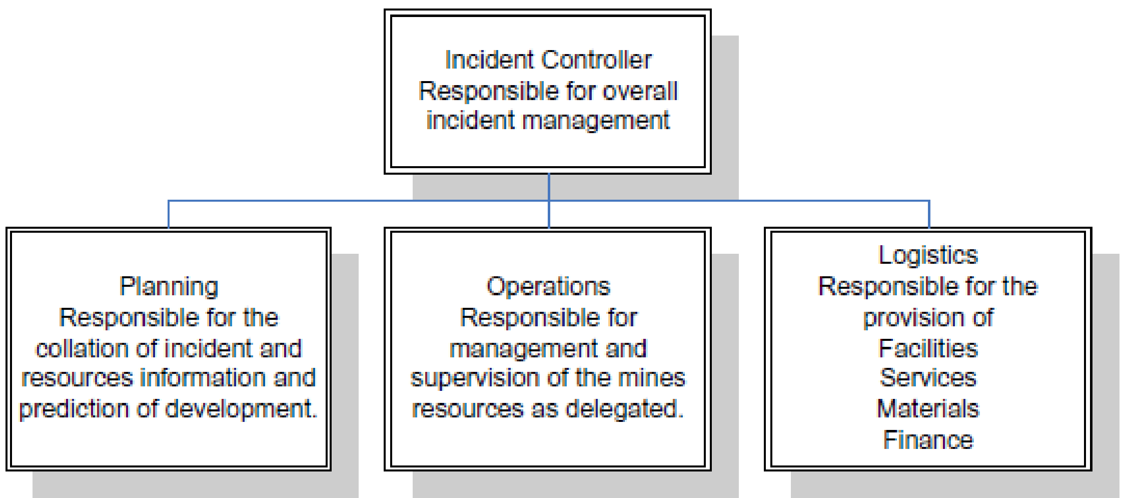
People working at the mining operation may be directly affected by the emergency, physically or emotionally. Knowledge and experience in the relevant disciplines should be considered in the selection of people for the team.

Those that may be expected to participate in such a team should be advised beforehand and provided with details of the mining and procedures for dealing with mine emergencies.

There is a tendency for incident management teams to become too large by including those that represent all interest groups in isolation rather than also considering the expertise and contribution individuals may make to solving very difficult problems.

A system adopted in most mines in Queensland, Figure 25, makes used of a management team approach that allows simultaneous tasking and improves the effective performance of the incident management group.

Figure 26: Incident Management Team



7.2.3 Monitoring under Emergency Conditions

The need to monitor under emergency conditions should be considered. Analysers that are suitable for normal operating mine environments may not be suitable for emergency conditions where gas levels exceed the range of the analysers.

The ability to monitor from the locations required under conditions that may negate access to underground workings should also be considered.

Surface access may be required to sample the atmosphere in the area of the heating by means of surface to seam boreholes. In a major event it is possible tube bundle lines could be damaged.

7.3 INTERACTION WITH OUTSIDE AGENCIES

In an emergency, assistance may be required from outside agencies such as Mines Rescue, Ambulance, Fire Brigade, mobile gas laboratory, etc.

Provision should be made for the placement of equipment provided by these agencies in a safe and secure area, clear of mine operational areas and mine entries. Such agencies will require communication, power etc.

Mobile gas laboratories and personnel to operate and interpret atmospheric analysis results are provided by two organisations in NSW:

- Coal Mine Technical Services (CMTS) in Wollongong – a division of Coal Services
- Department of Industry & Investment facility at Thornton

Arrangements for organising a gas laboratory on site can be made by contacting staff at the nearest NSW Mines Rescue station.

In Queensland, SIMTARS maintain a mobile gas laboratory facility.

7.4 INERTISATION

Inertisation of a goaf area can be an effective immediate control if provision has been made for it to done quickly. This allows time for investigation and implementation of a long term control.

If surface access is not available for inertisation and an underground supply system has not been installed, sealing the whole of the mine may be the only option. Inertisation of the whole of the mine will then extinguish the heating.

The following description of inertisation equipment is based on available information. Improvements in capacity, pressure, operation and monitoring are being developed for several items and should be researched by those seeking such equipment.

7.4.1 Flooding

Inertisation by gases is effective in controlling and extinguishing a heating but not in cooling the area. Flooding is most effective for this purpose. Conditions in the mine are conducive to the retention of heat and it requires several months for a significant reduction in temperature.

7.4.2 Seam Gas

Gas from gas drainage systems may be directed into a goaf area to render the atmosphere inert. Flammable gases are suitable provide that the atmosphere is rendered inert and no ignition source is present.

7.4.3 Mineshield

The "Mineshield" inertisation unit, Figure 27, is kept in readiness for use by NSW Mines Rescue at the Hunter Valley Rescue Station. The equipment kept on this site includes the evaporator units and not the prime movers.

The unit operates by vaporising liquid nitrogen on the mine site. The location at the mine site where the nitrogen is to be delivered requires a hardstand area with sufficient space for B doubles to turn.

Liquid nitrogen is supplied by a contracting firm using road tankers. Each tanker supplies between 20 and 28 tonnes of liquid nitrogen.

The flow rate is variable between 0.5 and 20 tonnes per hour. The long term flow rate is approximately 10 tonnes per hour and is dependent upon the road tankers continuing delivery of the liquid nitrogen. Drivers must be specially licensed and there are limitations on the number of trucks available, and the distance between supply depot and mine site

One (1) tonne of liquid nitrogen equates to 844 m^3 of gaseous nitrogen.

Advantages are:

- Relatively high flow rate
- Nitrogen is an inert gas
- Nitrogen is low temperature
- Sufficient pressure to conduct nitrogen through several km of pipeline

Figure 27: Mineshield inertisation unit

7.4.4 Ambient Air Vapouriser

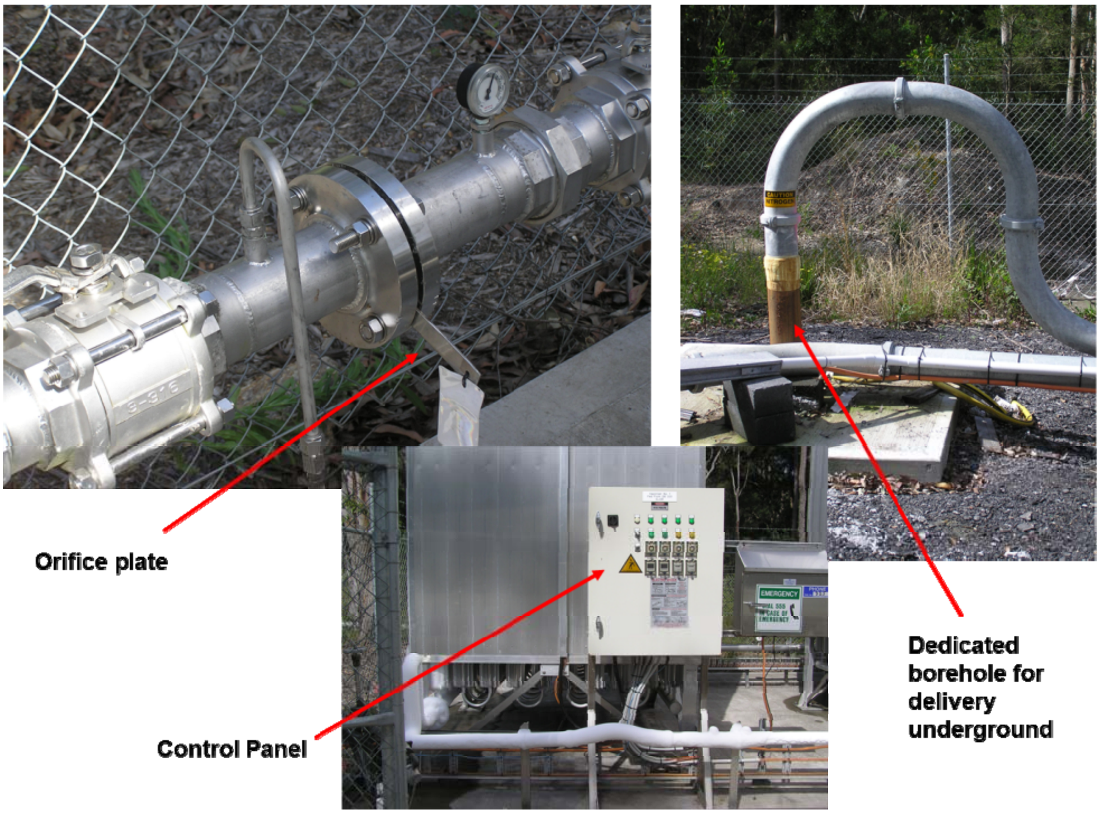
The ambient air vapouriser, Figure 28, is a nitrogen plant that can be set up on the surface of a mine site, perhaps for a longer term solution after the initial requirement was met by the NSW Mine Rescue mobile inertisation unit. A typical unit would have the following specification:

- 2 x 45,000l (30 tonne) liquid Nitrogen storage vessels
- 2 x vaporiser units, one operating and one on standby.
- Fully automated operation.
- Rates controlled by orifice plates, min 0.5t/hr. to max 5t/hr.

A telemetry system monitors operation 24 x 7 and when tanks are depleted they are refilled by a nitrogen supplier. Figure 29 shows the control pane land orifice plate that meters the flow of N_2 down a borehole.

Figure 28: Ambient Air Vapouriser

Figure 29: Control panel and orifice plate for flow rate control



7.4.5 Membrane Separation Nitrogen Generators

An example of a membrane separation nitrogen generator, Figure 30, is the Floxal system. Membrane separation units filter compressed air across hollow polymer membrane fibres, Figure 30, causing nitrogen to separate from oxygen and other components of atmospheric air. The compressed air is dried and filtered. Air temperature is heated 45^0C to maintain a constant temperature

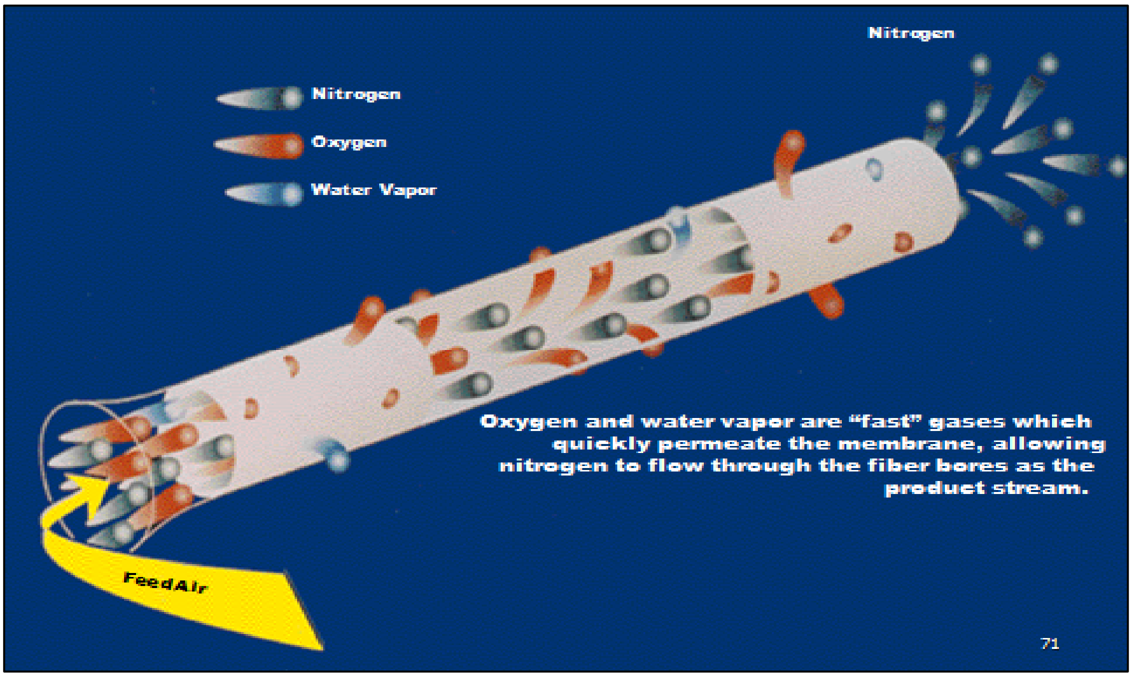
Units have been supplied with capacities of $500m^3$/hr and 1,934 m^3/hr and are available with flow-rates in excess of 2000 m^3/hr. The nitrogen purity is set at 97% for optimum results but can be adjusted to 99%. Unit capacity depends upon the purity of the Nitrogen. For a nominal 1,934m3/hr unit:

- At 98% - capacity is 1517 m3/hr.
- At 97% - capacity is1934 m3/hr.
- At 96% - capacity is 2353 m3/hr.

The unit runs on electricity. No fuel or water is required. A $1,934m^3$/hr. capacity unit requires 3 phase 415 VAC, 805 kW, 981 kVA. The Floxal system does not require an operator. The system starts up and shuts down automatically. Gas is delivered at pressure (230 Kpa reported on trial with a maximum potential of 800Kpa) and can be reticulated over 12.5 Km through a 4”pipe.

Figure 30: Membrane Separation Nitrogen Generator (Floxal)

Figure 31: Membrane Separation Generator technology



7.4.6 Tomlinson boiler

The Tomlinson boiler, Figure 32, produces exhaust gases from a diesel engine that can be discharged into a mine. Diesel usage for 100kw is 200l/ hr.

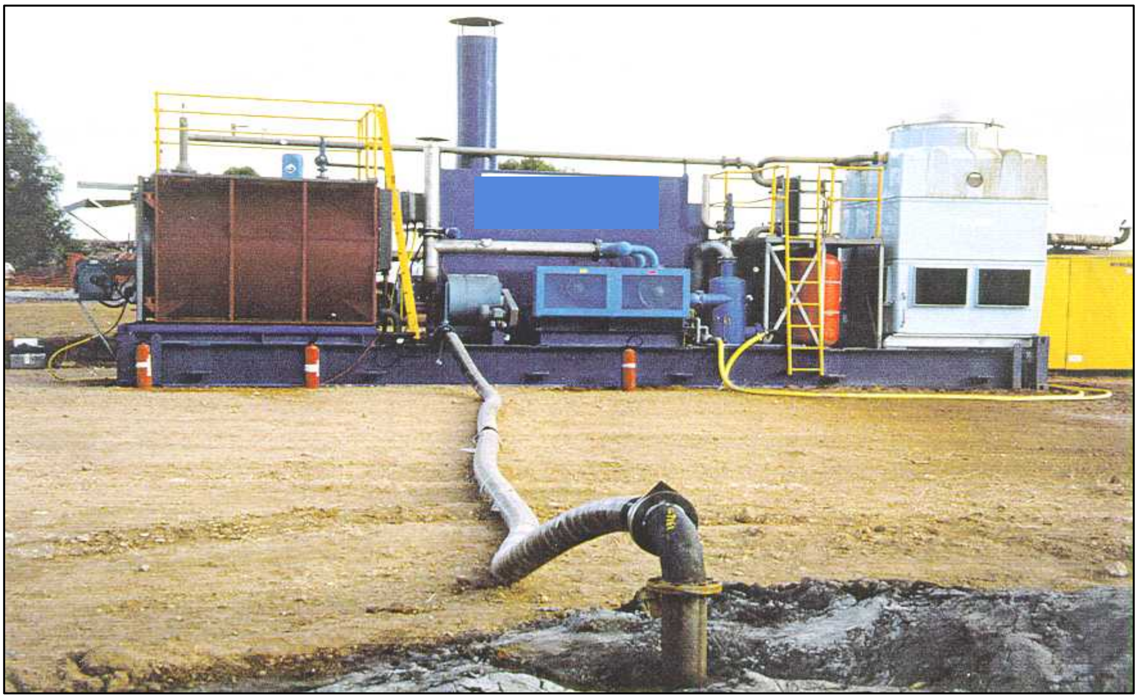
Composition of the exhaust gases is approx.

- 13% CO_2,
- 84% N_2,
- < 2% O_2,

Flow rate is 0.5 m^3/s current with plans to increase to 3 m^3/s. Pressure developed on one trial was reported as a maximum of 10Kpa.

The exhaust gas temperature is about 50^0C.

Figure 32: Tomlinson Boiler unit discharging down a borehole



7.4.7 GAG engine

The GAG engine concept was developed in Poland and demonstrated in Australia in 1997 (QMRS). It has since been used successfully in several mine incidents in Australia and overseas in recent years. Figure 32 shows the GAG engine set up on a pantechnicon for quick transportation and setup.

The exhaust from a jet engine is used to produce the inert gas. Engine capacity is (5 MW) + afterburner. Aviation fuel usage is about 1,500 l/hr. and 66,000 l/hr. water is required.

Flow rate is 25 m^3/s and this is the highest flow of all the inertisation units. It converts 40,000 l/hr. of water to steam. It is estimated that after the water vapour cools and drops out, there is 7m_3/sec of inert gas.

Exhaust gases composition is:

- < 2% O_2,
- CO_2 10% to 15%,
- CO varies but is usually 400 ppm when tuned correctly.

The output temperature is approx. 80^0C. The unit is suitable for inertisation of a mine before sealing but not for re-entry of persons until the high temperature air if flushed with cool fresh air.

Figure 33: GAG engine set up for transportation and use

7.4.8 Pressure Swing Adsorption

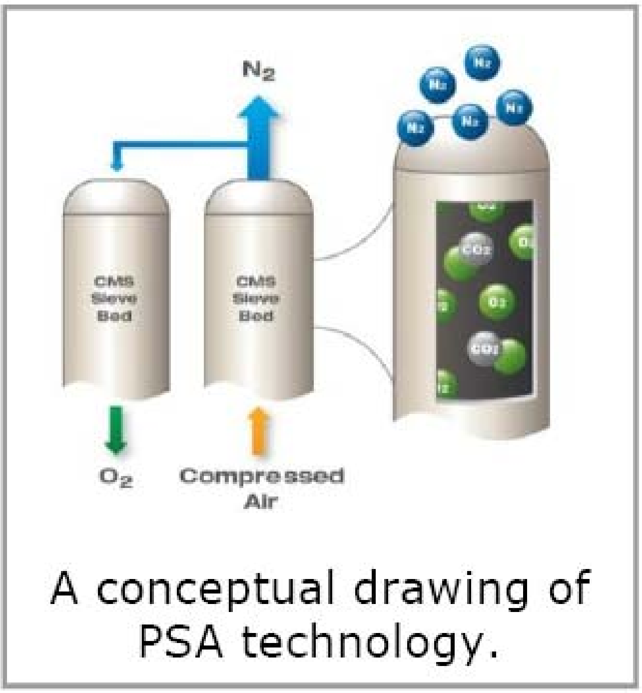
PSA technology uses a carbon sieve material and pressure to adsorb O_2 molecules while allowing N_2 molecules to pass through the sieve material as shown in Figure 33. Compressed air is used to pressurise a vessel filled with sieve material, which sifts the air molecules by physical composition or structure.

Later, the pressure in the sieve bed is reduced, drawing off N_2 molecules and collecting them in the surge tank for use in the application.

A valve is then opened in the sieve bed which releases the remaining pressure, and allows the escape of the O_2 molecules back into the atmosphere (the molecules of the released gas immediately diffuse back into the atmosphere at close to ambient percentages). This cycle is repeated continuously and, with multiple sieve beds working in opposition, a consistent flow of N_2 gas is produced.

A unit tested at the NIOSH Safety Research coal mine at Pittsburgh USA was capable of producing 0.15 m^3/s at a pressure of 335 Kpa. The O_2 content of a sealed area was reduced to 5.4% which was said to be close to the O_2 level produced by the PSA system.

Figure 34: PSA technology



7.5 RAPID SEALING

Provision for rapid sealing of parts of the mine is an important element of a spontaneous combustion management plan.

During the withdrawal process there is the possibility of isolating the part of the mine affected by rapid sealing such as closing doors etc. if this contingency has been foreseen and effective provisions put in place. If not, any actions taken to control or ameliorate the effects of the heating after withdrawal of people will have to be developed and designed and carried out remotely. This may be difficult and time consuming.

If there is surface access to the area above the heating site, the option of sealing the panel or part of the mine is available by using fly ash or other roadway filler. This also allows water or an inert gas to be introduced into the affected area from the surface by means of the Mineshield (nitrogen), Thomlinson boiler, Floxal unit or other means.

When making provision for sealing mine entries, the risk of explosion needs to be considered. Sealing the entries may have to be done without placing personnel in front of the entries where they may be harmed by an explosion.

7.6 REMOTE SEALING

If persons are withdrawn from the mine because of a serious spontaneous combustion event, sealing of the affected part of the mine will allow and facilitate recovery of the remainder of the mine. Techniques for remote sealing include:

- Injection of fly ash through boreholes
- Injection of roadway filler materials such as “Rocsil”
- Inflatable seals
- Remotely operated fire doors

A number of proprietary products are available for roadway filling, inflatable seals and remotely operated doors. Some are described here. Users are advised to research the specifications of these products and satisfy themselves as to the suitable application for the task.

7.6.1 Fly Ash

Fly ash is a by-product of coal combustion. It is a fine powder, light to dark grey in colour. Boiling/ melting point is > 1400^0 C. Specific gravity is 2.05 to 2.8. It is non-flammable. Approx. 20% to 40% of particles are below 7 microns in diameter. The material is composed primarily of complex aluminosilicate glass, mullite, hematite, magnetite spinel and quartz. Silica-crystalline as quartz is 1 – 5% and mullite 1 – 5%. It does not decompose on heating.

The Fly ash is readily available from Power stations and can be injected through boreholes to underground roadways in a wet or dry state. It has been used successfully in both forms in a number of events.

The angle of repose of fly ash when taken straight out the power station is about 3 degrees from the horizontal. When the ash has cooled and taken up some moisture the angle of repose is about 11 degrees from the horizontal.

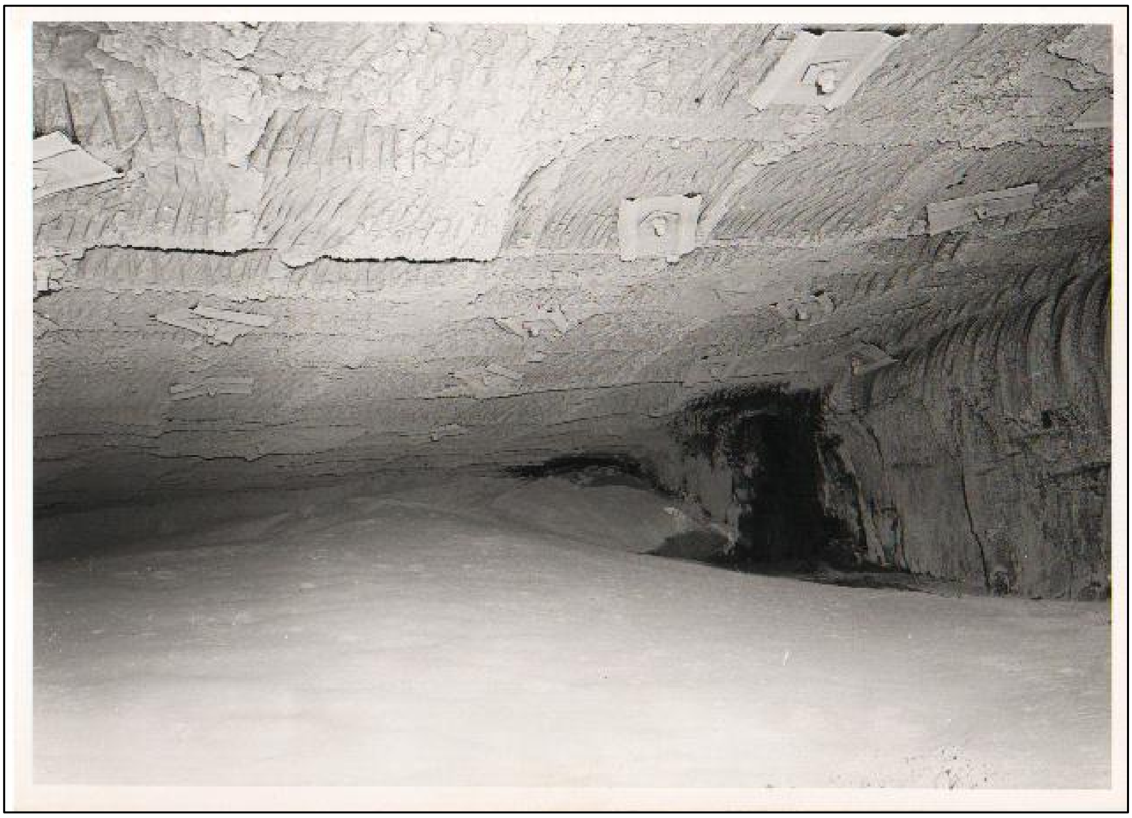
Fly ash can be placed dry in a way that gets the angle of repose up to 40 degrees from the horizontal and still get a very good seal in the roadway. This is achieved by first putting down the hole about 40,000 l of water. Dry fly ash is then pumped into the roadway as seen in Figure 34. Another 5,000l of water is placed and then more fly ash. This causes the ash to bank up on a steeper angle of repose.

For a roadway 3.5m high and 5.2m wide, on a level course, approx. 400 tonnes of fly ash would be required to plug the roadway.

Fly ash placed wet is first pre-mixed in a slurry plant as shown in Figure 35, to achieve the optimum pulp density for pumping down the borehole. There is less chance of blocking and wet fly ash can be pumped a greater distance.

Figure 35: Fly ash wet slurry plant

Figure 36: Fly ash seal in underground roadway at Moura #4



7.6.2 Roadway Filler Material

Examples of roadway filler materials are the Rocsil and Carbofill products. To fill a roadway via a borehole, two hoses are attached to a catenary wire and lowered into the borehole. Nozzle heads and check valves on hoses discharge two chemicals into the roadway. As the two chemicals mix the foam expands at about 10 to 1 and then to 35 to 1 as it sets. The phenolic foam sets to ultimate strength in about 5 minutes. Bulkheads or barriers are not required in the roadway to contain the foam.

The plug formed is estimated to be 5 to 6m wide. A description of the material flow properties is that it flows like lava, i.e., flows and sets with fresh material building on that previously discharged and set. Material strength when set is about 2 mpa. It is a sealant that should fill the roadway without voids. The material does not support combustion and has been used extensively for cavity filling and the control of spontaneous combustion.

7.6.3 Inflatable Seals

The Shaft Plug Void Sealing System (VSS), as shown in Figure 6, is designed to provide emergency and short-term sealing of an intake or exhaust shaft. The Shaft Plug can be installed remotely using a long boom crane and is also suitable for horizontal or inclined applications.

Figure 37: Inflatable Shaft Seal

The Ventstop ventilation control unit for use in underground roadways, as shown in Figure 38, has these features:

- Suitable for any size or shape of roadway

- Portable and re-usable
- Available in standard or FRAS (Fire Retardant Anti-Static) fabrics
- Continuous air trickle or bottle feed
- Available with sleeves through the seal

Figure 38: Inflatable roadway seal - Ventstop

Both the inflatable shaft seal and roadway seal require periodic topping up with compressed air to maintain the seal. In this regard they should be regarded as a short term solution. For a longer term solution, an option is to fill the bag with a foam material.

7.6.4 Remotely operated Doors

Doors that are capable of being remotely operated to close off an airway in the event of an emergency are available from a number of manufacturers. Issues in the successful design and operation of these doors include:

- Surface access
- Energy sources and means to operate doors remotely in an emergency
- Elimination of interference with door closure due to services in the roadway

The following Figures 39, 40 and 41, illustrate a system of remote door operation installed in a Queensland mine.

Figure 39: Remotely operated door

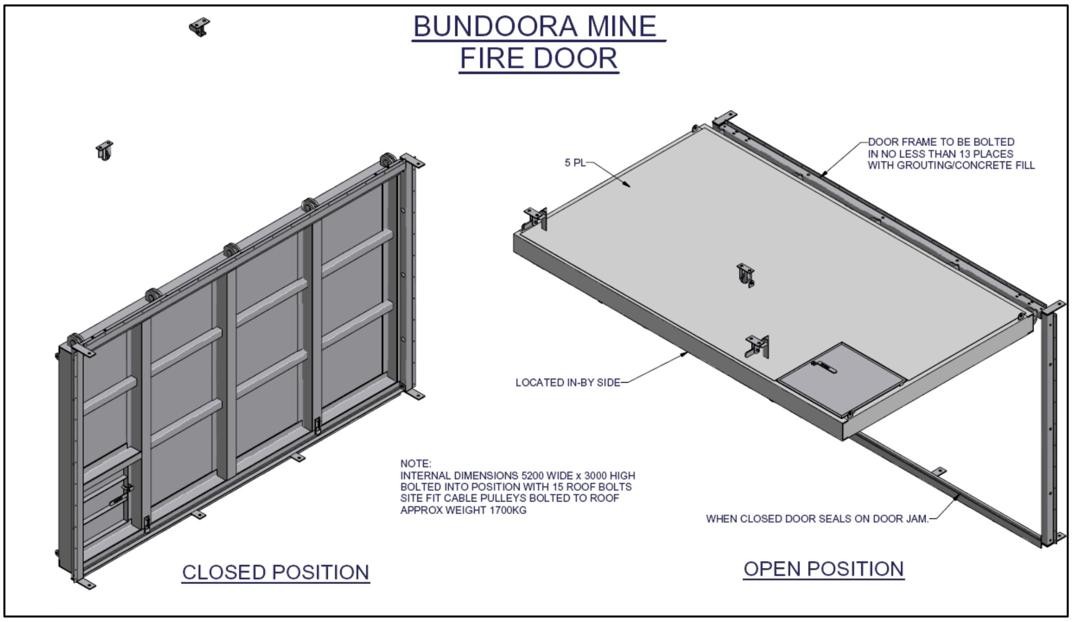


Figure 40: Remote door arrangement in an underground mine

Attached to the door is a manual winch, operated from the surface of the mine but which can also be configured to be operated from a location within the mine such as an outbye cut-through etc. The door resides in the open position & if required, (emergency), the winch firstly lifts the door slightly & the roof latches let go & allow the self-locking door to be lowered to seal the area.

Figure 41: Surface located winch to activate door

8 APPENDICES

8.1 EVENTS

There have been in excess of 125 incidents reported in NSW since 1960, most occurring in the Greta seam and Liddell seam. In Queensland, there have been in excess of 68 incidents since 1960. The following are some of the more serious or unusual events that have been documented.

An outbreak of spontaneous combustion is a potentially very serious event which can result in the following underground hazards causing harm to people:

- Fire
- Explosion
- Toxic gases
- Heat and humidity
- Poor visibility

The following events demonstrate the serious nature of heatings and different types. Common elements for a number of these incidents that should be considered in the development of a spontaneous combustion management plan are:

- Early warning signs were not detected or not acted upon
- Heatings were often first detected by a Deputy conducting a routine inspection
- Once detected, the development of the heating was very rapid

8.1.1 North Tunnel - 1970

A fall of top coal took place and the fallen coal was discovered to be on fire. Mines rescue teams were summoned and, using breathing apparatus, extinguished the flames and cooled the fallen coal.

The Greta seam has a high propensity for spontaneous combustion and there have been many spontaneous combustion events.

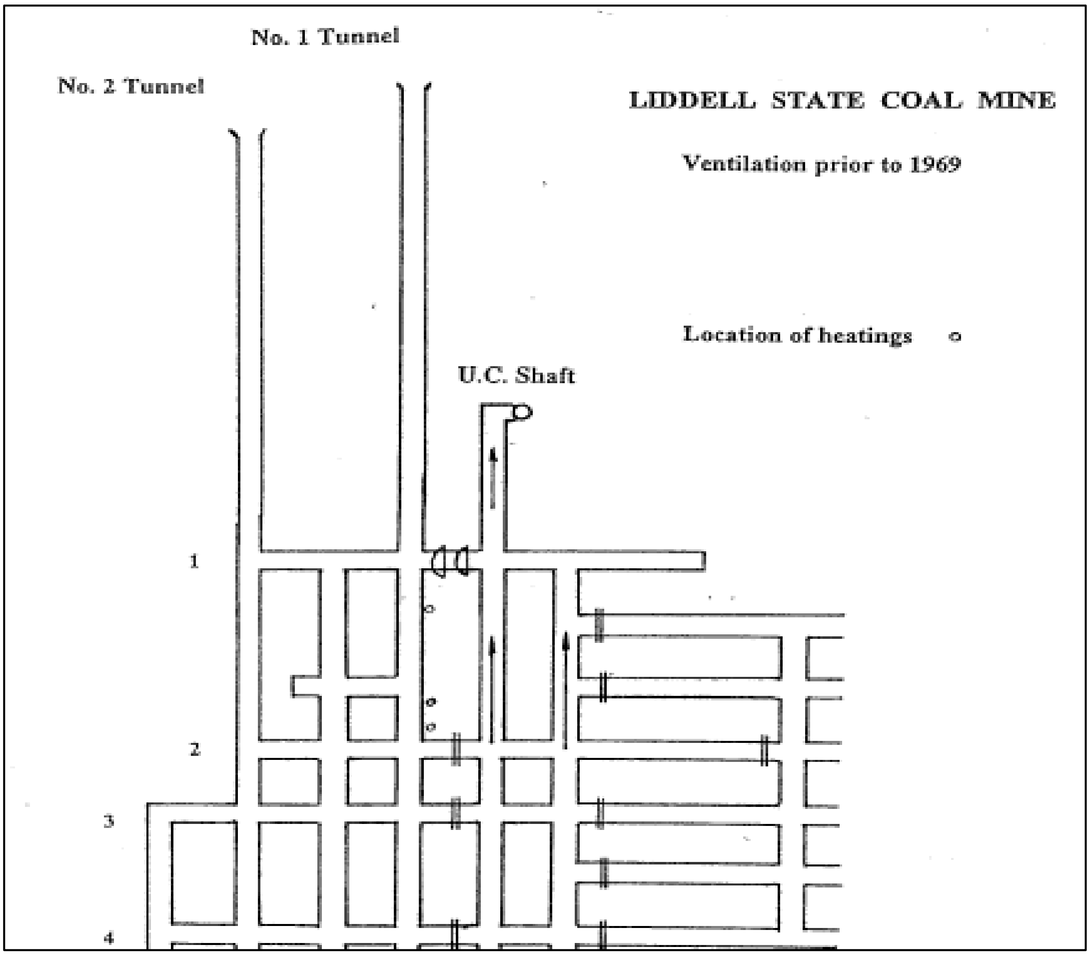
The roadway had been driven to 3m height with approx. 4m top coal supported with timber cross supports and props.

It was readily apparent that the heating had taken place in the top coal before the fall took place. The tops burst into flames after falling. Mine rescue teams report the fire hose discharge turned to steam when directed to the sides of the roof cavity.

8.1.2 Liddell - Oct 1971

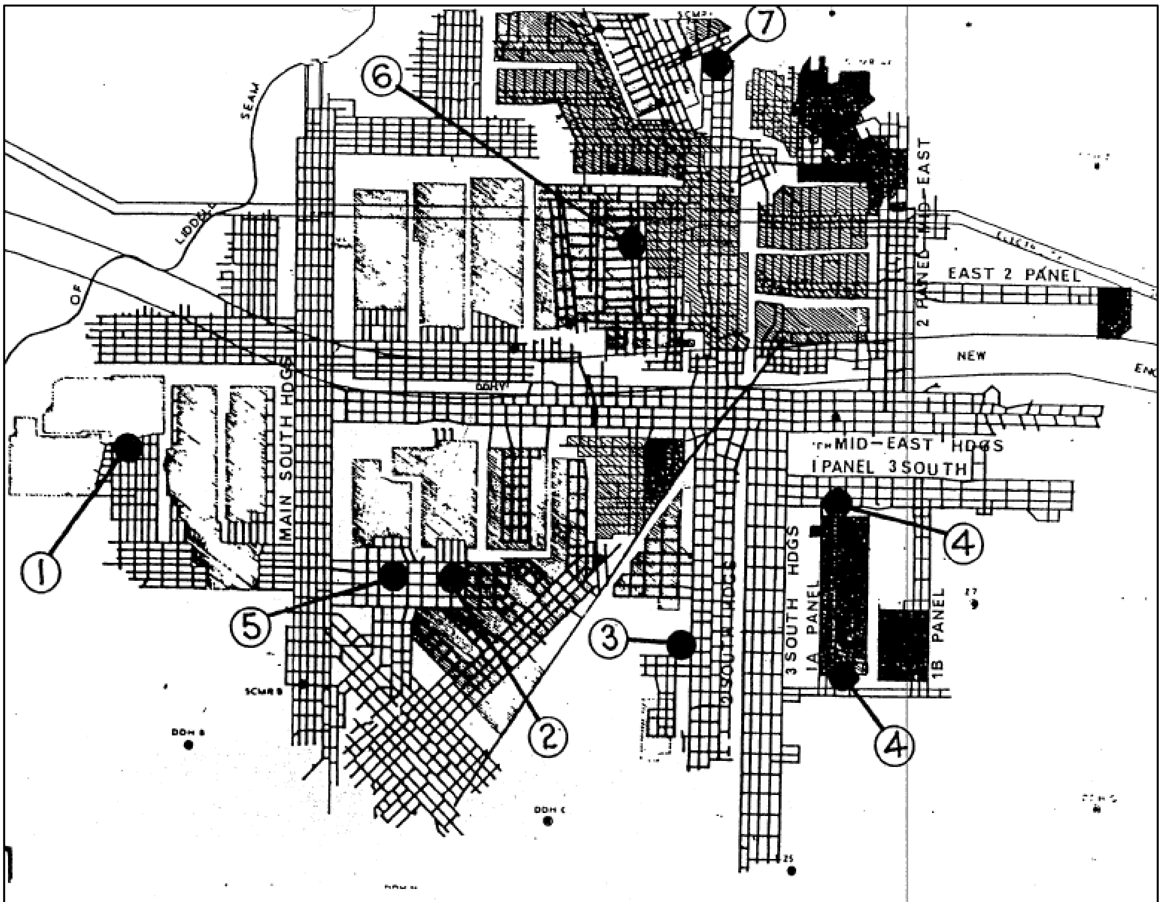
The seam mined was the Liddell. A number of heatings had occurred in the mine. Two heatings in bord and pillar extraction panels resulted in the panels being sealed. Roof falls invariably contained 0.5m or more of top coal and were another source of heatings. The background level for CO was around 3ppm and a concentration of 8ppm usually indicated a heating.

Prior to 1969 there were several heatings in the ribsides of pillars separating the main intake and return. These were dug out and the pillar treated with water infusion. An attempt was made to affect a more permanent solution by balancing the pressures around the pillar. No 2 Heading return was placed on intake pressure.



In September 1969 another heating took place in the ribside of the pillar between the intake and return. The heating was dug out and the pillar infused with water. No 2 Heading was sealed off completely between No 1 and No 4 c/t's and daily inspections and regular sampling carried out.

In October 1969 smoke issued from the edges of the seals and the carbon monoxide concentration rose from 1.04% to 2.53% on successive days. The temperature was 44^0 C. Leaks around the seals were repaired, the pillar ribs gunited and carbon dioxide injected into the area. There was improvement over the following weeks but the signs that heating was not under control. A rescue team entered the area found a heating in the ribside showing as white ash and a red glow. The seals in No 2 Heading were breached and a rescue team hosed and dug out the heating. Again water infusion was carried out.



LIDDELL STATE COAL MINE
SECTION HEATING LOCATIONS
1964 to 1973

In December 1969 a new return airway was driven and the pillars between No 1 and 2 Headings were put on intake pressure (Fig 4). Inspections and monitoring were carried out on a close regular basis. For 22 months there were no indications that anything was other than normal until a fire broke out on 21 Oct 1971. The mine was evacuated and sealed.

A pre-shift inspection of the mine at 9.30 pm on Sunday 24/10/71 determined nothing abnormal. A Deputy later said that there was some haze that he thought was diesel smoke in the transport road. The first transport left the surface at 12.10am on Monday 25/10/71 and at No 5 c/t the driver stopped the transport when he noticed an unusual smell. Smoke was found issuing from No 1 and 3 c/t's into the belt heading but was too thick to allow entry to locate the source.

Brattice stoppings were erected across the main intakes below No 1 c/t and a hole in the No 3 c/t stopping enlarged to clear the smoke. The smoke was forced back to the second ventilation door at No 1 c/t but was never cleared beyond that point. Fire hoses at this stage were being directed at the smoke.

At 1.00am the concentration at the fan was 10ppm but there was no fire smell. At 2.30am some burning material was seen falling from the roof at the intersection of No 2 Heading and No 1 c/t and it was obvious that a fire was located in the top coal.

At 4.00am the CO level at the fan had risen to 50ppm and the situation was worsening. A rescue team was sent to open a stopping at No 4 c/t to short circuit the ventilation. The team had just left the FAB when a fall occurred outbye flooding the FAB with dense smoke and catching the standby team uncoupled. Both teams eventually retreated in nil visibility across the No 2 c/t to No 5 Heading and fresh air. The fall had occurred in the intersection of No 1 c/t and No 2 Heading and was a mass of flame.

Attempts to fight the fire with water and foam were not successful. During a 40 minute period when foam ran out, results improved. Variations to ventilation made no improvement. There was a sudden increase in black smoke at the fan shaft. The heavy smoke from the fan was soon followed by flames rising to a height of about 15-20 metres. Soon after, the fan stopped and, the fan building collapsed. The mine was then sealed.

Contributing factors to the heating that occurred near the entries of the mine were determined to be:

- General nature of the Liddell seam coal
- Porous nature of the coal, particularly near the outcrop
- Pressure difference between intake and return (approx. 500 pascals)
- Relatively small size of pillars between intake and return (22m)

8.1.3 North Tunnel - 1975

The heating developed in the goaf of a bord and pillar extraction panel.

The Greta seam ranges in thickness from 7m to 10m. The bottom section was mined on development for reasons of coal quality and some top coal recovered during the extraction process. Generally, methane is not detected in the Greta seam up to a depth of cover of about 300m. Methane and other explosive gases are produced by the distillation of coal.

Attempts were made to construct 6 seals to isolate the goaf where the heating took place. During this process three small explosions in the goaf area took place. This caused the sealing of the panel to be abandoned and the mine was sealed at the entries. It was subsequently re-entered and production resumed.

8.1.4 Kianga No.1 - Sep 1975

About 5.10pm on Saturday, September 20th, 1975, an explosion occurred in the Kianga No. 1 underground mine. Thirteen men lost their lives. The men were engaged in sealing a heating in the No. 4 section of the mine at the time of the explosion. The magnitude of the explosion was such that sections of the main conveyor were blown out of the mine. Belt rollers were blown 200m to 300m from the tunnel mouth.

A deputy commencing a pre-shift inspection on 20th September entered the return at 7.30am and noticed a slight haze. He walked inbye for 2 pillars without noticing anything unusual and then returned to the surface to take observations at the fan. He saw no smoke but attributed a fire stink to a fire that had occurred previously in a bolter shunt outbye of the fan shaft. Nevertheless he was still suspicious and immediately reported to the manager.

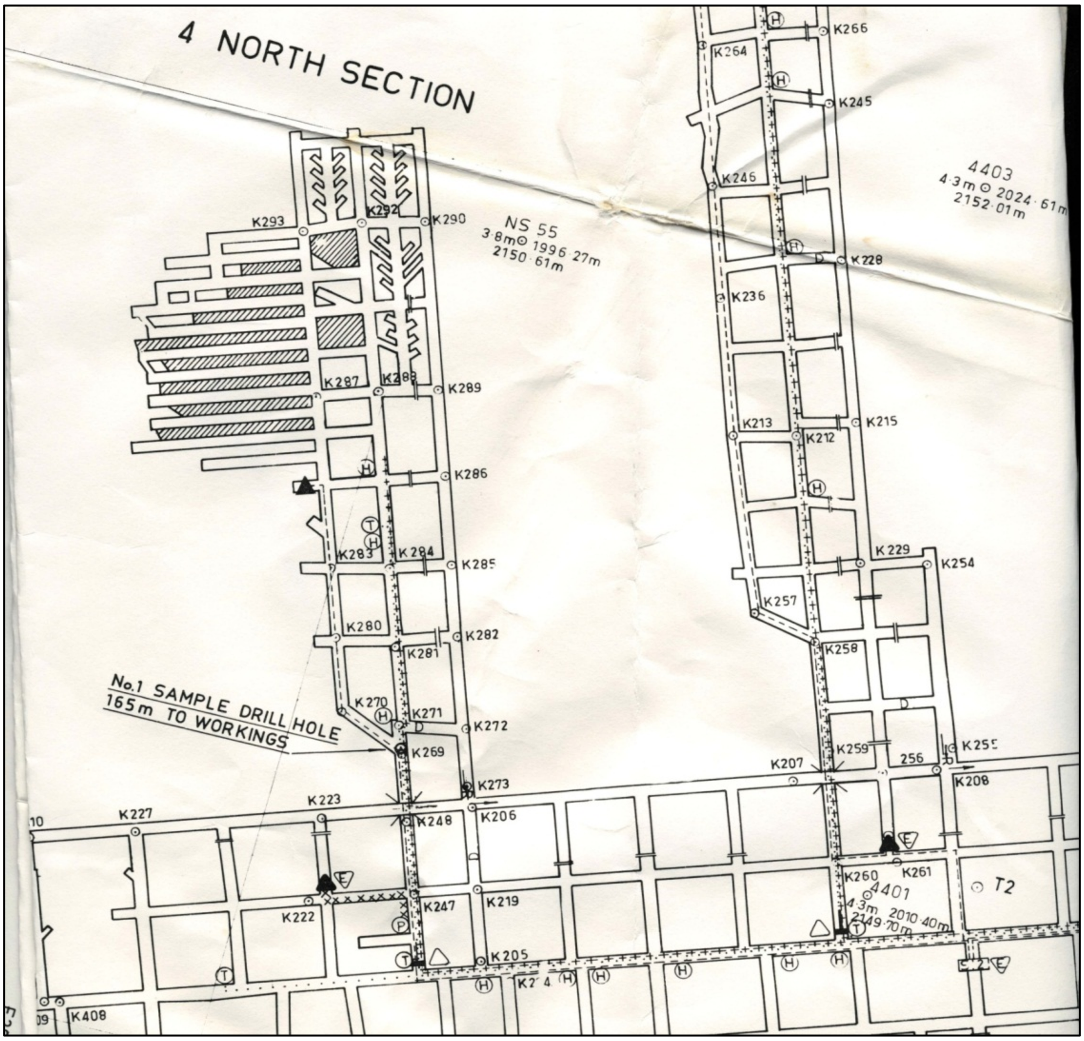
The manager and the deputy went underground to 2 North return and then 4 North where, in the return, smoke was obvious and the fire stink smell more obvious. Gas readings were taken with 25ppm CO being detected.

Construction of the brick seals commenced at 11.30am. Readings of 80ppm CO and 1% CH_4 were noted. At about 5.10pm, a popping sound was heard, lights flickered and an explosion took place.

Approx. 3m of the bottom section of a 4.2m seam was mined by the use of continuous miners. The seam was gassy and liable to spontaneous combustion. The goaf was partially ventilated. Eight (8) rows of pillars had been developed and 3 rows of pillars extracted. This was about 6 months work.

Evidence of a heating had been discovered in the goaf area of 4 North. There appears to have been a large body of methane in the goaf with 3% to 4% found at the edge of the goaf at 7 c/t. Over the 6hr period prior to the explosion, a barometric drop of not less than 5 millibars occurred.

The use at the mine of a Beckman gas analyser was a considerable improvement on methods generally in use in Queensland at the time. The normal signs of sweating, fire stink and haze were not reported in 4 North return prior to the discovery of smoke.



8.1.5 Leichardt Colliery - Dec 1981

Leichardt Colliery is located near Blackwater in Queensland. Mine entries were two vertical shafts equipped with winders. The mine had a history of outbursts and high methane emissions. (CH_4 16m3/ tonne) The 6m thick Gemini seam was mined at a depth of cover of about 400m. Mining in the Gemini seam commenced in 1969 and spontaneous combustions problems were not experienced until 1981.

On 6.20am on 29th December 1981, a Deputy on a pre-shift inspection discovered a smoke haze at the pit bottom of No. 2 Shaft. Further investigation revealed thick smoke coming from the main east return and that 10ppm CO was present in the return shaft.

At 11.30am on 29th December, the IMT group decided no further action would be taken until a detailed analysis of the atmosphere was available. Gas samples were dispatched to Brisbane and ACIRL asked to supply a gas chromatograph.

At about 8.40pm, the first results were obtained from Brisbane. The chromatograph was damaged in transit and failed to function for about 9 hours.

On the basis of the gas results, a rescue team was send underground at 12 noon to establish the source of the heating. The first team entering the mine detected thick smoke and about 800 ppm CO at No 4 c/t on the east belt road. Smoke reduced visibility to an intolerable level and the team retreated to the adjacent track road. The team continued inbye to No 5 c/t and again encountered thick smoke and high temperature.

An exploration across No 5 c/t revealed the source of the heating in a pile of slack coal in a stub end in the right hand rib of the belt road inbye No 5 c/t. Open flame was visible in a number of areas over a distance of about 5m. They determined the fire was poorly ventilated but relatively stable and the gases produced were non-explosive.

Mine rescue teams fought the fire during the remainder of the day. At midnight, the area was declared safe and they continued to hose down the heated zone for the following 12 hours before the slack coal was loaded out. This was completed by 4.30am on 31st December 1981.

8.1.6 Laleham No.1 1982

Laleham Colliery is located about 40km south of Blackwater in Queensland. The 3.7m thick Pollux seam is worked. During 1974 and 1975, serious heatings were experienced in pillars between main intake and return roadways and in July 1975, a serious heating was detected in the goaf of a pillar extraction panel.

In 4th May 1982, a Deputy on a routine inspection of outbye intake roadways discovered smoke and 150 ppm CO on a Drager tube. Subsequent investigations revealed dense smoke and CO in excess of 3000 ppm issuing from an inaccessible area of intake roadway.

This event resulted in the closure of the mine for about seven days and required a major reorganisation of the ventilation network which was not completed until 25th September 1982.

Monitoring showed that the situation was very serious with the atmosphere in the sealed area either explosive or potentially explosive on number of occasions. The major flammable contribution to the explosive atmosphere was hydrogen which reached a peak of 4.57% on 8th May 1982. This abnormally high concentration may have been due to the injection of water onto the heated zone which may have produced water gas.

Attempts were made to inject water into the barrier pillar between the approx. site of the heating and the main intake roadway. At the same time, attempts were made to gain access to the site by loading out a fall. This was abandoned due to boggy floor, heavy roof and a clear indication the fire area was increasing. It was then decided to seal the area. While this was being done, four long holes were drilled through the barrier pillar and these were connected with hoses for water infusion.

The final sealing was accomplished on 5th May 1982. By this time, the atmosphere coming from the fire area contained thick black smoke a strong tarry odour.

Holes were drilled from the surface to intersect roadways affected by the fire, and to fill the voids with concrete slurry. Despite 1,562 m^3 of concrete slurry being used to affect a seal, high levels of CO were still being produced.

An attempt was made to inject the pillar with bentonite grout and concrete slurry and when this proved unsuccessful, a further six holes were drilled from the surface to fill any remaining voids with fly ash. Although a total of 360 tonnes of fly ash was used, high levels of CO continued to be liberated from cracks in the pillar. These areas were treated by the injection of bentonite and cement grout. The problem was not entirely solved until the pressure differential was removed on 25th September 1982.

8.1.7 Newstan - 1982

The heating took place in a bord and pillar extraction panel that had been completed and sealed for some time. The borehole seam was overlain by the Dudley seam in close proximity. Inter-burden between the borehole and Dudley failed over the seals and allowed air ingress.

Additional stoppings were placed outbye where seals had failed to control the heating

8.1.8 Moura No.2 - Apr 1986

Moura No 2 mine is located in the Moura coalfield in the south eastern part of the Bowen Basin in Queensland. There are five economically exploitable seams, varying from about 2.1m to 7m in thickness. Seam thickness in the “D” seam worked varies from 2.4m to greater than 6m. Extraction panels are driven off main headings and several methods of extraction by continuous miners are employed, including mining of bottom coal and partial extraction.

At 6.40 am on 19th April 1986, a Deputy on a routine inspection of the face area of 5NW pillar extraction unit sampled 13ppm CO and 0.09% CO_2. At the same time, the mine monitoring system recorded 12 ppm CO at a monitoring point about 800m outbye the face of 5NW. Determinations made closer to the goaf edge detected 40 ppm and a slight haze.

At 11.45 am, a “non-typical” gob stink was noted with a definite smoke haze visible in the beam of a cap lamp. By about 2.15pm, 90 ppm CO was detected at the goaf edge, the smoke haze was heavier and a gob stink clearly evident.

Sealing of the area was affected by bricking up the openings in four preparatory seals. This was completed by 5.10am on 19th April 1986. All men were then withdrawn from the mine. During sealing operations, gases were monitored. The highest level of CO detected in the east return was 150 ppm. Monitoring of the atmosphere behind the seals was not possible and the mine was shut down until 5.30am the next day when an inspection revealed the atmosphere has passed through the explosive range.

Monitoring of the atmosphere behind the seals continued over the weeks that followed and indications were that the area was stable. A seal was breached on 10th May 1986 and a rescue team entered via an airlock. After advancing about 200m they reported CO levels in excess of 3,000 ppm. A tube bundle line was advanced to this point and the team retreated. Monitoring of the atmosphere continued during the following weeks and when the CO level dropped to about 1100 ppm, a second attempt was made on 24th May 1986.

Rescue teams constructed brattice stoppings immediately outbye of the goaf edge and the panel was ready for re-ventilation on 2nd June 1986.

After a detailed inspection by rescue teams, the seals were breached on 2nd June 1986. A team of 28 miners, fitters and electricians began a well-executed recovery operation which was completed by 7.15am on 3rd June 1986.

8.1.9 New Hope - Jun 1989

New Hope No.1 mine mined the Bluff seam using bord and pillar methods with the splitting of pillars and the taking of the bottoms on retreat. The seam is around 9.1 metres thick and dips at 1 in 2.8. The seam contains a low level of methane.

The primary indicator of spontaneous combustion was the detection of CO. Background levels in panel returns were typically 1 to 2ppm. The mine had installed a Maihak UNOR 6N analyser with a tube bundle system. This was augmented by hand held monitors.

On Wednesday 31 May 1989, the afternoon shift deputy noticed a reading of 4 ppm of CO in the return for WL1 A section. He established that this was a continuous reading and not due to diesel vehicle emissions. He noted this in his report but took no other action.

On 1 June 1989 the day shift deputy noted the monitor was showing 6 ppm. Further inspection revealed 30 ppm CO immediately after passing through the stoppings. As they went further into the panel they found the O2 content of the air was decreasing evenly across all three roadways and that CO content was rising. This rise was greatest in the belt road and three pillars inbye of the stoppings 45 ppm CO was found. By 10 am the CO content in the supply road had risen to 45 ppm. Men were withdrawn from the panel and Flygty seals placed.

On Monday 5 June, the atmosphere was 89 ppm CO and 17.9 % O_2. The situation seemed stable. Men were detailed to erect permanent brick stoppings directly outbye the flygty stoppings. Falls were occurring in the waste workings which were thought to cause damage to the flygty seals.

On Tuesday 101 ppm CO and 16.6 % O_2, was recorded and on Wednesday, readings were 130 ppm CO and 16.1 % O_2. Mining in the WL1 B section continued and the erection of brick seals progressed.

Samples collected at 7 am on Thursday 8 June indicated 66 ppm H_2. Production in 1B was stopped and all efforts were devoted to completing the brick seals. The stoppings were completed by midday Friday and production recommenced in 1B section. At 4:30 pm the GC results indicated 250 ppm CO and 200 ppm H_2 with 14.4% 02. Because of the rapid rise in H_2 all men were withdrawn from the mine.

By 14 June the oxygen concentration had fallen to 11.5% and the CO was 389 ppm and H_2 was 307 ppm. Men re-entered the mine and brick stoppings were bond-creted to ensure that the seals were air tight. As the fire gas and oxygen concentrations fell the frequency of sampling decreased until gas chromatographic analysis was discontinued on 4 July. No evidence of an activity has since been detected.

8.1.10 Lemington - Jan 1991

A spontaneous combustion event took place in the goaf area of panel 131. There was no seam gas and the working height was between 3 and 4 metres. The 6 heading panel had 3 intakes and 3 returns, with flanking returns.

The fire was caused by spontaneous combustion and activated when mining recommenced after a two month break in production. The area was eventually sealed by a line of stoppings a pillar length outbye the goaf edge. An “inert-rich” atmosphere developed within the sealed area which extinguished the fire.

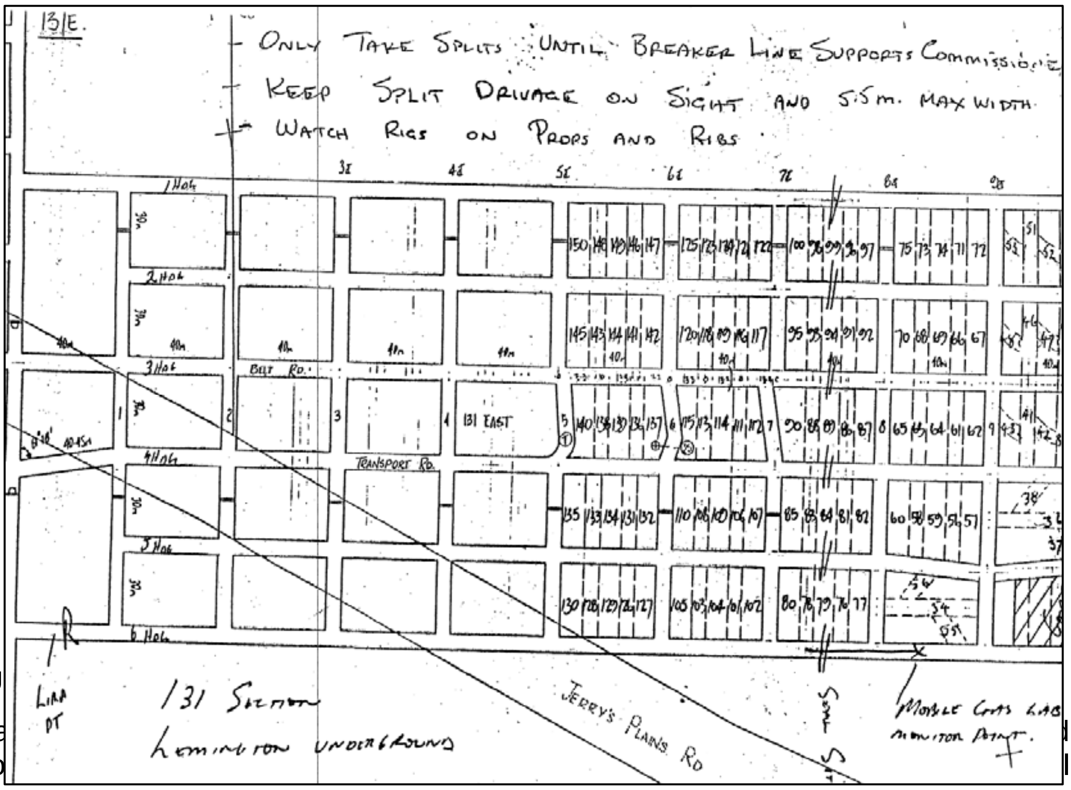
The mine was evacuated during the crucial phase when remote tube monitoring indicated that the sealed area atmosphere passed through the explosive range.

Events reported were:

- Sep 3rd 1990 Pillar extraction in 131 east panel ceased when the continuous miner was buried in a goaf fall

- Sep 17th 1990 A continuous miner was set up in the section. Mine. Ventilation was increased to levels required by statutory limits. 50ppm CO was detected in the return at the goaf edge. The Lira tube bundle system showed no cause for alarm.
- Jan 18th 1991 Production recommenced on afternoon shift splitting pillars. Low levels of CO were detected at the goaf edge, peaking at 80ppm
- Jan 21st 1991 Production on 3 shifts. Low levels of CO (70ppm) were detected in the return at the goaf edge.
Jan 22nd 1991 Production on 3 shifts.
- Low levels of CO. (70ppm) detected in the return at the goaf edge. At 4.00am, the Lira detected CO above the background level (22ppm)

- Jan 23rd 1991 Production on 3 shifts. Low levels of CO (70ppm) were detected in the return at the goaf edge. CO incursion on afternoon shift, increase by a significant fall in the barometer.
- Jan 24th 1991 At midnight, heavy smoke was detected in 6 heading return. Production did not recommence. Men were withdrawn to a fresh air area. At 7.00am they decided to seal the area. Seal construction commenced at 11.00am and was completed by 9.25pm.Continuous monitoring within the sealed area by the mobile lab commenced at 5.00pm. All men out of the mine by 10.05pm.
- Jan 26th 1991 Monitoring indicated the area had passed through the explosive range.
- Jan 28th 1991 Pre-shift inspection of the mine with a view to restoring power. Bag samples were taken from the sealed area. Maintenance work commenced to resume production.



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8.1.11 U

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that heating through improved sealing of the bleeder roadways when another major event occurred.

On 9th July 1991, a rise in oxygen was noted behind seal 23 and a rise in CO. The rate of seal repairs increased. On 4th August, H2 was first detected.

On 7th August, 2250ppm CO was detected in the goaf and H_2 increased to 0.25%.

On 8th August 1991, at approximately 6.15pm smoke was noticed on longwall 5 face and a red glow reflecting on the coal rib-side was observe> 3000ppm, 2% CO & 2% CH_4 in Longwall 6 tailgate at 21 c/t. At 6.25pm, Drager readings at L2 6 were 7000ppm CO and 4% H_2. At 6.30pm there was an alarm at the fan. Evacuation of employees commenced at 6.40pm. At 7.55pm, Drager readings at the fan were CO > 3000ppm, 2%CO_2 & 2% CH_4.

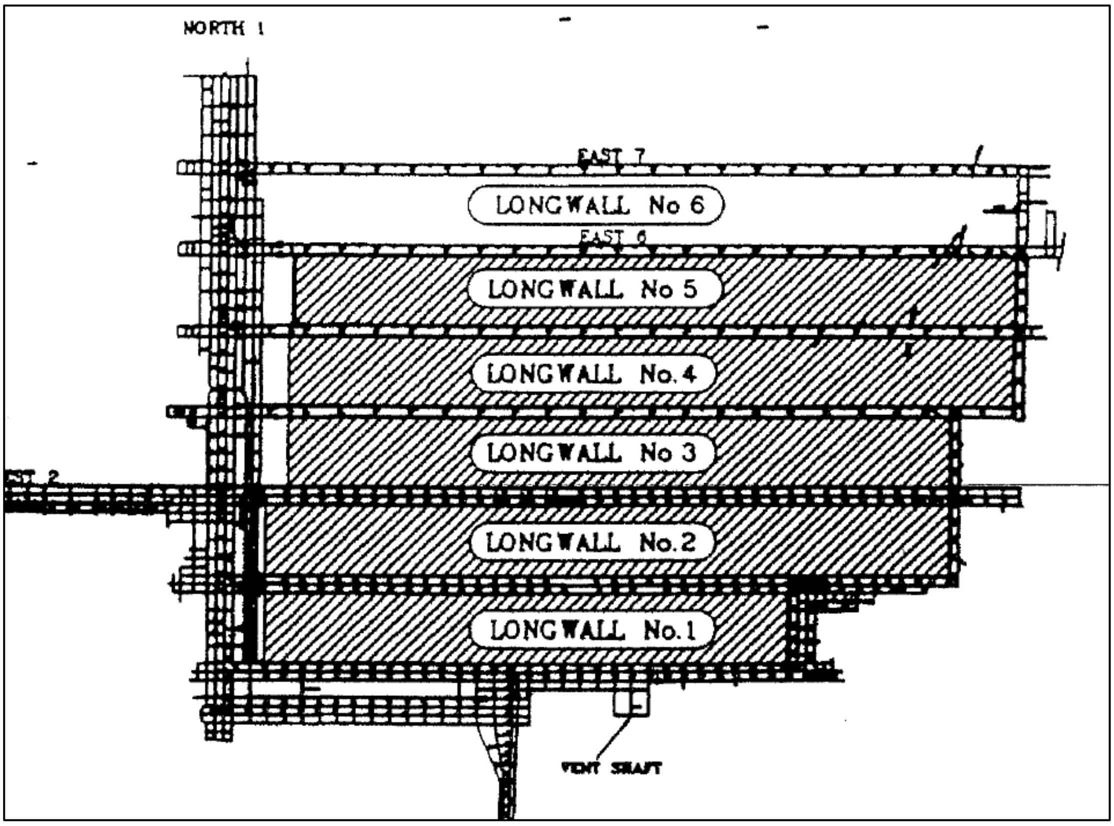
The mine was then sealed. Subsequently, the area suspected of heating was flooded and the atmosphere inertised by the introduction of gaseous nitrogen.

The Ulan seam was believed to have a low liability to spontaneous combustion. There was no seam gas. The bottom 3m section of the 10 to 14m thick seam was worked. Gate road stoppings were constructed from plasterboard.

The main contributing factors to the heating were considered to be:

- Lack of appreciation of the liability of the seam to spontaneous combustion
- Inappropriate ventilation layout
- Lack of understanding of spontaneous combustion initiation
- Incorrect interpretation/ analysis of monitoring results
- Insufficient monitoring information
- Inadequate ventilation standards
- Lack of pre-determined action plan
- Unclear definition of responsibilities

The mine resumed operations in March 1992.



8.1.12 Huntly West, New Zealand – Sep. 1992

Huntly West is a State owned mine developed in the Waikato region of New Zealand. The Kupakupa seam is mined and is a sub-bituminous coal with a very high propensity to spontaneous combustion. R70 = 10 to 16.5.

Depth of cover is approx. 300m. The seam is up to 6m thick with an undulating pavement and a number of structures. The coal deposit has varying thickness and roof and floor gradients. The roof and floor lithology is weaker than the coal and the optimum roadway stability is achieved with a coal roof and floor. Methane drainage is practiced and the returns contain 0.4% CH_4

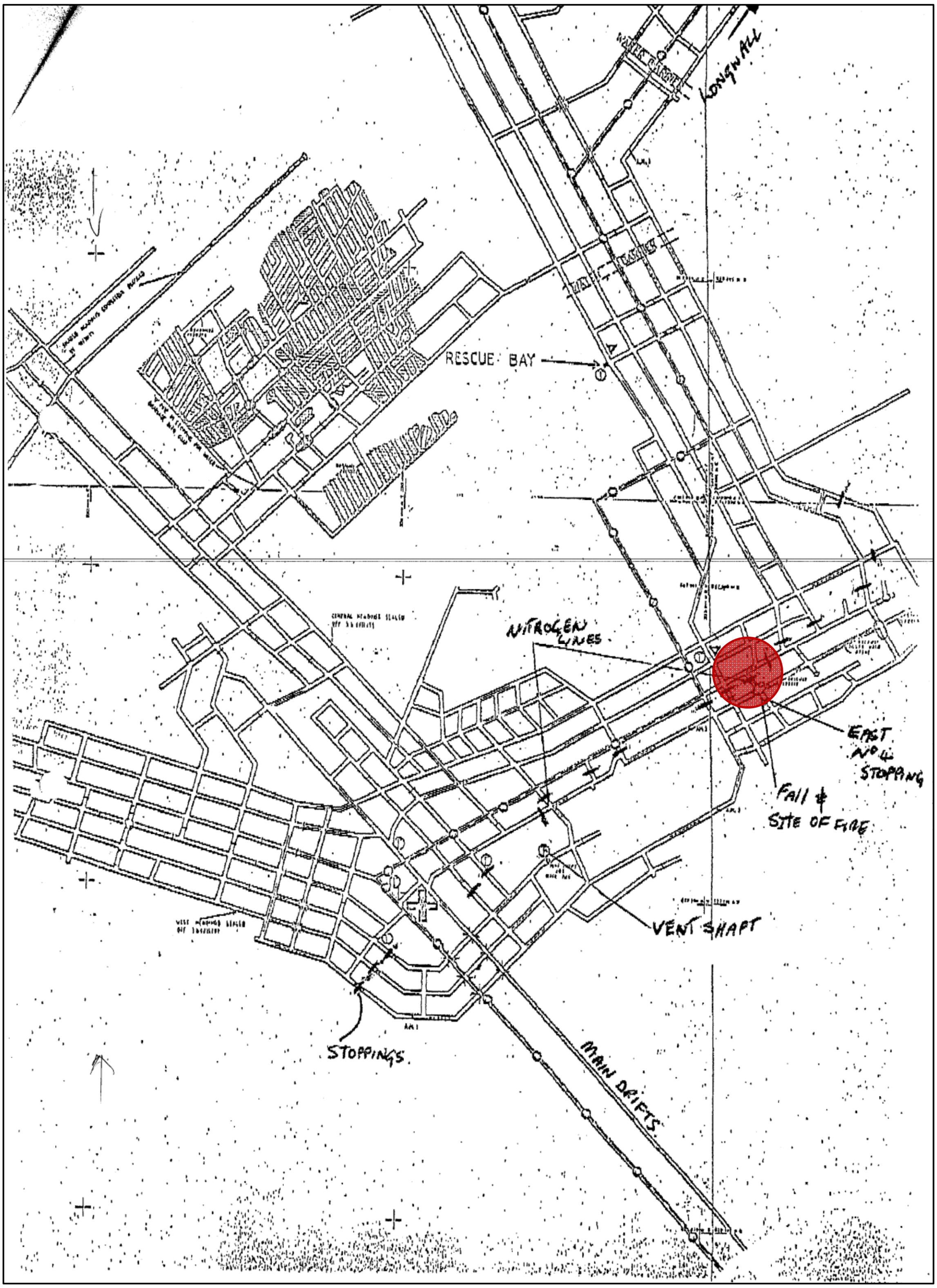
Initially coal was won on development only. Trials on total extraction and hydraulic mining failed. There was a history of spontaneous combustion in roadway sides and junctions. Longwall equipment was purchased in 1986 and the longwall commenced in September 1991. There were problems with face guttering and frequent spontaneous combustion issues in the longwall goaf.

Vaporised liquid Nitrogen was routinely injected into the Longwall goaf from pipes through trailing longwall supports at the rate of 100 – 100 m^3/hr. It was common for CO to exceed 1000ppm after sealing the longwall

Events were:

- Nov 11th 1991 longwall sealed due to heating
- Feb 13th 1992 longwall re-ventilated
- Mar 24th 1992 fall on face and heating in goaf

- Apr 18th 1992 longwall re-ventilated
- May 5th 1992 longwall sealed due to heating
- May14th 1992 longwall re-ventilated and re-sealed 5 hr. later
- June 30 1992 longwall re-ventilated then re-sealed 30 hr. Later
- June 29 1992 - longwall re-ventilated
- July 15 1992 - longwall sealed
- Sept 16 1992 - fire observed at No. 4 seal (see plan for location)
- Sept 18 1992 - fire out, minor smoke cleared, 48 hr. evacuation
- Sep 19 1992 - inspection determined all okay
- Sept 20 1992 - High CO in East Returns
- Sept 20 1992 - fighting fire with water and foam failed. Roof fall at No.4 seal. In seam sealing attempts failed. It was sealed at the surface with tarpaulins and Nitrogen pumped into the mine. The main fan was stopped.
- Sept 23 1992 4:45 pm - atmosphere explosive 9.5 % CH_4


LONGWALL
RESCUE BAY
NITROGEN LINES
EAST No 4 STOPPING
FALL & SITE OF FIRE
VENT SHAFT
STOPPINGS
MAIN DRIFTS

The following photograph shows the resulting damage to the mine transport portal.

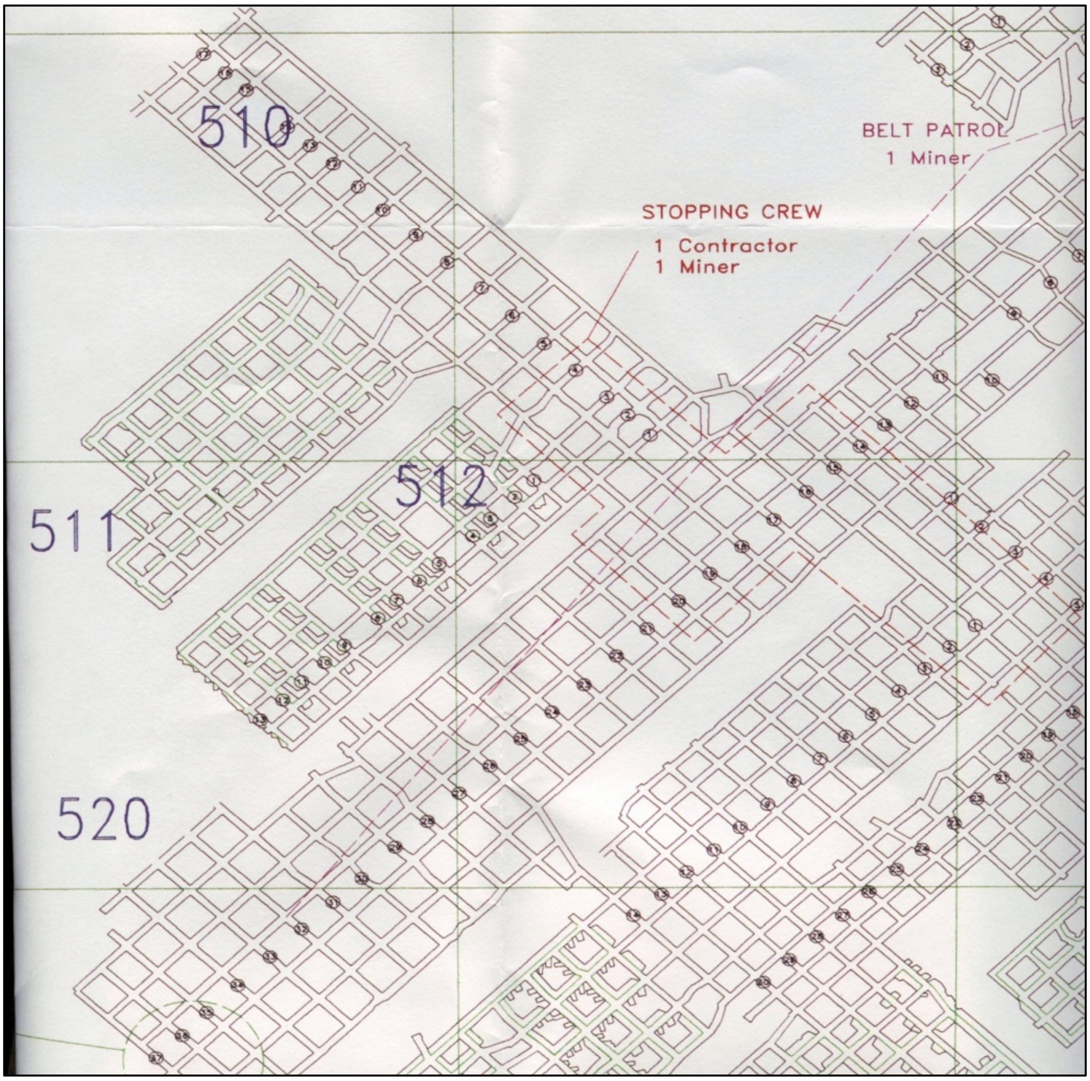
8.1.13 Moura No.2 – Aug 1994 – 11 Fatalities

The sealing of a bord and pillar extraction panel, 512 section, commenced on Saturday 6th August 1994. Sealing was completed at about 1am the following day.

Eleven (11) mineworkers were killed by an explosion that took place at 11.40pm. Ten (10) men were working in the 5 south production panel, which is located in another air split some three kilometres inbye and they self-escaped. It is believed the first explosion originated in 512 panel when a heating ignited flammable gas.

No.2 coal mine seam had inseam methane drainage. Seam gas content was $15m^3/t$ before drainage. The 5 South panel had been drained of methane and the panel was on development. The amount of methane being released out of the coal at the time of cutting was not high, less than one-half per cent in the section return.

The seam was 4.5m in thickness and the gradient 1 in 8. The top 3m was mined on development and bottoms mined on extraction. It was a partial extraction system.



8.1.14 North Goonyella – 1997

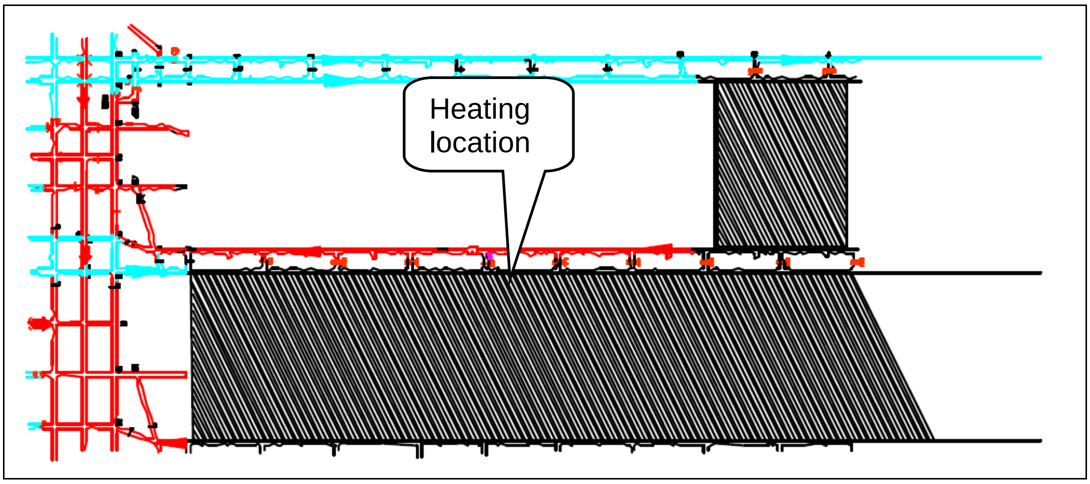
North Goonyella is a large longwall operation mining the Goonyella Middle Seam. Due to the seam thickness significant quantities of roof coal are left in the goaf. At the time of this incident, at the end of 1997, the mine operated two longwalls, numbers 3 and 4 South concurrently.

Three South longwall was only 9 metres from the take-off line whist four south face was just outbye 9 cut-through. An advanced heating was detected in the goaf of LW3.

On the afternoon of the 28/12/97 a deputy detected 25ppm of Carbon Monoxide (CO) in the general body of 6 c/t in Longwall 4 Tailgate. This reading was followed up with bag samples from the Longwall 3 goaf out of the 5 and 7 cut-through seals. The 6 c/t seal sample pipes were blocked with mud and water. The manager ordered the evacuation of the mine at 5.55pm on the 29/12/97 following the confirmation of the results of these bag samples. The bag sample results were as follows:

	H_2	CO_2	C_2H_6	O_2	CO	CH_4
7 c/t	0.4%	14.0%	0.08%	3.15%	0.13%	2.09%
5 c/t	0.43%	4.6%	0.05%	14.86%	0.12%	0.90%

This event is generally recognised as the most serious spontaneous combustion event to have occurred in Queensland since the Moura No. 4 explosion. Following as it did only a few months after the trials of the Tomlinson Boiler it was to be a crucial event in proving the ability of low flow inertisation techniques to treat serious goaf heatings. A heating which would have historically resulted in the loss of at least the section and possibly the mine's ability to produce for many weeks was controlled over a period of five days.



Plan of longwall 3 and 4 South Panels North Goonyella

8.1.15 Newlands - 1998

The Upper Newlands seam varies in thickness from 6 to 7 metres, with the lower 3m being mined. The predominant seam gas is methane in concentrations of above 95% Early tests on the Upper Newlands seam classified it as having a 'moderate to high propensity' for spontaneous combustion

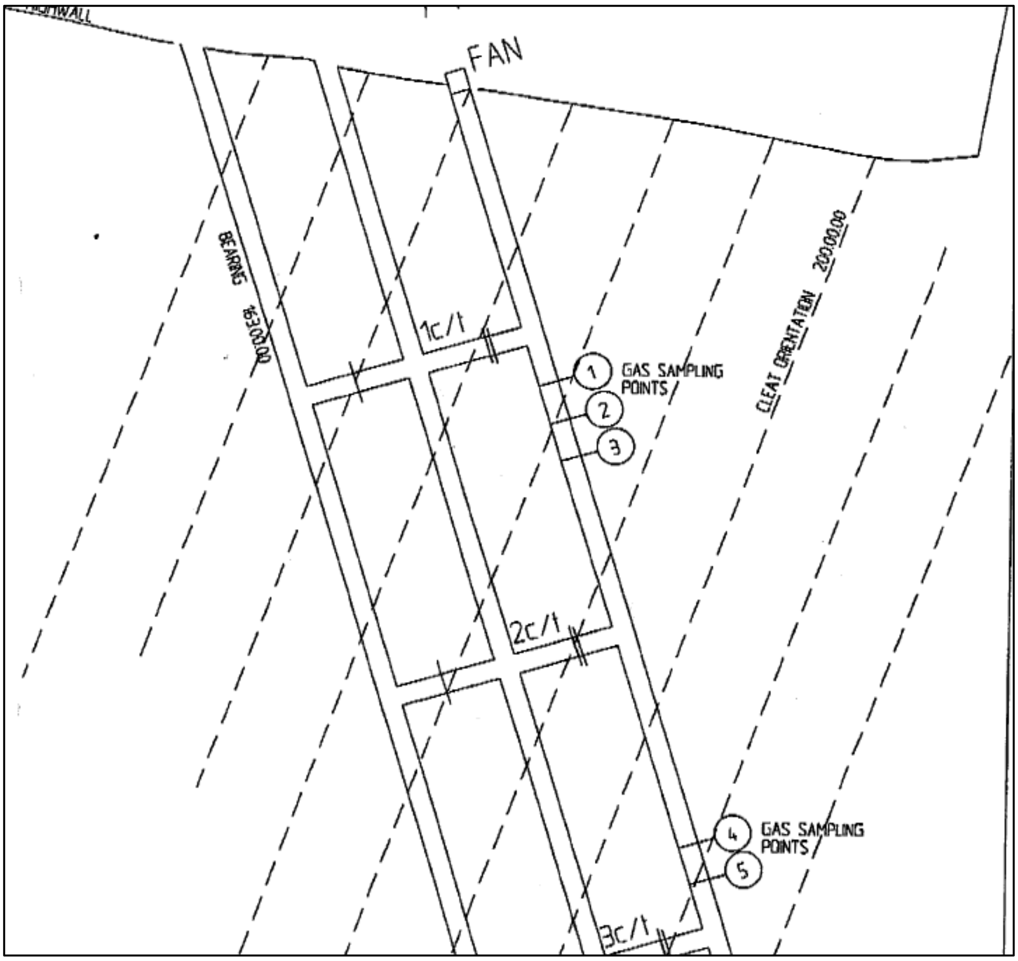
A number of small heatings took place near the portal entries in April and May of 1998 prior to longwall mining commencement. Newlands now classes the seam as having a high propensity.

Contributing factors to the heatings were considered to be:

- A sustained pressure differential of 400Pa across coal pillars that contained a large amount of open fractures
- Air migration through the fractures.
- High ventilation quantities concealing any products of the heating from monitoring devices located outbye.

The heatings were first contained by water injection and later by injection of both silicate resin and strata seal products. Injection varied in depth and direction in relation to the cleat direction for maximum results.

Temperature monitoring of roof and rib surfaces along with in seam measurements showed that a surface temperature of 30 to 35 degrees was an indicator of higher heat below the surface. Temperatures below the surface ranged up to 300 degrees. The 30 degree trigger was a valuable tool for the identification of further hot spots. Atmospheric monitoring, including minigas readings and bag samples was also used in the system for detection.



Plan of Newland entries

8.1.16 Blair Athol - 1999

Blair Athol Coal (BAC) is an open cut coal mine producing 11mtpa of export quality steaming coal from the 30m thick Number 3 Seam. The No.3 Seam is overlaid by an average of 40m of overburden. The 1-2m thick No.2 Seam, is spoiled with the overburden.

Underground coal mining commenced at Blair Athol in the 1890's. Four (4) underground mines have affected approx. 50% of the coal deposit. The No 3 Colliery closed in the 1950's. It had three levels of workings and a history of heatings.

The most significant intersection of the open-cut and underground workings was when strip 16 east intersected the Blair Athol No 3 Colliery in 1999. A significant fire started in exposed coal at the end of the strip. This fire was successfully smothered with overburden and it was assumed that any heating in the old workings could be treated this way.

However, as the dragline began to uncover the coal a number of openings on top of coal and in the new highwall began to emit smoke and steam. BAC were alerted to two major hazards. Firstly, concentrations of CO up to 1.2% (1200ppm) were present in the smoke venting from the workings. Secondly, there was a risk that an explosive mixture of distilled gases from the fires could be present in the workings. The area of the mine was evacuated until the composition of the atmosphere within the workings could be determined.

After some discussion, the DME recommended CO exposure limits of:

- Time Weighted Average (8 hours) 30ppm
- Short Term Exposure Limit (15 minutes) 200ppm
- Absolute Limit 400ppm

Exceedance of any these limits resulted in withdrawal of persons for the remainder of their shift. Research indicated that these limits were appropriate to ensure blood carboxyhaemoglobin levels were maintained at acceptable levels.

Monitoring boreholes were drilled into the workings from the surface, and a bundle tube system set up to sample gas from the holes. A typical analysis from the holes was 10% Carbon Monoxide, 12% Hydrogen, 4% Methane and less than 1% Oxygen; a very fuel rich, but inert, atmosphere. Such an atmosphere was somewhat outside that normally experienced in Australian mines.

Flooding the colliery with water was attempted to treat the fires. Water was pumped down 2 x 150mm boreholes into the workings for several weeks. The water had limited success, only marginally reducing the level of combustible gases.

By this stage, the uncovered strip of coal was burning strongly. The roof of a number of roadways had burnt through to the surface, exposing glowing hot ashes and a number were also burning with a blue flame. Water was diverted straight over the highwall onto the top of coal and mud and water were washed over the coal surface, smothering many of the more active fires. However, the atmosphere within the colliery remained rich with combustibles.

To control the explosion risk and continue mining operations it was decided to isolate the main section of the colliery from the current strip, inertise the atmosphere in the underground openings and smother the burning top of coal with a thin layer of overburden.

A GAG unit, Tomlinson boiler and Floxal units were used to flush and inertise the underground atmosphere.

To access the underground workings for use of the GAG, a 900mm hole was drilled into the workings. The GAG typically generated output of around 17.5m^3/s of inert gas at less than 0.1% Oxygen, and this has been adequate to generate an inert atmosphere within the workings in between 1 and 4 hours. In total 5 GAG campaigns have been run at BAC. QMRS recommend a minimum of 6 operators

8.1.17 Wallarah – Aug 2001

A heating took place in the Great Northern seam in August 2001. The seam was not considered to be a spontaneous combustion risk in underground mining, and this was the first recorded event despite the seam being mined for many years.

The exact site of the heating was not determined but considered to be in an area of old waste workings influenced by a ventilation connection between Wallarah and Moonie Collieries near Lake Macquarie NSW.

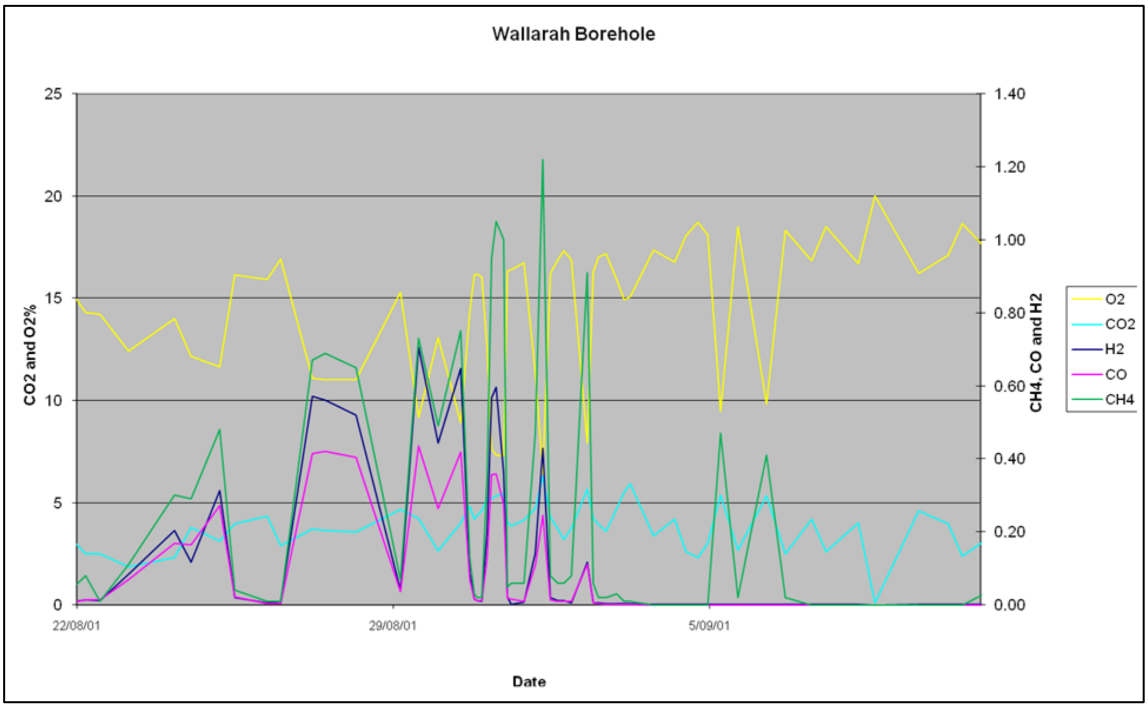
Increased readings of CO had been detected in Moonie Colliery dating back to March 2001. In the period leading up to the increased readings, two changes occurred that may have had an influence on the detection of gases and these were

- Improvements to ventilation at Moonie resulting in an increase in airflow through the connection from $4m^3/s$ to $20m^3/s$.
- Reduction in water flow into the Wallarah goaf from Moonie Colliery.

Checks on the Wallarah goaf showed no signs of heating.

The estimated size and shape of the void and the inability to successfully seal or isolate from the Wallarah seam (40m above) led to the decision to deviate from the normal process of nitrogen inertisation and instead, use the Mineshield unit to deliver carbon dioxide. Pumping of carbon dioxide ceased on 2 September. Since then, gas readings have been safe and stable.

The following diagram shows the influence of variations in the barometer on the atmosphere in the area.



Results of sampling of the atmosphere in the mine via a borehole

8.1.18 Beltana - Dec. 2002

Beltana mine is located in the Hunter Valley district of NSW. Mine entries are from the highwall of an open cut. The return highwall entry had an axial primary ventilation fan installation.

On 15th December 2002 physical indications (smell) and products of combustion from a heating (high CO) were detected in the first pillar between intake & return highwall entries of the Longwall 1 Panel Tailgate. The maximum differential pressure experienced during the life of the pillar was 250Pa, and pillar dimensions were 30m by 90m.

The heating had developed from airflow through open joints in the coal pillar and roof, and via blast induced fracturing from the highwall. These entries were also immediately adjacent to the endwall & therefore subjected to stress concentrations.

Over several weeks the heating was brought under control by sealing off the air paths using 6m drill holes and microfine cement grout injection. Also applied to the ribs and areas of roof in and adjacent to the fractured zones was a flexible surface sealant (cement in a latex binder).

Five meter long temperature probes & gas sampling holes were also installed during these remediation measures. Temperature monitors recorded peaks of 67^0C, whereas comparison of gas samples with gas evolution test results indicated the gas resulted from coal temperatures of 350^0C. Thermographic camera imaging was unable to detect any heat source or warm gas release.

Following Christmas 2002, the gas sampling indicated no detectable products of combustion other than small CO values. These fluctuated with barometric pressure and air temperature (night vs. day).

Temperature probes continued to show elevated temperatures, so in May 2003 seven 47mm diameter inseam boreholes were drilled at lengths between 20 & 40m into the pillar along different axis's in an attempt to locate the heat source. This was unsuccessful & water was injected for 1½ days to remove remnant heat, followed by microfine cement injection to seal any further potential leakage paths.

Further remnant fire gases could not be found & temperatures remained at normal background levels until the area was sealed & access lost in March 2005.

8.1.19 Beltana - Mar 2003

On 31st March 2003, off scale values of CO were detected using handheld gas instruments from open cleat cracks in two pillars each side of the overcast structures separating intake and return airways of Longwall 1 Maingate Panel.

These pillars were the third & fourth pillars inbye from the highwall entries, had dimensions of 17m by 30m, and were subjected to a pressure differential of 115Pa at the time of heating discovery (this was also the maximum pressure differential during the pillars life to-date).

Gas sampling indicated approximately 150oC heating temperatures. Thermographic camera imaging was capable of identifying hot/warm gas release from the open cleat.

A 20m long 47mm diameter hole was drilled in each pillar and water injected for 1½ days. These holes were then microfine cement injected. Leakage paths were identified through each pillar using smoke tubes and visual inspection and then targeted for drilling with short holes and grout injection.

The pillars continued to remain benign, and access was lost for inspection when the area was sealed in March 2005.

8.1.20 Southland – Dec 2003

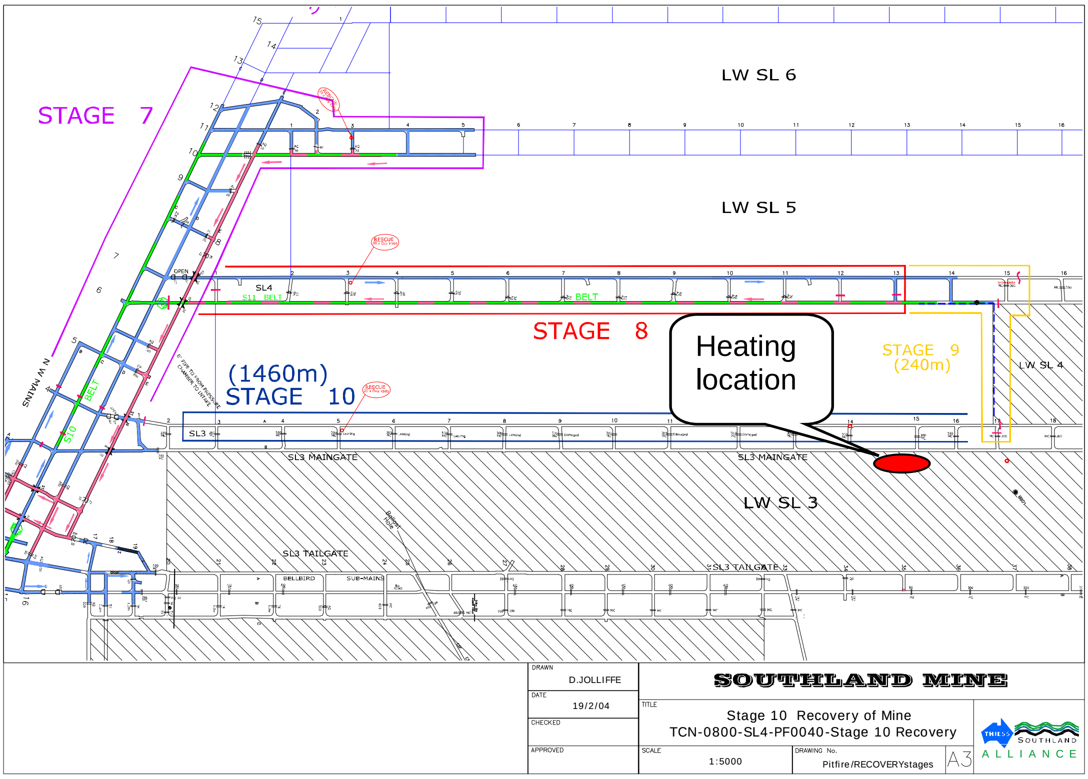
The heating took place in the longwall panel adjacent to the active panel. On 23rd December, a high CO alarm caused the mine to be evacuated. The goaf stopping adjacent to the longwall face crushed and air was entering the adjacent goaf.

On 24th December, black smoke issued from the upcast shaft. On 25th December, the colour of the smoke changed to light grey and it was believed the fire had broken out of the goaf into the longwall tailgate.

The GAG jet engine was used in an attempt to inertise the mine on 27th and 28th December. This was abandoned when the mine fan failed on 29th December. The mine was then sealed to extinguish the fire.

The bottom 3m of a 3 to 7m thick Greta seam was mined for reasons of coal quality.

A Tomlinson boiler was used to assist in the re-entry of the mine.


LW SL 6
STAGE 7
LW SL 5
SL4
STAGE 8
Heating
location
STAGE 9
(240m)
LW SL 4
(1460m)
STAGE 10
SL3
SL3 MAINGATE
LW SL 3
SL3 TAILGATE
N W MAINS
DRAWN
D.JOLLIFFE
DATE
19/2/04
CHECKED
APPROVED
SOUTHLAND MINE
TITLE
Stage 10 Recovery of Mine
TCN-0800-SL4-PF0040-Stage 10 Recovery
SCALE
1:5000
DRAWING No.
Pitfire/RECOVERYstages
A3
SOUTHLAND
ALLIANCE

8.1.21 Newstan - 2005

A heating took place in a sealed longwall goaf that was remote from current operations. The ventilation circuit was such that the main returns were adjacent to the seals of the current longwall goaves.

Normally, the goaf of each longwall block would become inert because of the liberation of seam gas. On this occasion, loss of inertisation was caused by interconnection from the goaf to surface cracks. The depth of cover was approx. 110m.

The location of the heating was derived from extensive surface drilling and associated monitoring and was adjacent to a fault system. At the time of mining with the longwall the fault had resulted in a large fall and a resultant cavity on the face.

The Mineshield was used to inject nitrogen into the goaf and stabilise the heating. However, oxygen from the surface was being continually drawn into the underground workings via the interconnection of the subsidence and goaf cracks by the negative pressure generated from the main mine fan.

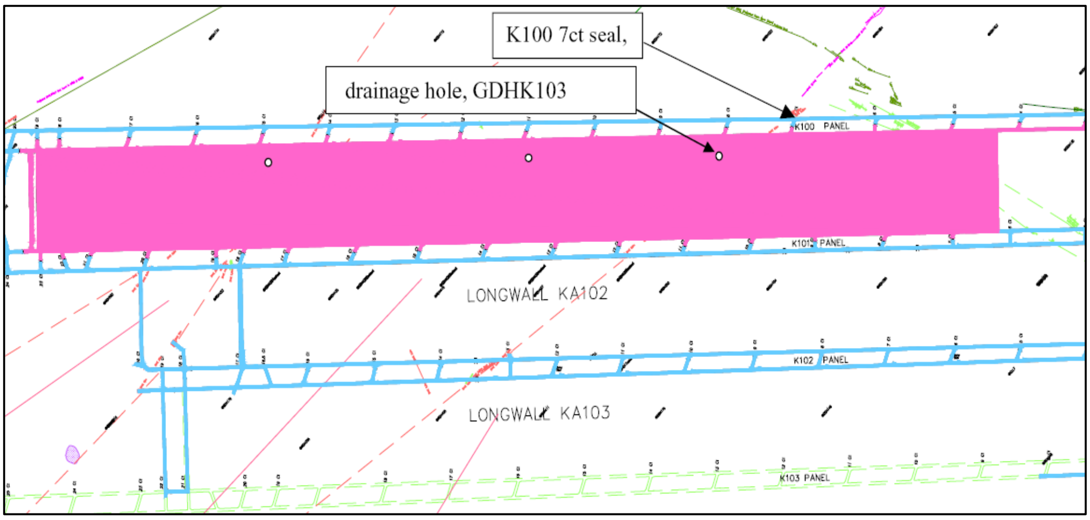
The long term solution was to inject fly ash to seal the cracks to the surface and to reverse the ventilation underground. This involved placing the longwall seals on intake ventilation. The result of these two actions was to reduce the pressure differential across the seals of the longwall that was allowing oxygen access to the heating. This allowed seam gases to build up and naturally inert the goaf.

8.1.22 Dartbrook 2005

Operations re-located from Wynn seam to Kayuga seam in 2004. The Kayuga seam is overlain by the Mt Arthur seam. The heating took place in the first longwall block mined in the Kayuga seam.

Both seams were considered to have a medium to high propensity for spontaneous combustion. Goaf drainage & Perimeter road established for gas management. Systematic goaf inertisation, thermal imaging of seals and tube bundle monitoring of seals used as precautionary measures against spontaneous combustion.

.
The Mineshield was used to inject Nitrogen into the area through a goaf drainage borehole and the mine fan was slowed to reduce longwall quantity from 80m^3/s to 60m^3/sec. When the heating was controlled, mining recommenced with two Floxal units replacing the Mineshield.



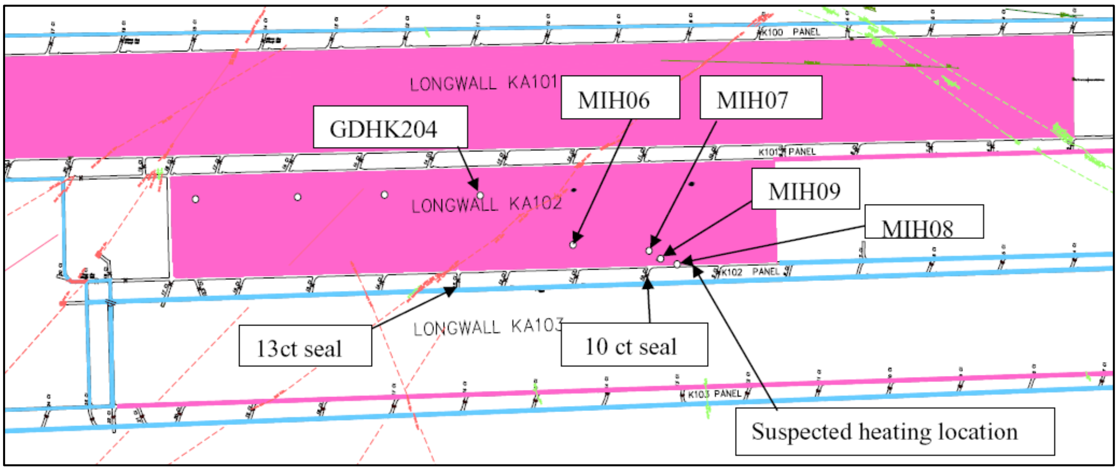
8.1.23 Dartbrook - 2006

The heating took place in the second active longwall block in the Kayuga seam. Bag samples from 13c/t and 10 c/t stoppings indicated unusual hydrogen levels. Subsequent samples indicated the presence of ethylene. The mine was evacuated on 19th January and tube bundle locations established into the goaf at 10, 11 and 13 c/t stoppings. Subsequent samples indicated ethylene

The mine was re-entered on 21st January with limited activities. The mine was again evacuated and further inertisation took place with a reduced mine fan speed.

The mine was re-entered on 18th February 2006 with limited activities taking place.

In the circumstances at the mine with high seam gas content, the ratio of Hydrogen to Carbon monoxide is considered a very useful indicator.



8.2 USEFUL FORMULAE

8.2.1 CO Make

CO Make is the volume of Carbon Monoxide flowing past a fixed point per unit time. This indicator removes the effect of dilution by general body air.

$$CO\ Make = K \times CO \times Q$$

Where: CO Make is measured in litres/minute.
Q is airflow measured in m3/second.
CO is the measured concentration of carbon monoxide in the air.
K is a factor as determined as follows:
If CO is measured in ppm then K = 0.06
If CO is measured in % then K = 600

As the equation requires air quantity, CO Make is only valid for roadways with airflow and cannot be used behind seals or closed boreholes. But because this indicator does make full allowance for changes in airflow to a heating it is suitable for monitoring the effects of oxygen deprivation on a heating.

8.2.2 Graham's Ratio

The equation is commonly expressed as:

$$GR = \frac{CO}{O_2\ Deficiency} = \frac{100 \times CO}{0.265 \times N_2 - O_2}$$

Where: GR is the Graham's Ratio calculated percentage depletion of oxygen in normal air.

CO is the measured percentage concentration of carbon monoxide.
N_2 is the measured percentage concentration of nitrogen.
O_2 is the measured percentage concentration of oxygen.

The CO/O2 deficiency ratio may underestimate the state of progression of a heating, but combined with other monitoring and analysis methods it provides an extremely useful indication of the state of a heating. The main application of Graham's Ratio is in the detection of heatings or fires that may otherwise be disguised by changes in ventilation and for monitoring their progress. The trend of the readings is more important than absolute values, with an increasing trend indicating increasing temperature within the fire.

8.2.3 Young's Ratio

Young's Ratio is the same as Graham's Ratio except that CO is replaced by CO2 as the indicator of oxidation of the coal. Because of the size of the CO2 concentration it is not usually multiplied by 100 and thus is a fraction not a percentage as is Graham's Ratio:

$$YR = \frac{CO_2}{O_2\ Deficiency} = \frac{CO_2}{0.265 \times N_2 - O_2}$$

There are no universally acceptable trigger levels because carbon dioxide generation as a function of temperature is very coal dependent. The ratio trend is more important than absolute ratio values.

The limitations of this ratio include other sources of CO_2 from seam gas or vehicle exhaust, the potential loss of CO_2 as it readily dissolves in water, and the same problems with oxygen deficiency as Graham's Ratio. Decaying timber can be a source of CO_2 than could unbalance the use of this ratio.

8.2.4 CO/CO_2 Ratio

This ratio is independent of oxygen deficiency and so overcomes a lot of the problems associated with other ratios that are dependent of that deficiency. It is based on the change in ratio of carbon monoxide produced to carbon dioxide produced as a function of the coal temperature during the initial development of a heating. It therefore defines typical coal temperature values. Obviously this index can be used only where no carbon dioxide occurs naturally in the strata.
The index increases rapidly during the early stages of a heating, but the rate of increase slows at high temperatures. However, the rate of change at higher temperatures is sufficient to provide a very useful indicator of the progress of a well-established fire.

8.2.5 Morris Ratio

This ratio is essentially the inverse of Graham's and Young's Ratios. It is a measure of the amount of oxygen absorbed/destroyed (as determined by the excess of nitrogen over that required to balance the amount of oxygen present) by the coal to the amount of oxidation produced by the coal. Where the inlet is fresh air the ratio is expressed as:

$$MR = \frac{N_2\ Excess}{CO + CO_2} = \frac{N_2 - 3.774 \times O_2}{CO + CO_2}$$

An unusual feature of this ratio is that the ratio increases to a maximum at approximately 120^0C then decreases, and the size of the peak is very coal dependent. Because of this peaked behaviour, it cannot be used alone to indicate temperature of a coal heating as one cannot estimate on which side of the maximum a data point lies. Therefore valid in early stages of heating when increasing trend indicates increasing heating activity.

8.2.6 Jones-Trickett Ratio

Not suitable for sealed areas. This ratio is based on the measurement of the amount of oxygen required to be consumed to produce the oxidation products observed compared to the amount of oxygen actually removed from the inlet gas stream. Increasing ratio indicates intensifying heating / temp increase.

$$JTR = \frac{CO_2 + 0.75 \times CO - 0.25 \times H_2}{O_2\ Deficiency} = \frac{CO_2 + 0.75 \times CO - 0.25 \times H_2}{0.265 \times N_2 - O_2}$$

Research has shown that the type of fire or heating that has occurred can be determined from the product gas mix using the Jones-Trickett Ratio. Literature based indicator levels for the ratio are:

- < 0.4 Normal
- < 0.5 Methane Fire possible
- < 1.0 Coal Fire possible
- > 1.6 Impossible

Note that the Jones-Trickett Ratio is invalid if the intake air is oxygen deficient through the injection of nitrogen or carbon dioxide or through a high methane make. In addition the dilution with fresh air of the combustion products has no effect on the ratio.

8.2.7 Litton Ratio

$$LR = \frac{1}{3} CO_s \, (\%R_g)^{-1.5} \, (\%O_2)^{-0.5}$$

Where:

COs - carbon monoxide concentration in ppm

O2 - oxygen concentration in percent

%Rg - percentage residual gas - originally specified as = 100 - 4.774 O_2 - CH_4

but more generally = 100 - 4.774 O_2 - seam gases

This ratio is a measure of the oxidation efficiency and has mainly been applied in evaluating the level of activity of fires, with low temperature oxidation having a low conversion efficiency of oxygen to carbon monoxide. The situation is unsafe if the ratio is greater than 1, and can only be defined as safe if the ratio is less than 1 and stabilised. Decreasing values for the ratio even if less than 1 indicates that equilibrium (i.e. normal temperature oxidation) has not been reached.

This ratio is able to detect actual combustion, but is not sensitive enough to identify the preliminary phase of heating.

8.2.8 Willett Ratio

$$Willett\ Ratio = \frac{CO_2\ produced}{Blackdamp + Combustibles}\%$$

Blackdamp is a term generally applied to carbon dioxide, but also includes nitrogen. Combustibles include all combustible gases present (methane, carbon monoxide, hydrogen and any higher hydrocarbons).

Level of activity is indicated by the value obtained, with a falling trend indicating decreasing activity. Stable values may indicate no activity. This ratio has been found to be more effective than Graham's Ratio in determining the state of spontaneous combustion activity behind sealed areas.

8.2.9 H_2/CO Ratio

This ratio indicates temperature of a heating. An increasing ratio indicates intensifying heating or temperature increase. The ratio is independent of dilution with fresh air or seam gas or oxygen deficiency.

Limitations of this ratio include: CO depleted by bacteria, vehicle emissions, ratio rate of change slowed in sealed areas resulting in 'averaged' values, inaccurate for low H2 values due to analysis limitations.

8.2.10 Air Free Analysis

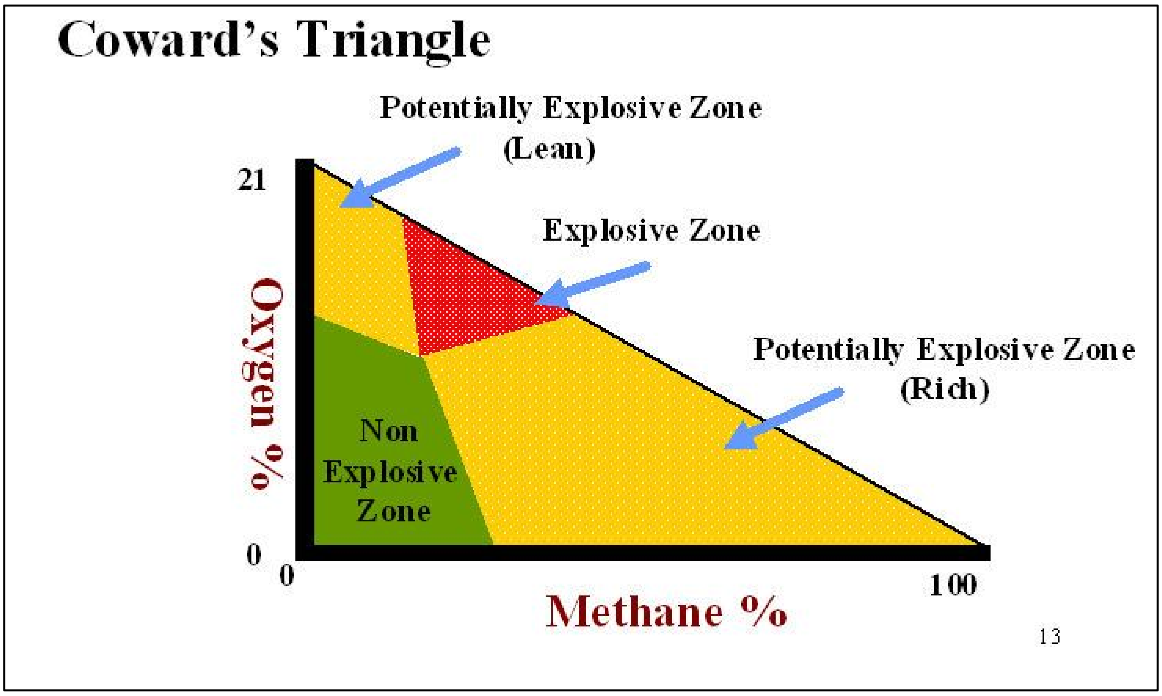
Air free calculation:

$$\frac{100\ \text{gas component\%}}{100 - 4.778 \times O_2\ \text{(as analysed)}}$$

Seam gas free calculation:

$$\frac{100\ \text{gas component\%}}{100 - \%\ \text{total seam gas (as analysed)}}$$

8.2.11 Coward Triangle



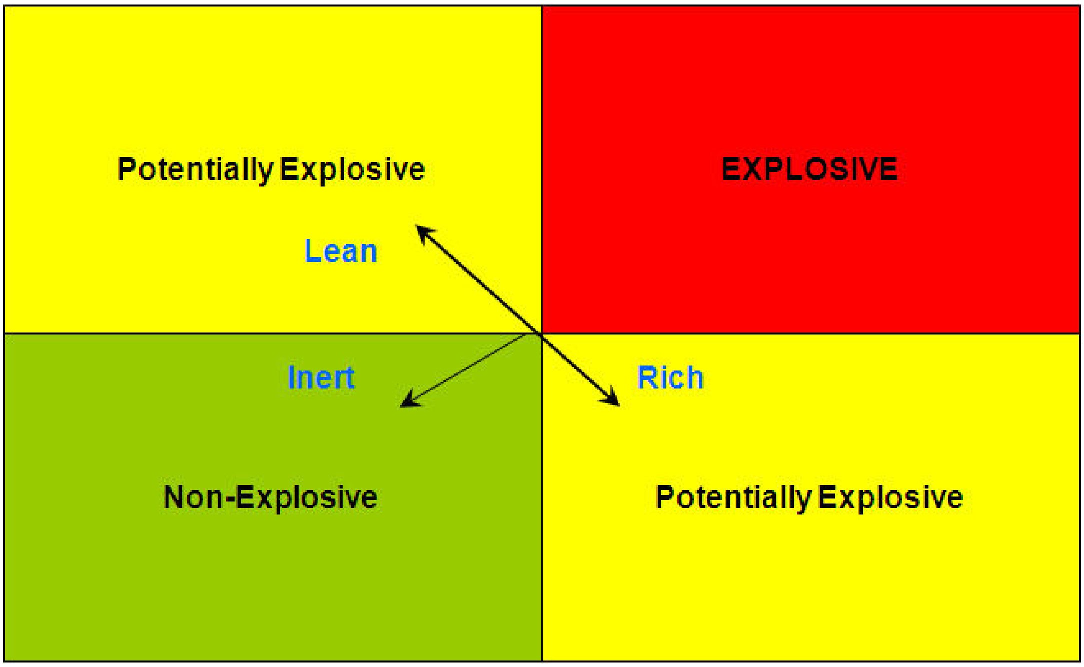
The Coward Triangle plots the percentage oxygen against the total percentage of methane gas in the gas sample. In addition, the barriers between the explosive, potentially explosive and non-explosive gas concentration zones are defined. The position of these barriers is calculated from the combination of the upper and lower explosive limits of the flammable gases present weighted by their concentration. The position of the datum point then indicates the potential for explosion. In addition the expected behaviour of the gas mixture under various scenarios can be predicted.

- Adding fresh air makes the datum point move toward the top left corner of the triangle.
- Adding inert gases makes the datum point move toward the bottom left corner.
- Adding more combustible gases makes the datum point move toward the bottom right corner.

The triangle limits are fresh air, inert gas and 100% flammable gas. The calculations are complex and are usually conducted using computer software.

Due to the changing size of the explosive zone with different explosive gas concentrations, it is not possible to use the Coward Triangle for trending a sample point over time.

8.2.12 Ellicott Diagram



The Ellicott Diagram is a modification of the Coward Triangle that allows trend analysis. The triangle is changed into a rectangle, with the centre of the diagram being the nose point and the axes radiating from there being defined by the upper explosive limit barrier (+X axis), the lower explosive limit barrier (+Y axis), the line from the fresh air limit on the Y axis to the nose point (- X axis), and the continuation of this line to intersect the Y axis of the Coward Triangle (-Y axis).

- Adding fresh air makes the datum point move toward the left end of the horizontal axis.
- Adding inert gases makes the datum point move toward the bottom left corner.
- Adding more combustible gases makes datum point move toward the bottom right corner.

One major advantage that the Ellicott Diagrams has over the Coward Diagram is the ability to plot a number of samples on the same graph and establish trends over time.

Care needs to be taken in comparing Ellicott Diagrams, as some of the information available in Coward Triangles is lost. In particular the size of the various sectors on the Coward Triangle may vary between analyses as the mixture varies, yet the Ellicott Diagram always allocates each segment the same size. The information conveyed through the relative sizes of the zones is lost as they are set to a fixed size on an Ellicott Diagram, with the non-explosive zone forming twice the size of the other zones.

8.3 REFERENCE MATERIAL

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- Department of Mineral resources, NSW, Spontaneous Combustion Seminar, Mudgee, 6th to 8th November 1995.

Feedback sheet

Your comments on **MDG1006 Spontaneous Combustion Management Guideline** and **MDG 1006 Technical Reference** will be very helpful in reviewing and improving these documents.

Please copy and complete the feedback sheet and return it to:

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Mine Safety Operations
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516 High St, Maitland NSW 2320
Phone: (02) 4931 6658 Fax: (02) 4931 6790
Email: david.nichols@industry.nsw.gov.au

How did you use, or intend to use, these guidelines?

What do you find most useful about these guidelines?

What do you find least useful?

Do you have any suggested changes to these guidelines?

Thank you for completing and returning this feedback sheet

Coal Operators' Conference

Faculty of Engineering

1998

Dartbrook Mine - a case study

J. Hayward
Dartbrook Mine

Publication Details

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Dartbrook Mine - A Case Study

J Hayward[1]

ABSTRACT

Any project carries a number of challenges and risks. Inappropriate design, poor project management, time, industrial disputation and capital over runs are all areas which can impact on a project cash flows and return on investment. In the resource Industry you have the added complexity and risk of geology which is difficult to predict and sometimes unforgiving. The Dartbrook Mine was designed and constructed as a high output longwall mine and has overcome a number of hurdles to produce over 2.3 million tonnes for the first year of operation. A number of problems were encountered during construction which resulted in the inseam development being delayed for six months. Dartbrook people have demonstrated they can manage adversity and produce world class results. The mine has been through a very steep learning curve during 1997 and with this experience behind the mine is now producing at an annualised rate of 3.4 MTPA.

INTRODUCTION

Dartbrook Mine was designed and constructed as a new underground longwall coal mine with a planned capacity of around 3.5 million tonnes per year. The mine is located in the upper Hunter Valley 10 kilometres north of Muswellbrook NSW. Construction of the mine commenced in June 1993 and inseam development commenced in October 1994 from the bottom of the 1200 metres 1 in 8 grade men and materials drift. The longwall was installed in September 1996 and when completed in mid October 1997 2.24 million tonnes had been mined by the longwall. Results from the first longwall block were exceptional considering the challenges of steep grade, new technology and managing abnormal quantities of gas.

MINE CONSTRUCTION - LESSONS LEARNT

The Shell Board approved the project in April 1993 and when construction commenced in June 1993 the project was already six months behind schedule. Project Managers were appointed to manage the construction of the mine. The contract was written for the Engineering, Procurement, Construction and Management (EPCM) of the project. The Dartbrook management team were employed before the project commenced which provided the opportunity for input into the mine design, contracts and equipment selection. Having a Project Manager also provided the opportunity for the management team to develop operating parameters for the mine, design systems, negotiate labour agreements and recruit people. However, Project Managers are expensive at 5% of the total Capital value of the Project. The Project Managers at Dartbrook were good at building offices, workshops and coal handling facilities but did not have the experience on underground construction work. We spent a lot of time arguing with our Project Managers and also employed contractors to " keep the Contractors honest". The end result was well engineered and constructed infrastructure with an good safety, Industrial and environmental record, but the project exceeded capital estimates by around 8% and underground development commenced 6 months behind schedule.

The experience gained indicated that an in house project team could reliably manage the construction for a new mine. There is sufficient expertise available in terms of experienced mining and engineering people now in consulting roles to assemble a well balanced team to design and construct a mine.

Mine Manager Dartbrook Mine

MINE DESIGN

The concept of a thick coal seams and the opportunity to develop 4 metre roadways sounds exciting and has a number of obvious advantages but also presents a number of challenges . Larger roadways assist in keeping ventilation pressures low and permit larger equipment to be transported and installed underground but presents difficulties in the installation of roadway supports and hanging the services at roof height .

Equipment to mine at 4.0 to 4.5 metres was not readily available "off the shelf" and Dartbrook has had to design and install some very innovative development, longwall and support equipment to meet the challenges of mining thick seams.

With a coal seam that has never been worked before we realised that we needed a comprehensive data base of information to be able to plan with high confidence levels. The mine plan was designed for the best recovery of the resource and orientation of mains and longwall blocks with due consideration given to geological features, grades and the management of gas, water and spontaneous combustion. The exploration program and mine design have been shown to be the best for Dartbrook with no major shocks after 3 years of operation. The coal seam is as good, or better than anticipated with only minor faulting and dykes. We always knew that gas and spontaneous combustion were issues that were unique in a 22 to 24 metres coal section and that we would have to come up with some unique solutions.

PEOPLE - RECRUITMENT, TRAINING AND EBA'S

The quality of people at Dartbrook has demonstrated the importance of the recruitment process. The process was time consuming, costly and involved a significant number of people .

Dartbrook personnel completed a seven week training program before entering the workplace. The value of this is bearing fruit, not only in terms of knowledge and skills but in the development of a unique safety and team oriented culture. This has led to a desire for training and qualifications, which has to be managed. The mine has been in operation for over three years and the energy and enthusiasm is still evident in the entire workforce.

During 1998 Dartbrook Mine will negotiate the third Enterprise Agreement with its workforce. The first EA was negotiated with the District Officials of the CFMEU and the majority of employees were recruited under this agreement. At the time the agreement contained some very innovative and ground breaking changes which have carried through to the Second EA which was settled for a three year term. The basis of the first EA was fixed salaries, all inclusive of allowances, overtime and production bonuses. In return employees were required to work rotating shifts of 8.5 hours duration five days per week and commit to work allocated overtime. The only changes to the second EA is a step change to level 4 and 6 of the Industry work model, greater use of contractors and the overtime component reduced to allow a separate payment for weekend overtime when worked. Of interest, the second EA included the option for all employees to salary sacrifice for a fully maintained company provided car. Approximately 60% of the workforce have availed themselves of this opportunity. It is worthy of note that the concept of production bonus payments has lost its appeal and Dartbrook people are pressing good performances because of personal pride in doing a good job and in their mine. Along with the rest of the Coal Industry we are paying our people higher salaries than other Industries however I believe time and technology will develop realistic labour cost. Industrial relations at Dartbrook remain to be very strong and robust.

Although Dartbrook has maintained excellent management / labour relationships over the three years of operation, I believe that it could be enhanced by talking direct with our people without District Union intervention on critical issues. We have built up a strong relationship with our people based on two way trust which could explore mutually agreed new opportunities outside the traditional union structure.

SAFETY

Another greenfields opportunity is the setting of high standards of safety which can be an integral part of the induction training program and be written into the mines operating procedures. From the design and construction stage, Dartbrook set very high safety standards with hazops, risk assessments, procedures and Risk Management Plans (RMP's) being utilised as tools for the mine construction and operation. Only Contractors with good safety performances were considered for construction work on the site. Contractors have to go through a rigerous pre-qualification process to work at Dartbrook and Contractors with poor safety history are not considered for the tendering process.

External and internal safety audits are an ongoing activity at Dartbrook and maintain the safety focus. Extensive induction and training programs have assisted in setting a unique safety culture at Dartbrook.

Safety performance has been excellent for the first 2 years with LTIFR's less than 10. However 1997 was disastrous in terms of safety performance with 12 LTI's and one fatality. The fatal accident to a young contractor in January 1997 was a very sobering reminder that too much time cannot be spent on safety. The accident was a major shock to all employees who did not believe that such a tragedy could occur in a mine with such high standards and a strong safety culture.

Initially contractor safety performance during the first year of construction was very poor with an LTIFR of around 25. We have now formed long term partnerships with contracting organisations who have accepted the mines high safety standards. The two major Contractors have retained a regular experienced workforce and employed fulltime safety and training persons. The Unions initially did not like to see regular contractor organisations on site because of the fear of Contractors taking potential jobs away from permanent workers. After the accident everyone realised that you cannot have new and unknown contractors on site and good safety is having well trained and experienced people who you can trust to maintain the standards.

RESOURCE

After an extensive exploration program, the coal seam is almost as predicted with excellent mining conditions, low water make and some localised steep grades. On the downside we have experienced higher than predicted gas levels.

The Wynn seam is Permian in age and is a bituminous coal of thermal quality. It is low in rank and has little to no swelling characteristics necessary for coking coal. Coal quality is generally as predicted with low sulphur and low ash but has high levels of Calcium Oxide (CaO) in the ash content. The higher than expected CaO provided the impetus to review the need to bring forward the construction of the coal preparation plant from year 10 of the project. The recently commissioned coal preparation plant enables Dartbrook to compete in the premium Pacific Rim markets and also to recover an additional 500mm to 700mm of the coal seam which partially offsets the washery capital cost and helps the economics of the project to look more respectable.

Exploration

At Dartbrook early drilling by the Department of Mineral Resources (DMR) and other organisations identified the presence of abundant coal reserves. Further drilling detailed the potential open-cut reserves. Prior to the commitment to longwall mining at Dartbrook, some 4 bores intersected the Wynn Seam every square kilometre. The Wynn Seam does not outcrop anywhere, nor has it been mined previously. All information about the seam came from boreholes or remote sensing.

With Shell's decision to commit to mining in April 1993 it was decided that we needed more information on an area to cover the first four longwall blocks. A further exploration program involving 24 extra boreholes was undertaken during 1993/94 to improve the knowledge of the area. Drilling across the lease and exploration areas still continues in an effort to better identify the risks associated with mining. Currently, the mine has a coverage of 18 bores per square kilometre, composed of 94 open holes and 86 cored holes.

Stress

Initial stress determinations at Dartbrook in sedimentary strata above the coal seams realised a high degree of variation of horizontal stress. The Bayswater seam, which is the immediate roof of the Wynn seam was targetted for follow up stress work. This work highlighted the relatively benign stress environment enveloping the target Wynn seam. Ultimately, this meant that the effect of stress at Dartbrook was not a critical factor to be taken into account with the mine layout.

Stress measurements were undertaken both from exploration bores and underground. In general there is an active east west, maximum principal stress in the horizontal direction, averaging 110 degrees. However, in the environment of the Wynn seam envelop between overlying and underlying coal, the effect of the horizontal stress is minimised to such an extent that the vertical stress is predominant. The combined effects of the vertical and horizontal stresses have resulted in only minor guttering and rib crush occurring in gateroads. It appears that the vertical stress is nearly perpendicular to the seam itself which is dipping to the west.

Structural geology

To assist in determining the layout of a mine, the accurate identification of geological structures is critical. Depending on thickness, hardness and consistency. Igneous intrusions can have a devastating impact on mining as has been demonstrated at a number of longwall mines with dramatic consequences. Identification of these structures during the exploration stage, allows the opportunity to plan for them, rather than deal with them as an emergency situation.

Techniques such as surface and aeromagnetic surveys, have a proven track record of accurately locating structures. At Dartbrook several surface and aeromagnetic surveys found intrusive anomalies were orientated in a north-east direction. Further investigations involving costeaning, helped to identify these structures. Intersecting dykes near parallel to a longwall face could cause difficulties with extraction. Intrusions oriented at an oblique angle to the faceline will minimise longwall delays.

Dartbrook Mine has a nominal borehole spacing of 250m. Uncertainty surrounding possible faulting (at the time) resulted in limited input into the mine design. Clearly, faulting could cause difficulties with mining. However, the extensive coal overlying and underlying the mining horizon would minimise these difficulties. The issue becomes one of coal quality rather than structural constraint associated with mines with stone roof and floor.

Evaluation and examination of borecores identified the presence of extensive jointing at Dartbrook. Determination of orientation became a priority during exploration. The RaaX photography method accurately identified the orientation of the joints. The jointing is ubiquitous and trends relatively consistently at 110 degrees. Underground measurements in the first workings confirmed the exploration data. The orientation of the longwall blocks took the jointing into consideration. With the thick seam and the jointing in mind, design of the longwall supports incorporated face spags (flippers). The flippers support the top of the seam to reduce spalling. and protect the operators from injury.

Coal quality

The mainroad development is in a westerly direction along the length of the southern lease boundary. These roadways experience relatively higher ash levels and lower seam height to facilitate the longwall extracting the premium quality coal.

Roof and floor geology

In the currect longwall mining area at Dartbrook, the typical roof comprises of about 14m of coal. While loading from the overlying sediments is unlikely, a 'risk averse' policy has selected longwall shields rated at 913 Tonne yield. The roof support density is 118 tonnes per square metre.

The mining floor comprises of 300mm of tuff which is quite competent with a compressive strength of between 14 and 28 Mpa. The Mine Technik Australia (MTA) longwall face supports at Dartbrook have a base lifting capacity to assist in keeping the face on the appropriate mining horizon. The Coal is very strong and both longwall one and two advanced around 15 to 20 metres past the installation roadway before the coal roof started to fail. No probems were experienced with wind blasts of gas inrushes on both longwall startups, although precautions were taken to minimise and manage these risks.

Gas

All coal seams in the Dartbrook mining lease contains a seam gas mixture of Methane (CH4) and Carbon Dioxide (CO2). In-situ gas contents range from 6.5 to 11 cubic metres per tonne in the Wynn seam. Gas contents for the upper seams are generally less. Carbon dioxide is the predominate gas with CO2/CH4 ratios ranging between 90:10 and 60:40.

Prior to the commitment to mine at Dartbrook, the gas data obtained from boreholes indicated the necessity for gas drainage. A variation between the surface exploration standard and the underground 'quick crush technique gave a discrepancy of between 1 to 2m^3/tonne. Absorption of CO_2 into the acidified brine caused this error. To make a more satisfactory comparison between the exploration and the underground data, the mine employs the quick crush technique in current exploration work.

Reservoir permeability, diffusivity, porosity and sorption isotherms have been determined by laboratory testing. In situ gas pressure has been directly measured. This work indicated that the coal has a fairly low permeability for its depth and is under-saturated with gas.

Gas reservoir modelling has been undertaken to determine gas emission rates upon development and longwall extraction of the Wynn seam. Development gas emission was modelled using the SIMED simulator from the Commonwealth Scientific and Industrial Research Organisation (CSIRO) which uses a two phase 3D multi-component simulator. The indicated rib emission rates range from 20 L/s to 50 L/S per 100 metres and would be markedly influenced by seam permeability and gas content. Based on predictions for a 4 kilometre roadway, two heading gateroad with 8.6 cubic metres per tonne disorpable gas of 80% CO2 and a permeability of 0.44 mD, a ventilation requirement of 55 cubic metres per second is required.

Longwall gas emission has been predicted using various empirical techniques. The modelling is limited due to most techniques being particular to methane gas. However, gas reservoir estimates indicate between 40 and 55 cubic metres of gas per tonne of coal to be mined, is contained in the immediate floor, working section and roof of the Wynn seam. It was estimated that without gas drainage on longwall one a ventilation requirement of 232 cubic metres per second across the longwall face would have been required to meet the statutory limit of 1.25% CO2.

The actual seam gas content was 1 to 2 cubic metres per tonne higher than predicted from the borehole data, however the experience to date has demonstrated that the permeability is higher than expected and the gas drains relatively easy. In fact a little too easy as the cumulative rib emissions for a four kilometre gate road (in 2 klm and out 2 klms) culminated in general body readings at the outbye end of the returns of around 1 to 1.20% of CO_2.

Two weeks of development was lost due to gas levels running around the legal limit of .25 % CO_2 Extensive rib drilling was carried out to reduce the rib emissions.

Initial block drainage was trialed using directional drilling to drill 450 metre holes across the block parallel with the gateroads. When gateroads were advanced 450 metres cross holes were drilled across the block at initial spacing of 20 metres which reduced to 10 metres approaching the face installation roadway.

The cross holes were drilled in the seam section , downholes into the 4 metre section below the 300mm tuff floor band and up holes into the 12 to 14 metre coal roof section. Roof holes were determined to be of poor value with only 30 % capture after 6 months due to impermeable carbonaceous bands. Holes were branched off the up holes perpendicular to the stratification which only marginally improved the capture rate. All holes drilled were by directional drilling using a number of drill rigs including LM35, LM55, Boyles and Diamec 262 and Diamec 252. As at November 1997 approximately 300 kilometres of gas drainage holes have been drilled.

With the pre-drainage of the block modelling demonstrated that production would be limited to around 50,000 to 55,000 tonnes per week of production. The Department of Mineral Resources provided special dispensation to allow a special fenced off toxic return up to 3% carbon dioxide. This was later raised to 3.5% CO_2 as the mine production had plateaued at around 60,000 tonnes per week with an average of 20 hours per week of lost production due to high CO_2. As soon as the exemption was given to mine up to 3.5% CO2 the longwall production levels increased to around 75,000 tonnes per week.

The ventilation system adopted for Longwall one was three intakes and one toxic return with two intakes on the maingate side and one intake up the blockside tailgate and a back return system. To ensure the back return remained open, initially two rows of 6 metre flexi bolts were installed. A number of 900mm auger holes were drilled through the tailgate chain pillars with minimal impact. The last 500 metres of the tailgate of Longwall 1 were supported with 900mm aerated concrete cans to maintain the back return. The cans were easily and safely set and maintained an excellent back return.

The longwall goaf was producing CO_2 and methane at the rate of 5 cubic metres per second and it was decided from the early modelling that post drainage of gas would be required if Dartbrook was ever to be able to meet the statutory limit of 1.25% CO2. Vertical goaf wells were drilled 70 metres from the installation roadway and approximately 70 metres from the maingate side of the block and connected to a high pressure exhaust fan. Research found there was no experience with goaf wells extracting carbon dioxide. Goaf holes have worked well at a number of mines in Australia and the USA extracting methane but greater suction was required to overcome the buoyancy effect of CO_2. The holes eventually proved to be very successful particularly on the tailgate side of the block at 200 metre intervals exhausting at the rate of 1 to 1.5 cubic metres per second of CO_2 and methane per hole

With two pumps operating, up to 2 cubic metres per second of gas was exhausted which dropped the gas levels in the toxic return by around 0.3% CO2. Longwall 2 has adopted a homotropal ventilation system which is a mirror image of Longwall 1. The Department of Mineral Resources (DMR) has given approval for longwall 2 based on additional post drainage facilities being adopted to reduce the 3% CO2 limit in the toxic return to 1.25% by the completion of Longwall 2. As well as post drainage from vertical holes post drainage from seals behind the face line have proved successful. The Longwall 1 face was ventilated with 95 to 100 cubic metres per second which is 3 to 4 times the ventilation quantity on the average longwall face. Longwall 2 is currently ventilated with around 70 cubic metres per second

The four kilometre Hunter Tunnel construction experienced high methane emissions and water ingress associated with a synclinal structure. With the exception of two major dykes (2m and 9m) mining conditions in the tunnel were generally good. Water flows peaked at around 60 litres per second and produced uncomfortable physical working conditions. Water flows are now down to an easily manageable 12 to 14 litres per second. The gas content in seam was as predicted at around 4 to 5 cubic metres per tonne, however the fractured ground associated with the synclinal structure provided a conduit for the high gas emissions. The Hunter Tunnel was holed on the 3rd January 1996 and the 1800mm conveyor was commissioned during Easter 1996.

Spontaneous combustion

Sub bituminous thermal coals of the Hunter Valley have a history of spontaneous combustion events in both longwall and bord and pillar mining operations. In order to determine the propensity of the coal seams at Dartbrook to spontaneous combustion, a series of laboratory tests were undertaken and expert advice sought.

Based on the four tests

- Relative ignition temperature;
- R 70 index;
- Initial rate of heating; and
- Total temperature rise.

It was decided that coal seams in the Dartbrook lease had a medium to high propensity for spontaneous combustion and this would have to be a major factor in the mine design. The risk rating used indicated that the thickness of the seam and the amount of coal left in the goaf to be of particular importance.

The longwall ventilation system was designed as a relatively simple "U" type system with no goaf bleeders. It was also recognised that effective seals and an accurate and reliable monitoring system would be a pre-requisite for safe longwall mining. When one million tonnes of coal was mined from Longwall 1 approximately 3.5 million tonnes of coal remained in the goaf. With the normal oxidation rate of coal we were running at 120 to 150 litres per minute CO make after 5 months of longwall mining. The Dartbrook conditions are very unique and cannot be compared to any conditions anywhere in Australia or overseas. Expert advice suggested that CO make could not be utilised as an accurate indicator of spontaneous combustion. The racking of CO levels and Grahams ratio (GR) and action response plans were incorporated in the Dartbrook Underground Environmental Management plan (DUEMP).

During April 1997 the CO levels were stable at around 400 ppm at the tailgate intersection of the installation roadway approximately 500 metres behind the faceline. On April the 15th the CO level rose from 450 ppm to 600ppm within 6 hours. At this time the Grahams ratio had risen to 0.62. All persons were withdrawn from the mine at 3.30 pm in accordance with the Spontaneous Combustion Management Plan (Action response plans) and a series of bag samples were taken and analysed by chromatography in Muswellbrook, Maitland and the Mine Rescue Mobile Laboratory to confirm results. The results indicated 80 to 100ppm of hydrogen from the tube bundle point at the edge of the installation roadway. At around 11.30 PM the CO at the installation roadway was 690 ppm CO with a GR of 0.72. Ventilation pressures were reduced across the longwall face by regulating the longwall return. The CO readings started to immediately decline and after 24 hours had settled back to normal. It was shown later than hydrogen was found in a number of vertical boreholes drilled from the surface for exploration and for hydofrac trials. Helium gas was also discounted from the readings to show a true hydrogen reading. It was later suggested that hydrogen may be a seam gas given off at a lower oxidation temperature.

A similar incident occurred during June where people were again withdrawn from the mine. The cause was believed to be the absence of stoppings in the toxic return inbye the last ventilation cut-through inbye the tailgate. The DMR would not allow these seals to be installed by Mines Rescue teams under breathing apparatus at concentrations above the statutory working limits. After the second incident drop doors were erected in the toxic return every 200 metres which maintained the ventilation fringe shallow behind the longwall face line. No further problems have been encountered since.

Hydrology

Hydrology studies were undertaken to determine water make on development and from longwall goaf caving. Although these studies produced a wide range of results Dartbrook adopted a risk averse strategy to install sufficient pumping to withstand the worst case scenario. Despite the high quantities of water experienced during construction of the Hunter Tunnel, the current working area and longwall goaf is only producing around 1.5 litre per second. The only water created in the mine workings is typical nuisance water, which we create during the mining process.

TECHNOLOGY

Working at 3.9 metres height and longwalling at 4.5 metres presented some interesting challenges and to some extent was underestimated. Unfortunately you cannot procure standard equipment off the shelf to operate at this working height. The question was asked by many people, “why don’t you mine at around 2.8 to 3 metres height to match the mining equipment available and ramp up from the gate roads for the longwall like the US mines do”. We took the view that higher is better for ventilation efficiency, and to get larger pieces of equipment underground. We saw opportunities to design equipment and systems to mine and install services at this height and our people have designed and developed some very innovative solutions to problems. The 3.9 metre high roadways allowed conveyor belts to be installed against the roof, which allows for machines to move under the belts and makes belt cleaning less labour intensive. It also allowed for gas drainage drilling of the longwall block.

In the current working area of the lease the seam section comprises of three working sections, the Wynn upper A, B and C sections. The Bayswater B seam section also joins the top of the Wynn section and results in a combined coal section of 18 to 24 metres. Dartbrook typically mines 3.9 to 4.0m on development and mines up to 4.5 metres with the longwall to optimise coal quality. Working at heights greater than 3 metres presents a whole range of problems. How to install conveyors, pipes, cables, etc at roof height. A number of very innovative designs were developed to manage the tasks. A platform 1.5 metres of the ground was installed on the Voest Alpine ABM20’s to provide a safe working area for operators and ease of installation of supports and ventilation mono-rails. Purpose designed platforms were manufactured for the installation of pipes and the 1800mm main conveyors. A double crawler mounted tailpiece was designed with a conveyor installation platform to dispense with the development coal and install the 1500mm gate road conveyors. The 3.9 metre height in the development units allowed the opportunity to install a specially designed ventilation and cable management system.

DEVELOPMENT SYSTEMS

Dartbrook's operating structure is based on 5 days per week and 3 by 8.5 hour shifts per day. Initially, our plans were to have 2 production shifts per day and one service shift per day. The idea of the service shift was to not only complete the equipment maintenance orders, it also completed all the mining related activities such as belt extensions, DCB moves and installs 60 metres of supplies on the continuous miner ready for the production shift to have a press button start.

Equipment was designed to enable the conveyor and services to be extended by short increments on the service shift. Unfortunately the development equipment available on the market did not suit the operational aspects of the 2 production and 1 service per day concept as well as the 3.9 metre working height. The equipment size and complexity of the tasks did not provide the 10 quality shifts of production per week desired, so a cyclic system was introduced to allow maintenance activities to fit with the completion of a development cycle.

A new mine has the opportunity to install the latest technology available to the Industry. Dartbrook selected three Voest Alpine ABM 20 Continuous miners with four hydraulic roof bolters and two rib bolters. The ABM20's have an installed horsepower of 542 KW with a 270 KW cutter motor, 2 by 36 KW conveyor motors and 2 by 100 KW pump motors. The TRS canopy provides a footprint force of 2 by 200 KN. The ABM20 loads at the rate of 25 tonnes per minute. Roof support is via four by 4000 series semi-automatic Hydramatic Engineering roof bolters and rib support is provided by 2 by series 5000 hydraulic rib bolters. The support pattern is four 2.1 metre roof bolts at 1 metre centres with 6 bolts through gate road intersections and 4 to 6 rib bolts per metre of advance. Typical mining conditions are shown in Fig. 1

During the evaluation process for suitable development machines for Dartbrook, the Voest Alpine ABM20 Miners was seen as the only tried and proven machine with the capability of installing roof and rib support contiguous with the cutting activity.

Considerable redesign of the machine was undertaken with Voest Alpine to permit the machine to cut to 4.3 metres height and to install an on board sizer. Although the Wynn Seam at Dartbrook has relatively strong coal we did not want operators working between large single pass machines and 3.9 metre high ribs. A work platform was designed for the operators to stand 1.5 metres off the ground, be able to touch the roof and have all tools and supplies at their finger tips. The excavation of an overcast in shown in Fig. 2.

The platform was designed for operator convenience with a bolt box on either side of the machine located immediately behind the operator. Baskets were designed for chemicals, mono- rail fittings etc. Racks were fitted along the sides of the platform rails for spare drill steels and mono-rails. The ABM20 can carry sufficient supplies for 60 metres of roadway drivage. What was considered as one of the hardest and most hazardous jobs in a coal mine is the easiest job at Dartbrook.

Horizon control is managed by setting the working height on the Penpeck radio control system and referencing from the tuffaceous floor. Although the machine is very close to 90 tonne in weight it has low ground pressures and does not break up the floor on intersections. The ABM20's from day one have averaged approximately 5 metres per cutting hour and at times have achieved over 7 metres per hour.

Joy 15SC-32 shuttle cars were chosen basically because there were no other shuttle cars on the market and after mixed results with mobile conveyors at Capcoal in Central Queensland we opted for tried and proven coal clearance systems. Joy Manufacturing were contracted to supply shuttle cars that hold 15 tonnes of coal and after considerable modifications we now have a 15 tonne car which permit us to cycle two cars per metre of advance. The seat on the 15SC's were raised by 500mm which significantly improves the visibility of the driver.

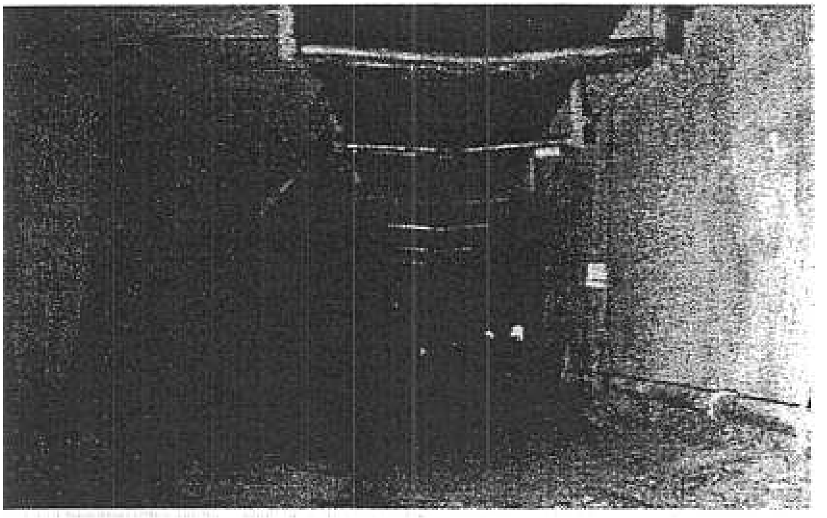

Fig. 1 – Typical mining conditions

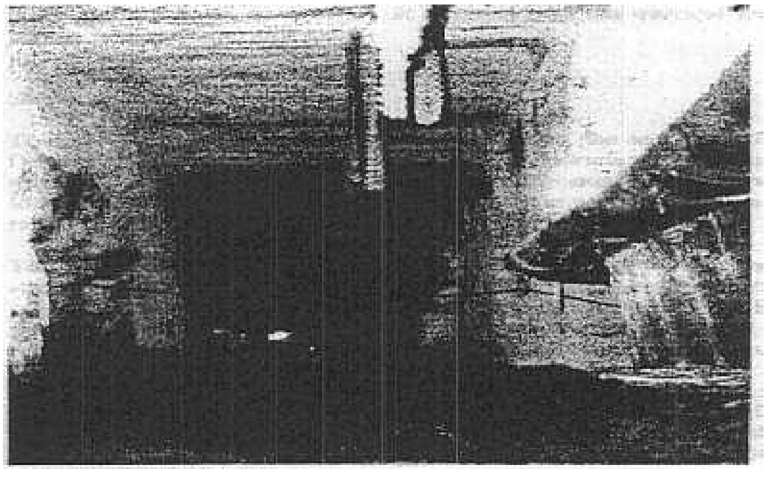

Fig. 2 -Eight (8) metre seam section excavated for overcast

With a new project and a "clean sheet of paper" a systems approach was used to design supply systems, ventilation and development systems. Materials handling is one of the most labour intensive activities and has the greatest potential for accidents in coal mines. Dartbrook designed a supply system of delivery from the manufacturer to the Continuous miner which involves the supplier loading and delivering boxes full of roof and rib bolts and trays that hold 60 straps. These are unloaded onto the ground or direct onto heavy duty trailers on the surface. Eimco EJC 130 LHD's tow the heavy duty trailer to the development panel were bolt boxes and strap trays are transferred to the ABM20 by a QDS hyab crane or by Eimco platforms. A panel support vehicle (PSV) was designed for this purpose and is currently away for modifications to install the bolt boxes onto the miner. The PSV is a crawler mounted machine with a flat deck and a hiab crane arrangement.

Face ventilation is provided by a 720mm diameter monorail mounted ventilation system which is attached to, and advanced by the ABM20 continuous miner and retreated by using a crawler mounted fan. The vent system also carries the miner cable and 50mm water and compressed air hoses which has resulted in substantial time savings in cable and hose movements. The vent system is a combination of fibreglass and flexible ducting which runs on 1.5 metre lengths of round mono rail.

To enable the 1500 mm gate road conveyor to be installed at roof height and in short increments on the service shift a Development Tail End (DTE) was designed jointly by Dartbrook / ACE and MTA. It is double crawler mounted and has two MBS bolters mounted on the front above the tailpiece to install roof bolts for the belt structure. The unit has side shift and levelling facilities for belt alignment and the unit is aligned by lasers and perspex site boards mounted on the side of the machine. Two lifting platforms are on the outbye end of the unit to enable the belt structure to be installed at roof height. Initially a DCB mounted on a sled was towed behind the DTE with a reticulation cable in a basket behind it. We now have the DCB maintained back at the section transformer and a mono-rail mounted cable and ventilation management system, which advances and retreats with the miner. The ventilation fan is crawler mounted and is used to advance and retreat the system. An outbye compressed air driver moves the reticulation cables outbye the fan. A 30 metres structure pod is positioned behind the DCB sled under the belt ready for the belt extensions on the service shifts.

EXPERIENCE OF THE FIRST LONGWALL BLOCK

The Dartbrook longwall is currently mining the second longwall block. With 18 to 24 metres of coal we have a very large reservoir of gas which have been extensively drilled but experience on Longwall 1 showed that gas capture above the mining horizon was only around 30%. Within the goaf envelope there is a total of three seams with a combined gas content of around 50 cubic metres per tonne extracted. It is difficult to accurately model how much gas is liberated from the goaf but experience from Longwall 1 shows that around 5 litres per second of gas has to be managed by the ventilation system and post drainage from vertical goaf wells and from seals behind the longwall face.

The Wynn seam at Dartbrook is longwalled at between 4.0 and 4.3 metres in height. The first longwall block was 2.2 kilometres in length and Longwall 2 is 2.4 kilometres. The longwall's retreat to the rise with goaf water pumped by a borehole pump at the back of the blocks. Longwall 2 is ventilated by a simple "U" type homotropal ventilation system with no bleed system due to the risk of spontaneous combustion. The first longwall face was installed on a grade of 1 in 5 for 80 metres of the 200 metre face line. The steep grade was managed without problems. We took the view that steep grades will be experienced in several areas and we need to learn how to manage them from the first longwall. The initial faceline is shown in Fig. 3.

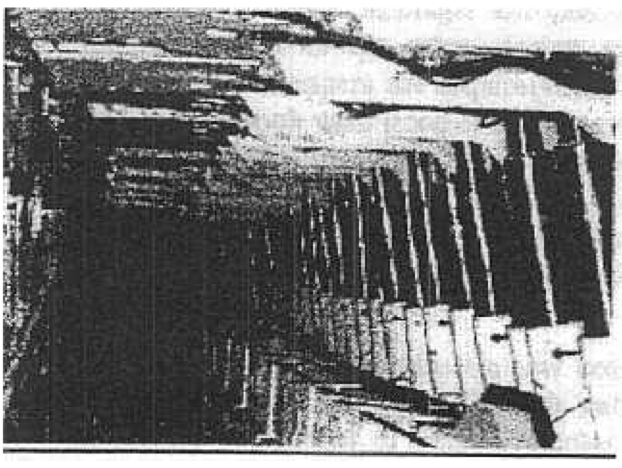

Fig. 3 - Longwall 1 faceline on startup

The longwall was supplied by Mine Technick Australia (MTA) and is 200 metres in length with sufficient horsepower to extend to 250 metres in the future. The Maingate equipment is installed on three self propelled crawler mounted trailers, which are retreated under the roof, mounted belt outbye of the Longwall Tail End (LTE). The LTE is a MTA/ ACE / Dartbrook designed unit which is skid mounted under the BSL and Crawler mounted on the outbye end and elevates the coal to roof mounted conveyors. The LTE has structure recovery platforms on either side at the outbye end where the structure is recovered and installed in 30 metre structure pods ready for installation in the gate road panels. The specifications of the longwall equipment is shown in Appendix 1. Fig. 4 shows Dartbrook roof supports and AFC.

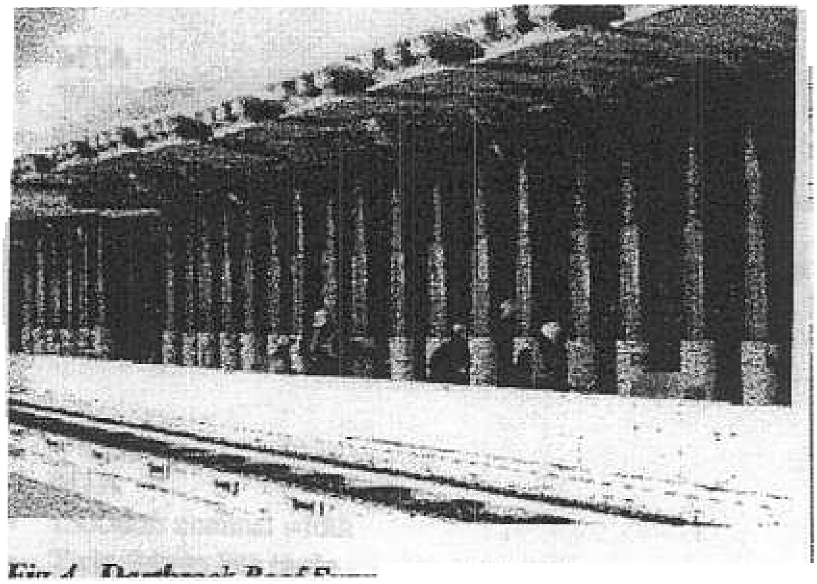

Fig. 4 - Dartbrook roof supports and AFC

Longwall changeover

The first longwall changeover was planned for 20 days and this was not achieved due to delays with the tailgate drive and crusher overhauls and a number of electrical problems. The longwall equipment move was actually completed in 15 days but we did not start cutting coal until 28 days after we started to bolt up the face. The Geogrid mesh, 14 metres in width was used in conjunction with one can and one timber crib per support during shield recovery. This process saved at least 4 days on the recovery and provided a safety barrier to prevent the goaf from flushing into the faceline during the recovery of shields.

CONVEYORS

For a high capacity longwall system the mine required a reliable high capacity coal clearance system. Like most modern underground mines in Australia and the USA, Dartbrook elected to not include a surge bin or bunker in the coal clearance system.

The main conveyor system is 1800mm in width operating at 4 metres per second speed. This gives a designed capacity of 4,200 tonnes per hour or a spill capacity of 5,300 tonnes per hour. The Hunter Tunnel conveyor hauls coal from the mining area to the west of the Hunter River 4 kilometres underground to the coal handling facilities on the eastern side of the New England Highway and main Northern Railway line. The first conveyor from the surface (HT01) is powered by a 2.2 MW drive and hauls coal up a 600 metre long 1 in 5 drift. The second in line conveyor (HT02) is 960 KW and roughly half way along has a 960 KW tripper drive. The drive units were designed by Dartbrook and manufactured by ACE Conveyors with the winches provided by Nepean Engineering. The gearboxes were supplied by Flender and CST drives by Dodge.

The maingate conveyors are also powered by the same drives as the mainraod conveyors (3 X 320KW power packs and CST drives. The gate road conveyors also have a tripper drive installed (960KW) to permit the haulage of longwall coal upgrade for 2.4 km. The maingate conveyors are 1500mm wide and have a designed capacity of 3200 tonnes per hour. After some compatibility problems with the CST software and minor winch programming problems the belts are settling down well. Coal on the surface is delivered to a 1100 tonne bin or can be diverted by a lufting boom to a 50,000 tonne emergency stockpile. The coal from the bin is fed through sizers and Syntron feeders, through to stackout on to two 200,000 tonne stockpiles. Coal can also report direct to the 1000 tonnes per hour washery.

PLANT CONTROL SYSTEM

To understand and manage equipment and the mining environment a comprehensive and responsive control and monitoring system is a must. After evaluating a number of systems a decision was made to install the Windows based Citect Plant Control System. The system is working extremely well and is very "user friendly". Dartbrook has a control centre manned 7 days per week to monitor the Citect system and acknowledge and respond to alarms.

Underground monitoring-

- Maihak Tube Bundle gas monitoring system;
- AMR environmental telemetry system;
- Continuous miners;
- Conveyor belts;
- Fans;
- Power distribution 66/11 KV;
- Gas drainage plant;
- Longwall;
- Vertical goaf gas drainage pumps; and
- Dewatering pumps.

Surface monitoring

- Compressors;
- Water reticulation ;
- Sewerage plant and water treatment plant;
- Irrigation system
- Office security and fire alarms;
- East site coal handling plant;
- East site bins and conveyors;
- Stackers and reclaimers;
- Train loading bin and system;

- Weather station

The system is PC based with Allen Bradley Programmable Logic Controller (PLC) and Small Logic Controller (SLC) equipment to provide data on a regular basis, which is collected by the Citect software for display to the control room operators and a number of site offices including the mine managers office.

FUTURE PLANS

Dartbrook is currently undertaking exploration work to the west and north of the current mining lease to extend the reserve base. Dartbrook needs to produce in the vicinity of 3.5 million tonnes per annum to provide the shareholders with an acceptable return on capital employed. Gas drainage is a large cost impost on any mine but at Dartbrook it represented around $6.00 per tonne in 1997 which has to be reduced for the mine to survive. Gas drainage is time dependent and we now have pre-drained for the next two longwall's beyond Longwall 2 and have reduced our gas drainage effort by one third. A lot was learnt on Longwall 1 which has allowed us to reduce the amount of pre-drainage with optimum drilling patterns and with the experience of successful goaf drainage holes. We believe that we now understand the parameters for effective pre and post gas drainage for longwall mining and understand the fundamentals to manage the risk of spontaneous combustion.

SUMMARY

Dartbrook Mine has not reached world class as yet but the basic elements are in place to provide a safe, efficient and profitable business.

The same level of enthusiasm and energy is still evident at Dartbrook that existed when the mine started 3 years ago. A high level of trust exists between all parties at Dartbrook and this helped through the early operational problems of learning to work at heights and the development of suitable equipment and systems to mine a thick coal seam full of gas and achieve high advance rates on development and planned tonnages of the longwall.

In summary the advice to any new Project Manager would be to allow sufficient time to gather and interpret an extensive data base of knowledge on the resource and to establish the best mine plan and cost structure. Invest money and time to employ the best people, train them well and have the best possible labour agreements in place. This is the best chance you will ever get to set the mine up on a stable foundation and take all the advantages presented by a greenfields opportunity.

APPENDIX 1

LONGWALL SPECIFICATIONS

MTA;

- Shearer Initiation;
- 116 X Two leg 913 tonne shields;
- 1.75 metres wide shields;
  Range 2.2 metres to 4.8 metres;
- Face support mass 26.56 tonnes;
- Gate road support mass 27 tonnes;
- High strength steels 700 UTS;
- Support density 118 tonnes per sq metre;
- Full automation and data transmission to surface through Citect system;
- PM4 electro hydraulic support control units (SCU's);
- Leg pressure and push ram displacement transducers;
- Water sprays in canopy's; and
- Rear walkway in canopy down to 3.3 metres operating height

Shearer

- Long Airdox (Electra 1000);
- Cutting range 2.5 m to 4.5 m;
- Installed power 1332 KW, 3.3KV;
- Cutting drum diameters 1.9m to 2.5 m;
- 2 X 500 KW cutter motors;
- 2 X 56 KW DC haulage motors;
- 200 KW lump breaker;
- 20 KW hydraulic pump motor; and
- Hiab type crane on MG end.

Pumping system

- MTA;
- 3 X 275 l/min @ 350 bar Hauhinco hydraulic pump sets mounted on a crawler mounter trailer;
- 250 KW motors on each pump;
- 10,000 litre stainless steel tank with 1,500 litre mixing tank;
- High pressure shearer water pump mounted on crawler mounted trailer with tank.

AFC

- MTA
- Width 1150mm;
- Capacity 3200 tonnes per hour;
- Twin 42 mm Compac link chain;
- Automatic chain tensioning at Tailgate;
- 2 X 800 KW motors driving CST's at Maingate and Tailgate;
- Slow chain speed running; and
- Provision for fitting of pan tilt cylinders.

BSL

- MTA;
- 1500mm nominal width;
- Twin 34mm link chain;
- 350KW drive motor;
- Slow running device fitted; and
- Full length dust suppression system

Crusher

- MTA (Westfalia Becorit);
- 1800 mm width;
- High Inertia impact roller; and
- Size coal to minus 150mm.

Management of Seam Gas Emission and Spontaneous Combustion in a Highly Gassy, Thick and Multi Seam Coal Mine — A Learning Experience

R Moreby[1]

ABSTRACT

This paper describes operational experience with seam gas and spontaneous combustion management, gained during ten years of production history (1993 to 2003) in the Wynn seam at Dartbrook Coal mine. In addition, comparisons are made with more recent operational experience in the overlying Kayuga seam.

Development and retreat longwall operations were undertaken in a 4.0 m thick working section of the Wynn seam, contained within the lower quarter of a circa 24 m thick mega seam, from 1993 until mining operations were relocated to the overlying Kayuga seam in 2004. Production from the first block of the Kayuga seam is currently underway with seam gas and spontaneous combustion management requirements being refined as further operational experience is gained.

Seam gas management techniques, including pre-drainage with pressurised fracturing, surface to underground goaf drainage of a carbon dioxide rich seam gas and novel, high capacity ventilation circuits, were employed in the Wynn seam. Ventilation strategies included operating up to carbon dioxide STEL concentrations of 3.0 per cent in segregated return airways rather than being limited to the TWA-TLV concentration of 1.25 per cent.

The spontaneous combustion experience at Dartbrook Coal has, to some degree, resulted in a change in the way the risk is managed in Australian coal mines. Values of carbon monoxide make and hydrogen concentrations are profoundly different from previously recommended management plan trigger levels and those used elsewhere in the industry. However, the detection of higher hydrocarbons together with rising oxygen depletion ratios and absolute hydrogen concentrations remain as indicators of advancing spontaneous combustion events. In this respect, experience gained in the Wynn seam was important in the initial management of the shallower Kayuga seam, an essentially greenfield site with no prior operational experience being available.

In 2002, very problematic conditions were encountered in Wynn seam longwall 7 that had passed through a fault zone and soon after also suffered a major fall on the face line. During the complex and protracted recovery of this block, spontaneous combustion indicator gases were observed in very abnormal, and previously unseen, ratios. In addition to conventional gas monitoring, underground thermography surveys and surface radon flux tests were employed to determine the location of the event. These techniques are being further developed and, together with learning experience in the Wynn seam, being applied to decision making in upper seam workings.

DARTBROOK MINE

Dartbrook mine is situated in the Wittingham coal measures of the Upper Hunter Valley of New South Wales, approximately 10 km north of Muswellbrook. Eight coal seam groups occur in the area, five of which have been identified as suitable for longwall mining, namely the Wynn A, Mt Arthur, Kayuga, Piercefield and Broonie seams. The targeted thermal coal seams are of late Permian age and rank classified as high volatile B bituminous.

A plan of the Wynn seam workings, as at mid 2002, together with an indicative stratigraphic section and detail of longwall 7's ventilation circuit are shown schematically in Figure 1. The spontaneous combustion history of longwall 7 is discussed below, however, the ventilation circuit shown is typical of the high capacity, homotropal two heading circuit employed previously in longwalls 2 to 6.

In 2004, mining was completed in the Wynn seam (longwalls 1 to 11) and mining operations were relocated to the shallower Kayuga seam. Kayuga seam longwall 1 commenced production in June 2004.

Prior to commencement of initial Wynn seam development in 1993, the target seams underwent detailed characterisation to assess hazards and quantify the controls required to minimise associated risks (Moreby 1995, 1997). Risks identified during initial phases of exploration included:

- medium to high propensity for spontaneous combustion,
- high gas emission during development,
- very high longwall gas emission rates, and
- potential for carbon dioxide outbursts on geological structures.

These issues arose principally from the significant gas reservoir contained in adjacent seams and many extrinsic factors influencing the risk of spontaneous combustion in active and sealed goaves. In particular, the need for high ventilation rates and goaf drainage from longwalls containing some 15 m of immediate roof coal.

Based on this initial assessment and later operational experience, the main controls put in place included:

- high capacity ventilation circuits including pre-holing of subsequent gate roads;
- use of STEL 3.0 per cent CO_2 in segregated longwall returns to maximise gas management capacity;
- exploratory drilling and core sampling ahead of development;
- intensive in seam pre-drainage of the working section and, initially, the immediate roof;
- high capacity goaf drainage plants on active and sealed goaves;
- main seal balance chambers;
- intensive gas monitoring including real time sensors, tube bundle system and, from 2000, a gas chromatograph; and
- underground environmental management plans with site-specific action response trigger levels.

In the Wynn seam, production was constrained to circa 80 000 tpw (initially on a five then later a seven day basis) due to seam gas emission combined with various geological and operational issues. In particular, the requirement for concurrent management of high gas emission rates and spontaneous combustion in active goaves.

Primary ventilation circuit

During mining of the Wynn seam, the ventilation circuit comprised two intake drifts (Hunter Tunnel/Western Drift), a 6.0 m diameter intake shaft together with a single 4.4 m diameter

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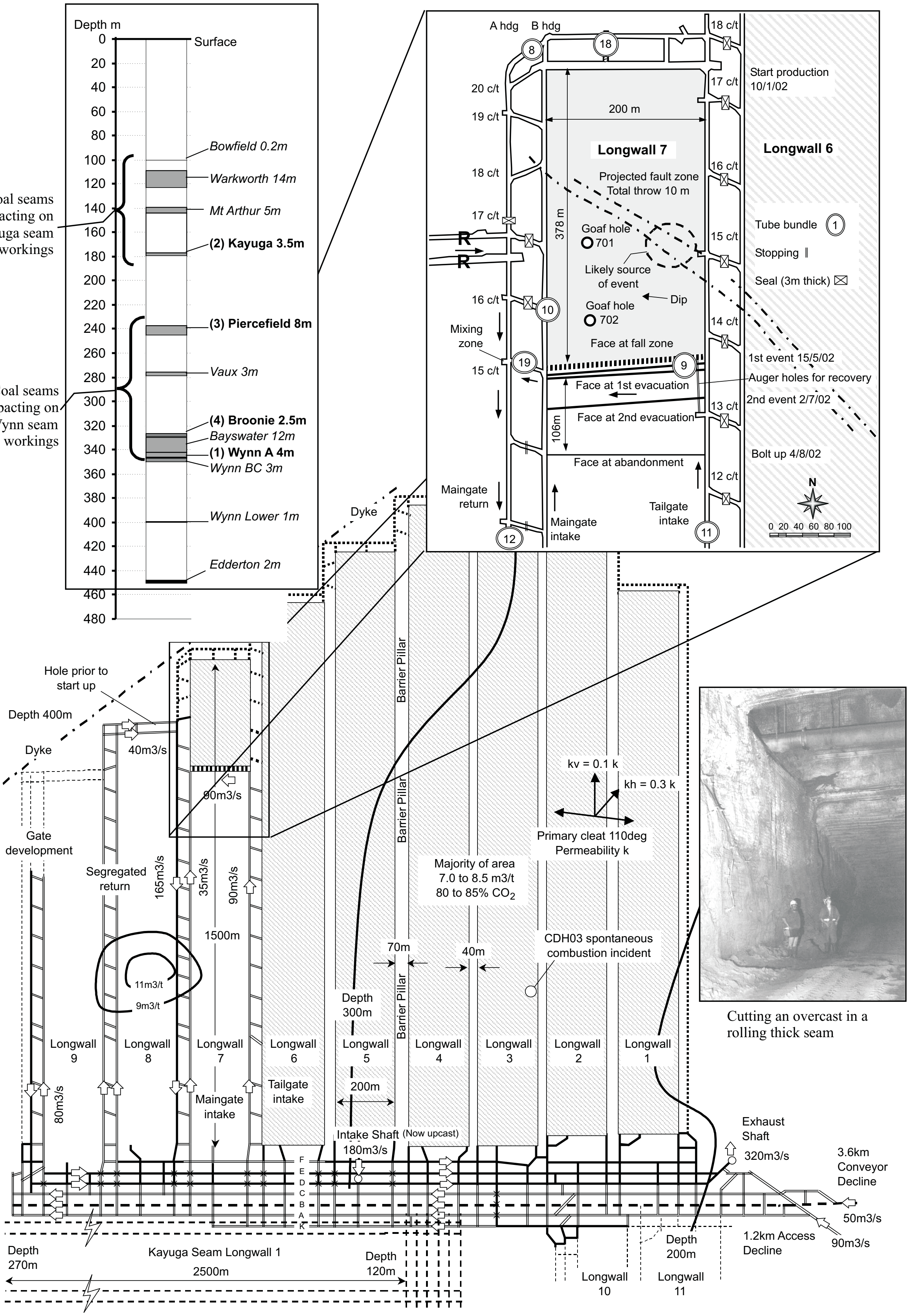

FIG 1 - Schematic mine plan (mid 2002), stratigraphic section and longwall 7.

steel lined exhaust shaft. The surface exhaust fan comprised three 2.8 m diameter variable speed centrifugal fans with a duty of 320 m^3/s at 2.5 kPa. The primary underground circuit involved three to four main intake roads, two returns inbye the maingate and three outbye returns. Development excavation profiles were 3.8 to 4.0 m high by 5.2 m wide of rectangular section.

Development stoppings were constructed from grout and sheet or block mesh with ribs sprayed back to minimise leakage. Permanent mains stoppings were 150 mm thick grout-filled block mesh.

All goaf seals were explosion resistant (>20 psi), consisting of poured 30 MPa concrete plugs, circa 3 m thick in tailgate and 5 m thick in the mains. Additional grout-filled steel column supports 'cans' or timber 'cribs' were placed either side to minimise damage from abutment stresses.

A number of alternative seal designs were trialled through longwall 1 but, due to applied stress resulting from separation of thick roof coal plys, thin rigid (600 mm at 60 MPa) or thick soft (3 m at <10 MPa) seals failed to maintain integrity.

Kayuga seam ventilation

To facilitate relocation of mining operations to the Kayuga seam, while production continued in the Wynn seam, a fourth surface fan was purchased and installed on No 2 shaft when holed from Kayuga mains development. This allowed surface fans to be relocated in sequence while maintaining high volumetric capacity in both mining horizons. Currently, four surface fans are installed on No 2 shaft, and No 1 shaft is down casting.

Being subject to significantly lower seam gas emission rates, Kayuga seam workings are planned to operate at 120 000 tpw on a seven day per week basis, using conventional two heading circuits without pre-holing subsequent gateroads and, possibly, with an increase in face width from 200 to 240 m. Available gas management controls have been, and continue to be, modified in response to differing operational requirements in the Kayuga seam compared with those encountered in the Wynn seam.

SEAM GAS MANAGEMENT

All seams in the Dartbrook lease contain a multi component seam gas comprising mainly carbon dioxide (CO_2) and methane (CH_4). In the Wynn seam, gas contents range from 6.5 to 11 m^3/t (total desorbable) with CO_2/CH_4 ratios ranging between 90:10 per cent and 70:30 per cent. Trace amounts of nitrogen and hydrogen sulfide have also been detected, but not in sufficient quantities to influence identified hazards.

Gas contents reduce with depth in shallower seams, with Kayuga seam gas contents ranging <1.0 m^3/t at 100 m to 5 m^3/t at 350 m (Figure 2).

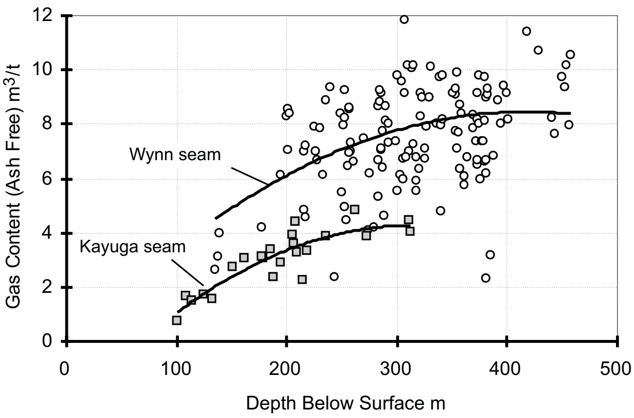

FIG 2 - Gas content and depth relationships.

In the Kayuga and other shallower seams, there is a greater variation in gas composition with CO_2/CH_4 ratios ranging from 90:10 per cent to 50:50 per cent. These variations occur in plan position with little relationship to depth or geological features. Consequently, management of both carbon dioxide and methane will remain an important aspect of mine design but with variations in the magnitude of gas emission in upper seams.

Gas reservoir characteristics

To generate empirical and theoretical models of reservoir behaviour during development and longwall operations further laboratory studies were undertaken to more completely characterise the reservoir. The results of this work are provided in Table 1.

TABLE 1

Wynn seam reservoir characteristics.

Factor	Value	Unit
Depth	210 to 380	m
Fluid pressure	Depth m - 30	mH_2O
Stress	5 to 6	MPa
Stress directivity	1.2	σv / σh
Coal UCS	6 to 30 Av 18	MPa
Cleat space	200 to 500	Mm
Orientation	110	deg
In fill	Calcite	
Permeability ratio vertical/horizontal	0.2 to 1.7 0.1	mD
Porosity	10 to 12	%
Linear shrinkage	2.6	%
Diffusivity CO_2	1.85×10^{-6}	cm^2/s
CH_4	8.55×10^{-7}	cm^2/s
Langmuir CO_2	34	m^3/t
Volume CH_4	16	m^3/t
Langmuir CO_2	1400	KPa
Pressure CH_4	2500	kPa
Sorption pressures	930 to 1700	kPa

The relationship between permeability, seam fluid pressure and depth is shown in Figure 3. Interference tests indicated that cross cleat permeability was of the order 0.3 times that along cleat and vertical permeability of the order 0.1 times that along cleat.

The effect of stress on permeability was consistent in all seams with fluid pressure indicating significantly under-saturated conditions. Whilst initial models indicated high gas emission during development and longwall phases, these characteristics also indicated that rib emission and effectiveness of pre-drainage would be strongly dependent on development and hole orientation. In addition, managing longwall gas emission by pre-drainage of roof seams would be problematic due to low vertical permeability and relatively low sorption pressure of the carbon dioxide rich seam gas.

Predicted seam gas emission – development

A parametric study of development gas emission was undertaken with SIMED (a two phase 3D multi component reservoir simulator) by CSIRO. This work indicated that rib emission rates would range from 20 to 50 L/s/100 m being markedly influenced by seam permeability and gas content.

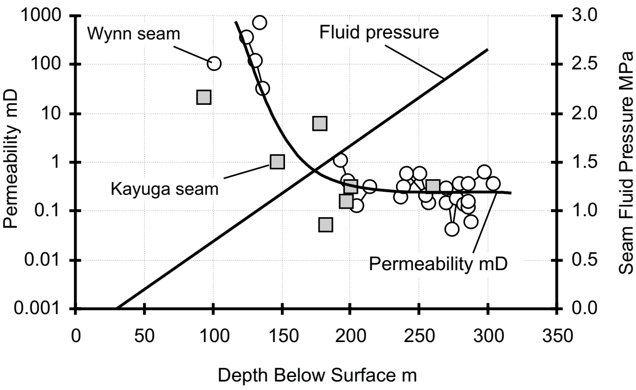

FIG 3 - Permeability and seam fluid pressure.

At 0.44 mD and 8.6 m^3/t (typical for initial Wynn seam conditions) the predicted total gas emission for a 4 km twin heading would be 860 L/s. For an 80 per cent CO_2 gas composition, this would require 55 m^3/s of ventilation for dilution to 1.25 per cent and could be provided for by the proposed ventilation circuit.

Outburst propensity

Some coal seams in the Southern Sydney and Bowen Basins are prone to outbursting with action plan gas content trigger levels between 6 and 9 m^3/t depending on seam gas composition. As carbon dioxide contents occurred above these trigger levels in the Wynn seam, possibly associated with geological structures, an assessment of propensity to outbursting was made.

With the exception of gas content and desorption rate, all other factors indicative of a seams propensity to outbursting, the relatively low stress regime and high compressive strength of coal in particular, pointed to a low risk. However, given the lack of historical data and gas contents present, existing precautions against outbursts were taken in the form of exploratory drilling, pre-drainage and self imposed limits on mining consistent with those used elsewhere.

Predicted seam gas emission – longwall

The geometry of the gas reservoir impacting on various working sections was determined by ply analysis and gas contents obtained from multiple vertical cores. This provided the data for specific gas emission estimates and hence gas drainage requirements. For example, from the core log shown in Figure 1, the total thickness of coal in close proximity to the Wynn seam is 18 m with the potential to release circa 6.5 m^3/t. The base case, steady state specific gas emission rate (SGE m^3 gas per tonne mined) is 31 m^3/t with a further circa 5 m^3/t arising from more remote seams, interburden and contribution from the working section.

At 100 000 tpw (20 000 tpd), an order of magnitude analysis predicted an average seam gas emission rate of 8300 L/s total gas or ≈6700 L/s CO_2. It was therefore immediately apparent that ventilation alone would not be able to manage longwall gas emission, even with exemption conditions to operate at 3.0 per cent CO_2 in segregated return airways. Consequently mechanically assisted in-seam post drainage was employed from the outset.

A number of empirical techniques and pore pressure finite element models (Ashelford, 2003) were also used to estimate longwall gas emission in the Wynn seam. It was found that, while empirical models were developed for methane emission and do not take into account the difference in sorption pressures or diffusion rates, they do provide an acceptable quantification of steady state gas emission above about 15 000 tpd. At lower daily production rates, specific gas emission (m^3/t) is higher than predicted by empirical models, due to reduced reconsolidation, however the net gas emission (L/s) does in fact reduce. Therefore both empirical and more sophisticated pore pressure models were found to be appropriate.

Operational experience

The mine started production in the Wynn seam expecting significant seam gas management issues even with pre- and post-drainage systems in place. However, it took time to refine application of these techniques and increase capacity accordingly. A typical distribution of carbon dioxide emission, from about longwall 3, is shown in Figure 4. Methane emission at this time amounted to circa 2000 L/s and was not a significant management problem compared to that of carbon dioxide.

With an acceptable balance between management of seam gas and spontaneous combustion, circa 37 per cent of seam gas emission was captured by pre- and post-drainage with the balance reporting to longwall and gate road ventilation circuits.

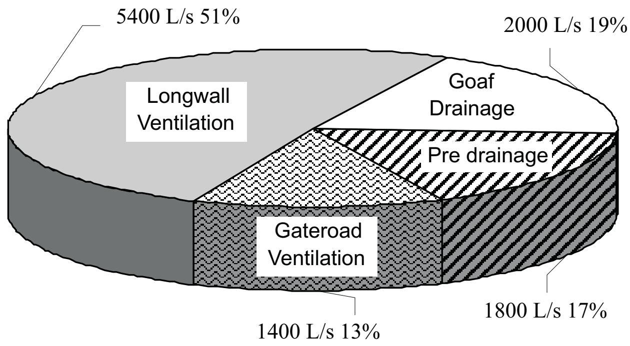

FIG 4 - Wynn seam – typical CO_2 emission distribution (total 10 600 L/s CO_2).

Development gas emission

The first significant step change in gas emission occurred when north – south orientated (cross cleat) gate roads were turned off from the east – west (subparallel to cleat) orientated mains. The resultant effect on pre-drained rib emission decay rates are shown in Figure 5, with actual gross gas emission reporting to gate road ventilation circuits shown in Figure 6. For clarity, only values for carbon dioxide are provided. Methane emission during development was between ten and 15 per cent of the total, ie 100 L/s CO_2 was associated with about 15 L/s CH_4.

Initial emission rates exceeded 80 L/s/100 m (effective for less than one shift) decaying over approximately ten days to 25 L/s/100 m and then to 15 L/s/100 m over the next ten months.

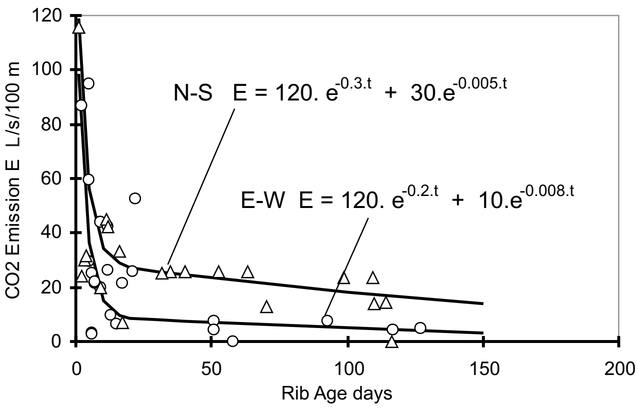

FIG 5 - Wynn seam rib emission decline curves.

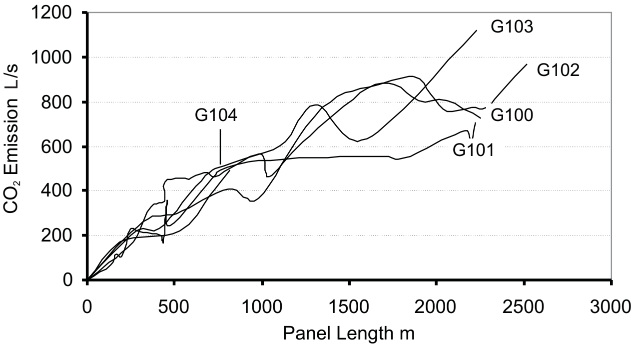

FIG 6 - Wynn seam gate road emission rate.

Rib capture and cross block pre-drainage holes were required for outburst control in any event and were drilled in an attempt to reduce this rate to less than 5 L/s/100 m. However, with large amounts of coal in the immediate floor and roof of the working section, this was not operationally or economically practicable and higher ventilation rates of 60 to 80 m^3/s were employed instead. In two heading gate roads, the return limit had to remain at 1.25 per cent CO_2.

This total gas make to the ventilation system during development was significantly greater than that anticipated from pre-mining characterisation. This was due, principally, to the difference between seam macro permeability and that measured in the laboratory or from single *in situ* draw down tests. It is likely that the effect was exacerbated by a further increase in permeability with shrinkage of the thick roof coal as gas desorbed. For this reason, roof extensometers at Dartbrook were often observed to go up instead of down.

The highest gas content encountered was 9.1 m^3/t (total desorbable) at 90 per cent CO_2 adjacent to a 1 m thick dyke. Seam gas pressure was measured at between 0.9 and 1.1 MPa within 3.0 m of developing faces. To date, with appropriate pre-drainage and compliance testing, no outburst events have occurred in any of the workings undertaken at Dartbrook.

Longwall gas emission

The observed relationship between carbon dioxide and methane specific gas emission with production in the Wynn seam is shown in Figure 7. By way of comparison, Kayuga seam values are shown to reflect the significant difference in reservoir properties. Observed values in the Wynn and Kayuga seams were consistent with predicted values with a characteristic reduction in specific gas emission rate with increased production as indicated by pore pressure models.

Peak gas emission, as a ratio of daily maximum to daily average, was found to range 1.1 to 1.25. This was to be expected

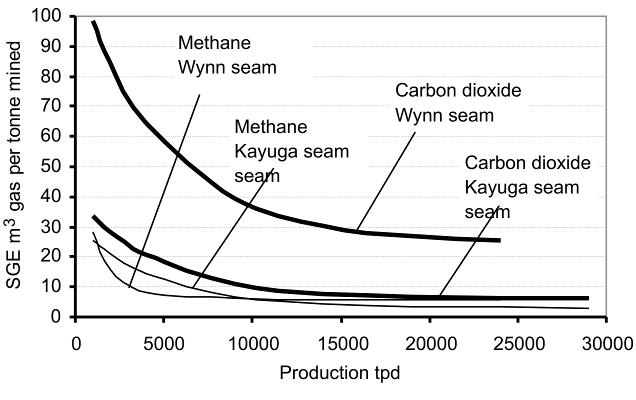

FIG 7 - Wynn and Kayuga seam observed specific gas emission.

due to the relatively even caving characteristics of the soft coal roof. Allowing for these peak emission rates and residual rib emission in intake airways, the observed relationships between gross Wynn seam longwall gas emission and production are shown in Figure 8.

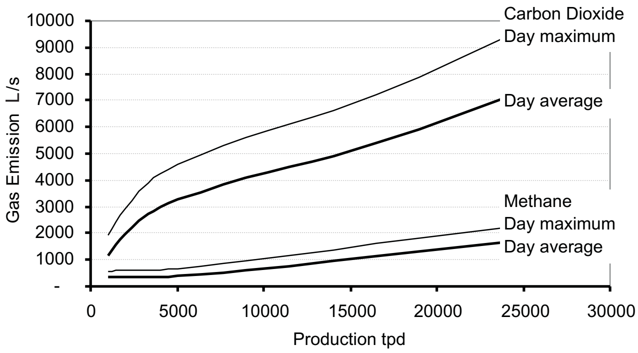

FIG 8 - Wynn seam longwall gas emission.

At plan production rates of 15 000 to 20 000 tpd, peak gas emission was 6500 to 8000 L/s CO_2 and 1500 to 2000 L/s CH_4. With 90 m^3/s air entering the face line from the tailgate intake, together with 35 m^3/s in the maingate travel road and 40m^3/s from the pre-holed gate road, the longwall return carried an average 165 m^3/s. Allowing for rib emission, the net dilution capacity of the longwall circuit at a TLV-TWA concentration of 1.25 per cent was 1500 L/s CO_2 and 4400 L/s CO_2 at the STEL concentration of 3.0 per cent.

Therefore, even with the highest volumetric capacity available and operating the segregated longwall return up to STEL limits, goaf drainage requirements remained up to 3700 L/s CO_2.

Allowing for 80 per cent CO_2 in seam gas and 25 per cent dilution with residual oxygen and nitrogen, the total volumetric capacity of goaf plants had to be 5800 L/s for a realistic gas capture efficiency of 46 per cent. If the longwall return had had to operate at 1.25 per cent CO_2, total goaf drainage requirements would have increased to 10 200 L/s for an unrealistic gas capture efficiency of 81 per cent. The risk of spontaneous combustion within the active goaf would then also have increased significantly.

Pre-drainage of the working section was undertaken and found to reduce immediate face gas emission together with reducing intake rib emission. However, core desorption tests and sampling of cut coal demonstrated that the working section only contributed 1 to 2 m^3/t to the total. That is, more intense pre-drainage of the working section would have had a negligible effect on specific gas emission.

Pre-drainage

The gas pre-drainage system was designed for a flow rate of 4 m^3/s (stp) at -40 kPa surface (20 kPa at collars) and comprised three Siemens ELMO liquid ring pumps (345 kW VVVF drive), 600 mm diameter surface to underground borehole, 600 mm diameter galvanised ERW pipe through mains access and 400 mm diameter galvanised spiral wound pipes in gate roads. In seam pre-drainage drilling was undertaken with drill rigs fitted with BQ rods 89 mm down hole motors and down hole navigation equipment.

Between 60 000 to 80 000 m of drilling was completed annually, depending on the development schedule. Holes were normally drilled in a westerly fan pattern to pre-drain the next block and development corridor. This method maximised lead times and minimised the need for parallel rib capture holes, (Figure 9). The target reduction in gas contents of 50 per cent was aimed for prior to commencement of longwall production.

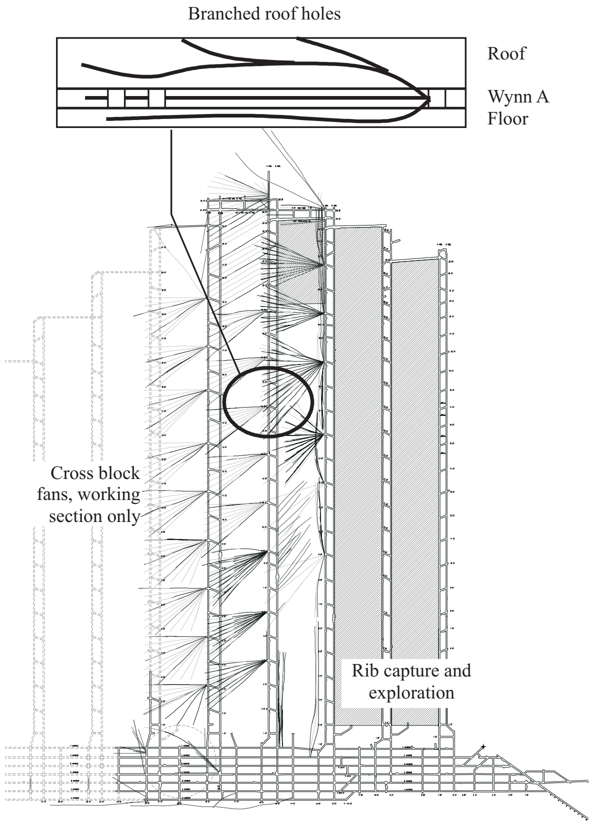

FIG 9 - Wynn seam pre-drainage patterns.

Although pre-drainage often achieved a gas content reduction of 50 per cent or more in the working section, success in roof seams was sporadic. This, together with the need for additional gas management and high operational costs were the driving factors for increasing post drainage from goaf holes. It was simply not economically viable to pre-drain roof seams as a means of reducing longwall specific gas emission rates.

A zone of significantly lower permeability was also encountered in the longwall 7 to 8 area leading to reduced pre-drainage effectiveness with consequently higher face gas outs during mining. The mine successfully overcame this problem by introducing hydro fracture with sand injection to working section holes. Unfortunately, the spontaneous combustion and tailgate fall events in longwall 7 occurred at about the same time, with the benefits of this work not being fully realised.

Post goaf drainage

When Wynn seam longwall 1 commenced production, it was understood that Dartbrook was the first coal mine in the world attempting to goaf drain large quantities of seam gas with such a high carbon dioxide composition. The in-seam gas density was 1.58 kg/m^3 with goaf atmosphere densities of about 1.4 kg/m^3. Although the negative buoyancy did promote development of inert goaf atmospheres, it was not sufficient to prevent ingress of air to active goaves as a result of 100 to 150 Pa frictional pressure drops across the face line.

Goaf drainage holes of 457 mm diameter connected to centrifugal fans (nominal 3.0 m^3/s at 10 kPa) were installed at 200 to 300 m spacing along each longwall block 30 to 35 m from the maingate rib. All plants and individual holes were fitted with monitoring equipment for carbon monoxide, carbon dioxide, methane and oxygen together with speed and pressure transducers.

Up to three plants were operational with each connected to two or three individual holes by 400 mm diameter overland pipes. This method allowed gas to be selectively drawn from near or deep goaf in the active longwall. To control leakage through tailgate seals, during periods of falling or rising barometric pressure, a goaf plant was also operated on, or connected to, the sealed goaf adjacent to the active longwall.

With consideration to goaf gas density and to minimise dilution with air from the face line, the lower casing slider sections were designed to end within, and not on, the goaf pile formed by roof seams (Figure 10).

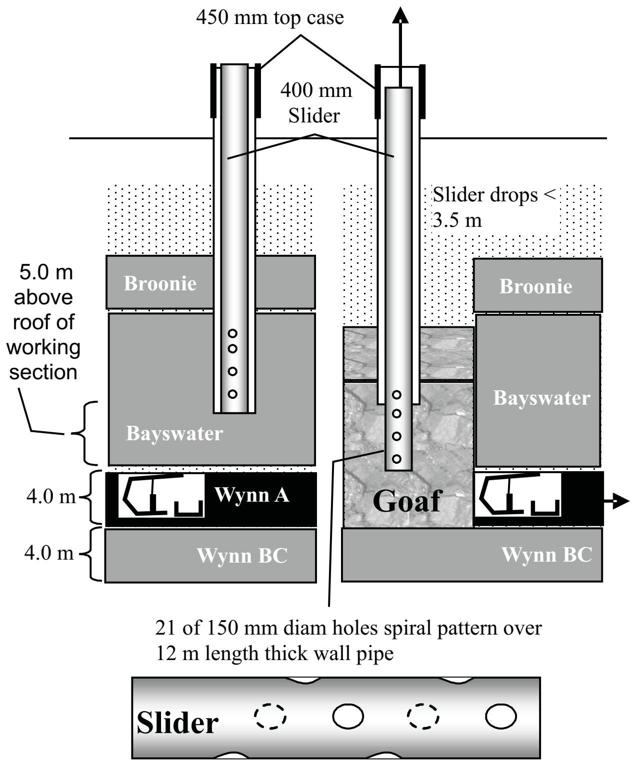

FIG 10 - Wynn seam goaf drainage method.

Goaf drainage plant shut down tests were also undertaken to quantify the effect of deep goaf ventilation on partial pressure and hence seam gas desorption. These tests involved shutting down all goaf plants instantaneously, then measuring the change in gas make reporting to return ventilation until conditions re-estabilised. It was found that only some 60 to 70 per cent of seam gas reporting to goaf plants reported to ventilation when they were turned off. This effect reduced in holes closer to the face being a result of their effect on goaf ventilation and partial pressures acting on desorbing coal.

Tracer gas studies

In order to improve understanding of goaf gas flow patterns, eight tracer gas studies, using sulfur hexafluride (SF_6), were undertaken by CSIRO (Balusu *et al*, 2002). Four tracer gas tests were aimed at quantifying gas flow dynamics, two investigated gas migration between adjacent panels and two studied gas flow patterns around faults/dykes. These tests were combined with modelling of the goaf using computational fluid mechanics (CFD) software to review and refine hole design and optimise flow rates (Figure 11).

The results of tracer gas studies and CFD modelling identified the following issues:

- goaves were highly consolidated 300 m behind the face line;

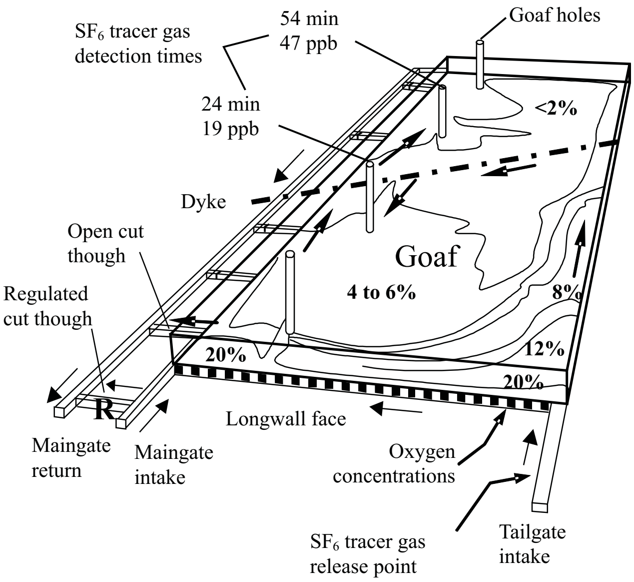

FIG 11 - CSIRO CFD model of goaf flow dynamics.

- installation of tailgate support increased ingress of air 200 to 300 m behind the face line;
- movement occurs along dyke zones, possibly right across the goaf; and
- goaves separated by conventional chain pillars were connected due to failure of seals and or separation of relaxed roof strata.

These results were also employed in the design rationale for the longwall 4 and 5 barrier pillar. Rather than developing an, as then, conventional total barrier pillar between two separate gateroads, segregation was provided by developing a gate road with 70 m C-C pillars and installing 6 to 10 m thick plug seals. Although not a trivial undertaking, this proved effective and significantly less expensive than having to develop an additional gate road.

Kayuga seam – initial seam gas experience

Kayuga seam longwall blocks are orientated east west (subparallel to cleat), with a 4.0 m working section and some 5.0 m coal (Mt Arthur seam) in the immediate roof. Gas contents range 4 to 5 m^3/t at the deeper inbye start line, reducing to less than 1.5 m^3/t at the outbye recovery line. Gas composition in longwall 1 is 65/35 per cent CO_2/CH_4 but changes to about 50/50 per cent in southern blocks.

Based on operational experience in the Wynn seam, together with application of similar characterisation and predictive models, development in the Kayuga seam commenced without pre-drainage of the working section but with provision to install Wynn seam goaf plants in the inbye 1500 m of blocks. Prior to mining, it was assumed that peak gas emission ratios (day peak/day average) would be similar to those in the Wynn seam.

As expected, rib emission in gateroad development is readily managed by available ventilation capacity provided by four fans on No 2 shaft. The first longwall in the Kayuga seam commenced production on 9 June 2004 and, as at end November 2004, has retreated some 1400 m. Observed longwall gas emission rates are shown in Figure 12.

Although specific gas emission rates are some five times lower than in the Wynn seam, peak gas emission rates are up to 1.6 times daily average as a result of more uneven caving of the sandstone/conglomerate strata above the immediate roof coal.

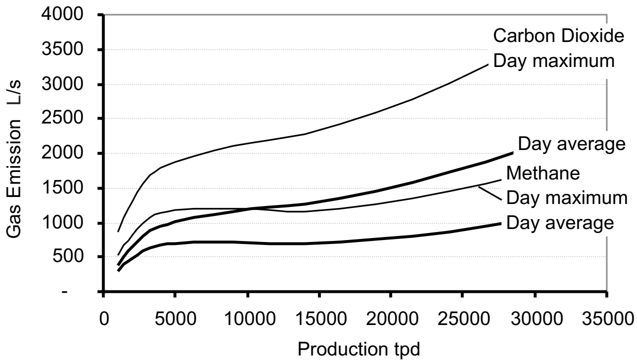

FIG 12 - Kayuga seam longwall gas emission.

As planned, goaf drainage holes are now installed in the longwall as it retreats from the high to low gas emission zones. As part of this process it has been necessary to change design of the end slider section to ensure that it is buried in the much smaller goaf pile. The longer Wynn seam slider design left end holes in the stone roof with a surprising degree of separation occurring between methane and carbon dioxide resulting in reduced effectiveness of carbon dioxide capture.

SPONTANEOUS COMBUSTION MANAGEMENT

The large amount of coal left in the goaf environment of Wynn seam longwalls, combined with the need to apply high-capacity ventilation circuits and goaf drainage, led to a challenging set of circumstances requiring intensive monitoring, continuous interpretation of data and the ability to make decisions quickly and at all times.

Intrinsic propensity for spontaneous combustion

High volatile B bituminous thermal coals of the Hunter Valley have a history of spontaneous combustion events in both longwall and bord and pillar operations. The intrinsic propensity of Dartbrook seams to spontaneous combustion was determined by laboratory tests and calculated properties (Table 2).

TABLE 2

Intrinsic propensity to spontaneous combustion for the Wittingham coal measures.

Target seams	R70°C/h†	Relative ignition temp °C	Heating rate IRR °C/hr	Temp rise TTR °C
Mt Arthur	4.48 - 5.81	129	2.91	5.89
Kayuga	3.16 - 5.16	132	1.17	3.00
Piercefield	-	134	1.9	6.00
Broonie	-	146	1.80	4.12
Wynn	2.20 - 3.61	139	1.83	5.89

† Source: Beamish (2005).

Dartbrook coal seams are classed as having a medium to high intrinsic propensity for spontaneous combustion. It is recognised that the propensity of a coal seam is also dependent on several extrinsic factors determined by mining method, mine geometry and other controls in place. In this respect seam geometry, ventilation capacity and gas drainage were likely to increase propensity, where as mine design (no bleeder airways and barrier pillar) together with seam gas emission would reduce propensity.

Laboratory tests for indicator gas emission with coal temperature showed that, at Dartbrook, carbon monoxide is the best tracer for low-level heatings because carbon dioxide and methane signature emissions would be masked by that of the seam gas.

Operational experience

During production of longwalls 1 to 6, the mine was faced with the task of applying action plan trigger levels to longwalls of, in Australian conditions, unprecedented seam gas and carbon monoxide emission rates. With respect to spontaneous combustion, carbon monoxide makes ranged from 200 to 250 L/min in some locations with hydrogen concentrations in the goaf up to 15 or 20 ppm under normal, low temperature (<50°C) conditions.

In initial longwalls, the mine was evacuated on a number of occasions when various trigger levels (absolute carbon monoxide concentrations >600 ppm, and rising Graham's ratio, >1.0, in particular) were exceeded.

With hindsight, it is possible that a number of these events did not warrant evacuation of the mine but, due to the dire consequences of underestimating the spontaneous combustion status of a goaf containing potentially explosive atmospheres, site management correctly erred on the side of caution. For example, it was during one of these evacuations that an advanced spontaneous combustion event, resulting from a concrete drop hole seal leaking into the goaf, was detected in longwall 3 (Figure 1).

Wynn seam longwall 7 incident

Longwall 7 commenced production on 10 January 2002, retreating through a fault zone 200 m from the start line as the goaf started to become inert. Production continued until 2 April 2002, when a significant fall of ground occurred at the tailgate. The mine took 20 days to recover the fall during which time the face was stationary and tailgate ventilation severely restricted.

Production commenced again, albeit slowly, on 24 April 2002 and continued until 12 May 2002 when signs of spontaneous combustion (rising carbon monoxide make and absolute concentrations of carbon monoxide and hydrogen) were observed following installation of the 16 c/t maingate seal. The longwall panel was sealed and the mine evacuated on 15 May 2002.

Following a period of inertisation with nitrogen and injection of fly ash, the face was recovered 20 days later on 4 June 2002 and production recommenced with continued inertisation. Trigger levels were again exceeded on the 2 July 2002 with indications of a very large medium temperature event taking place. The face was sealed and the mine evacuated for a second time.

The face was recovered again on 8 August 2002 after which the face line retreated to 14 c/t where it was removed and the block was abandoned.

Carbon monoxide make

Carbon monoxide makes with face advance for Wynn seam longwalls 1 to 7 are shown in Figure 13 together with that for the first Kayuga longwall.

In the Wynn seam, all goaves demonstrated a characteristic increase in make from 0 to 200 m after which values varied between 50 and 200 L/min on a daily average basis. In the Kayuga seam, the plateau is reached at a similar stage but absolute values are lower at 50 to 70 L/min.

In longwall 7, there was a small rise in carbon monoxide make when passing through the fault zone but conditions then returned to normal. The most significant increase in carbon monoxide make occurred prior to the tailgate fall during a period of slow, gas constrained production. Although all values were below evacuation trigger levels, it was apparent that the situation was deteriorating for which the best controls were continued production and control of the goaf atmosphere, ie installation of the 16 c/t seal.

It is important to note that, during this period, Graham's ratio remained below 1.0 and, although being actively sought, no trace of ethylene or visible signs were detected. The conclusion drawn at this time was, and with hindsight correctly, that a large event was developing but it was best managed by promoting an inert goaf atmosphere.

Hydrogen and ethylene

Hydrogen was observed in the goaves at detectable concentrations under normal, low temperature conditions. There is some academic debate about why this occurs at low temperatures, but potential sources are firstly, as seam gas resulting from prior igneous activity, secondly, as a result of acid-metal reactions (ACARP, 2001), and thirdly low temperature (<100°C) oxidation of coal (Nehemia, Davidi and Cohen, 1999).

Higher temperature emission points for hydrogen, due to advanced spontaneous combustion, are as follows:

1. medium temperature (100 to 300°C) oxidation of coal in aerobic conditions;
2. higher temperature (>250°C) distillation of coal; and
3. high combustion temperature (>500°C) coal when hydrogen emission may exceed that of carbon monoxide, as was the case in the longwall 3 concrete drop hole event.

Characterisation tests demonstrated that, although the ratio of hydrogen to carbon monoxide increases with temperature, in

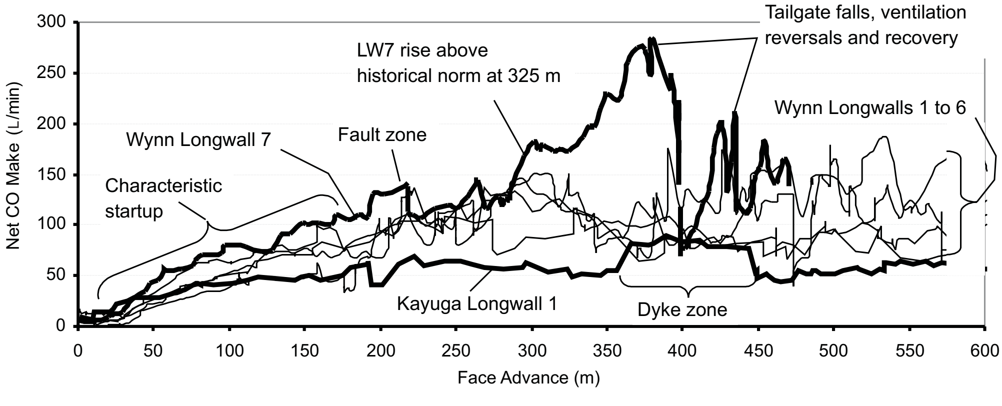

FIG 13 - Carbon monoxide make in Wynn and Kayuga seam longwalls.

aerobic conditions very much more carbon monoxide than hydrogen is produced and the H_2/CO ratio remains small (<0.01). In anaerobic conditions, less carbon monoxide is produced and the H_2/CO ratio is consequently higher, however remains below 1.0 up to about 350°C.

Ethylene is first formed in detectable concentrations at 120 to 150°C in aerobic conditions and above 300°C in anaerobic conditions. The presence of ethylene is therefore used as an indicator of an advancing event, but does not necessarily imply that high combustion temperature conditions exist.

Gas chromatograph and infrared detector data from monitoring tubes in longwall 7, for the period prior to evacuation to first recovery of the face, are summarised in Figure 14. The points of note leading up to the evacuation are as follows:

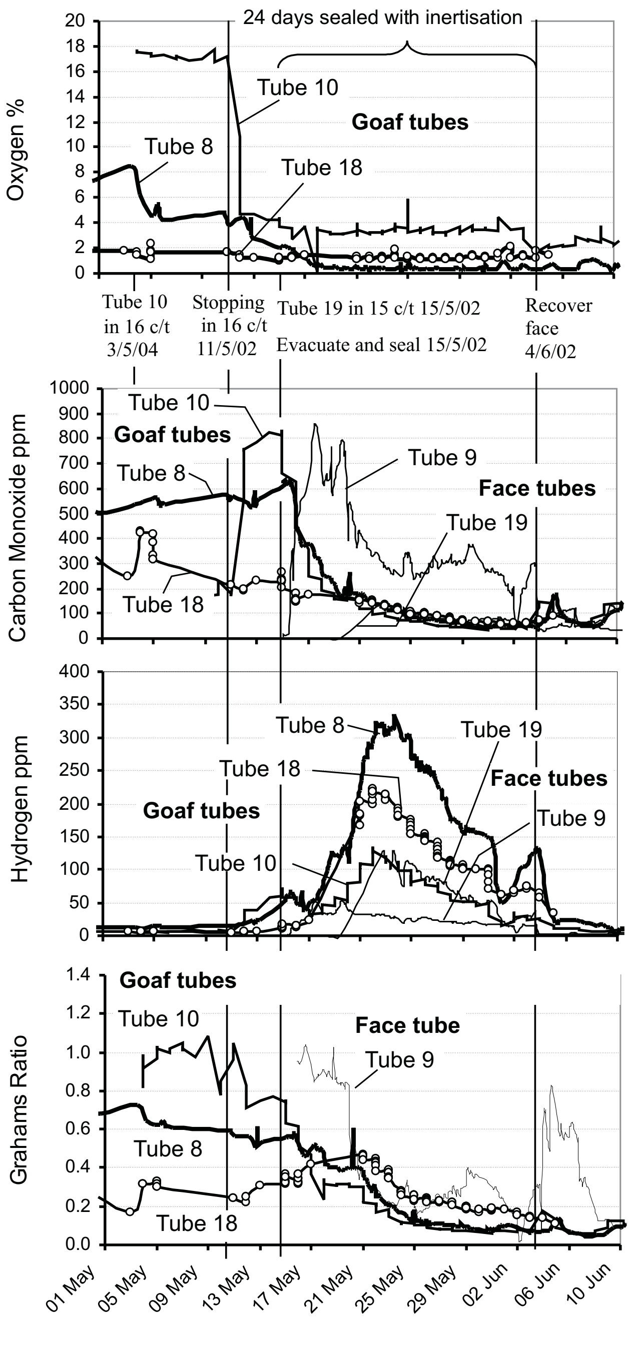

FIG 14 - Longwall 7 tube bundle data to recovery from first evacuation.

1. carbon monoxide make was increasing to 275 L/min prior to installing the 16 c/t seal;
2. oxygen concentrations fell to below four per cent after installation;
3. carbon monoxide concentrations in the goaf then rose to above 800 ppm at tube ten;
4. after evacuation, similar carbon monoxide concentrations were observed at the face line, tube 9; and
5. hydrogen concentrations reached 60 ppm at the time of evacuation.

The points of note during the evacuation are as follows:

1. after sealing the panel, oxygen concentrations remained low and carbon monoxide concentrations followed a characteristic decay curve;
2. hydrogen concentrations began to rise after sealing, reaching 300 ppm in the goaf and 100 ppm on the face;
3. during a period of increasing hydrogen concentrations, Graham's ratio on the face fell from 1.0 to between 0.2 and 0.4; and
4. ethylene was not detected at any time.

In isolation, hydrogen to carbon monoxide ratios could only be explained by combustion temperatures being reached in anaerobic conditions. However, the absence of ethylene in detectable concentrations indicated that the temperature of coal did not exceed 120 to 150°C. This conclusion was also consistent with observed Graham's ratio, low absolute carbon monoxide and hydrogen concentrations together with an absence of visible smoke.

The presence of hydrogen in concentrations of 100 to 300 ppm associated with falling carbon monoxide and oxygen concentrations together with H_2/CO ratios much larger than 1.0 had not been observed at Dartbrook prior to longwall 7. It was concluded that this phenomenon was due to differential mobility of gas within the goaf and influenced by a variety of physical processes. In these circumstances, it was not due to very hot coal (>150 to 200°C) being present in the goaf.

Thermography

Thermography had previously been used to monitor seals and pre-drainage hole collars for spontaneous combustion and is employed in a number of Australian mines for the detection of pillar events.

During the period of production, from second recovery to removal of the face from 14 c/t, eleven thermography surveys of the face line were undertaken. These provided indications of support temperatures ranging 30 to 50°C at the tailgate end to between 20 and 30°C at the maingate end.

Surface radon flux

The longwall 7 incident also provided an opportunity for CSIRO to undertake a field demonstration of their project concerned with locating spontaneous combustion events from surface by measuring changes in radon flux rates (Xue *et al*, 2003). The technique is based on the fact that the radon emanation rate of coal increases significantly from 60 to 100°C and, although the transport mechanisms are not yet fully understood, can manifest as a surface radon flux anomaly 300 m to 400 m from the seam. The demonstration involved measurement of surface flux rates at 15 m intervals on a 495 m × 210 m grid.

The main conclusions of the CSIRO report, that the event was initially extensive and in later stages hottest on the tailgate side approaching the face line, are consistent with those based on analysis of gas monitoring data. This indicated a large scale, low temperature (<120 to 150°C) event that propagated towards the face line as the deep goaf became progressively more inert.

Decision to abandon longwall 7

During the period of production, from second recovery to removal of the face at 14 c/t, carbon monoxide makes remained stable but high at circa 200 L/min, Graham's ratio remained between 0.4 and 0.8 but hydrogen make could be measured in the longwall return, at times reaching 40 L/min. This, together with other monitoring data, indicated the event remained active, was not responding to controls put in place and was advancing towards the face line.

At the same time, due to ineffective pre-drainage of the working section resulting from a reduction in localised permeability, it was not possible to manage seam gas emission onto the face at practicable production rates, ie the situation would have deteriorated if further production was attempted. For these reasons, the decision was made to abandon the block at 14 c/t.

The combination of factors leading to the longwall 7 incident were, to a large extent, a result of geological factors influencing pre-drainage efficiency and stability of the face line. In this case, management strategies employed mitigated the consequences, allowing the face to be sealed and recovered on two occasions followed by relocation to the next block. Without this level of management control, it is likely that the face equipment would have been lost entirely.

Kayuga seam – initial spontaneous combustion experience

Based on experience in the Wynn seam, a summary of actions and events undertaken to manage spontaneous combustion in Kayuga seam longwall 1 are is follows:

- Wynn seam management plan revised and implemented for Kayuga seam conditions;
- carbon monoxide and other management plan trigger levels reviewed to reflect differing norms;
- gas trends are being analysed to monitor changes that occur across geological structures;
- inertisation using a Floxal nitrogen plant has been employed to promote an inert goaf atmosphere on a proactive basis;
- all seals are inspected with thermographic cameras as a means of monitoring and recording seal integrity;
- bulk coal self-heating and additional R-70 tests have been conducted on both the Kayuga and Mt Arthur seams; and
- further radon flux tests are being considered.

CONCLUSIONS

Satisfactory management of spontaneous combustion, while at the same time managing high seam gas emission rates, at Dartbrook was only achieved by seam characterisation, data acquisition and analysis together with the implementation of appropriate management systems. In particular, management of longwall face, return and goaf atmospheres would not have been possible without the intense level of monitoring employed. These systems also enabled site-specific trigger levels, significantly different to industry 'norms', to be determined and applied with confidence.

With an increasing number of thick and/or multi seam projects being undertaken in Australian conditions, many with planned production rates twice that achieved in the Wynn seam, it is essential that a holistic approach to seam gas and spontaneous combustion management be taken. This is particularly the case in two heading longwall circuits in which the limiting volumetric capacity requires the application of more intense post drainage techniques.

Recognising the finite capacity of ventilation and post drainage methods, this suggests that alternative pre-drainage techniques from surface will be required to reduce the seam gas emission burden prior to mining. Surface to in-seam medium radius may prove more attractive in the future if coupled with some means of energy recovery or other gas utilisation strategies.

The detection of spontaneous combustion events, during early stages of development, in dynamic goaf environments of gassy mines, is challenging. With increasing block dimensions and production rates being employed in the Australian industry it will be necessary to consider new detection techniques to supplement more traditional ones. In addition, proactive, rather than reactive, inertisation is now considered an appropriate control at times of increased risk.

ACKNOWLEDGEMENT

Permission from Anglo Coal (Dartbrook Management) Pty Ltd and Anglo Coal (Australia) Pty Ltd to publish this paper is gratefully acknowledged.

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The implications of large scale tests for the detection and monitoring of spontaneous combustion in underground coal

by

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Abstract:

The large scale testing of the 15 tonne samples of Dartbrook coal indicates that it would be extremely difficult to detect the early stages of a spontaneous combustion event. Indeed it is most likely that detection will be delayed until the event has passed the point where it can be easily controlled. The emphasis thus must clearly be on prevention rather than detection and control.

Introduction:

This paper outlines key aspects of the final report of a three year research project testing the spontaneous combustibility of Australian coals under large scale laboratory conditions. The experimental details have been previously reported (Cliff, Davis, Bennett, Galvin and Clarkson, 1998 and 1999), and will only be briefly detailed here.

Typically laboratory assessment of the spontaneous combustibility of a coal is restricted to small scale (approximately 70 g) testing, whether it be R_{70}, Self Heating Temperature, Crossing Point Temperature or gas evolution rates (Cliff, and Bofinger (1999).). The validity of extrapolating these tests to full scale mines has not been established. In an effort to link small scale tests to the real world SIMTARS constructed a large scale reactor.

A number of large scale heating tests using approximately 15 tonnes of coal have been carried out at SIMTARS . The coal was arranged in a pile 2m wide, 2m high and 4m long in a reactor that could be sealed. Air was passed through the pile in an attempt to produce a spontaneous heating.

Approximately 15 tonnes of either crushed or run of mine coal was placed between the two block walls. Five layers each of 50 thermocouples were used to monitor the temperature throughout the pile and fifteen gas sampling tubes were inserted into and across the central axes of the reactor. Air was passed from one end to the other. Internal air flow was varied over the range of approximately 50 to 180 litres/minute.

The apparatus included a thermal cover in an effort to reduce the heat losses and facilitate the self heating of the coal and a series of heaters in the area between the reactor and the cover to preheat the air to approximately the edge temperature of the coal (Figure 1).

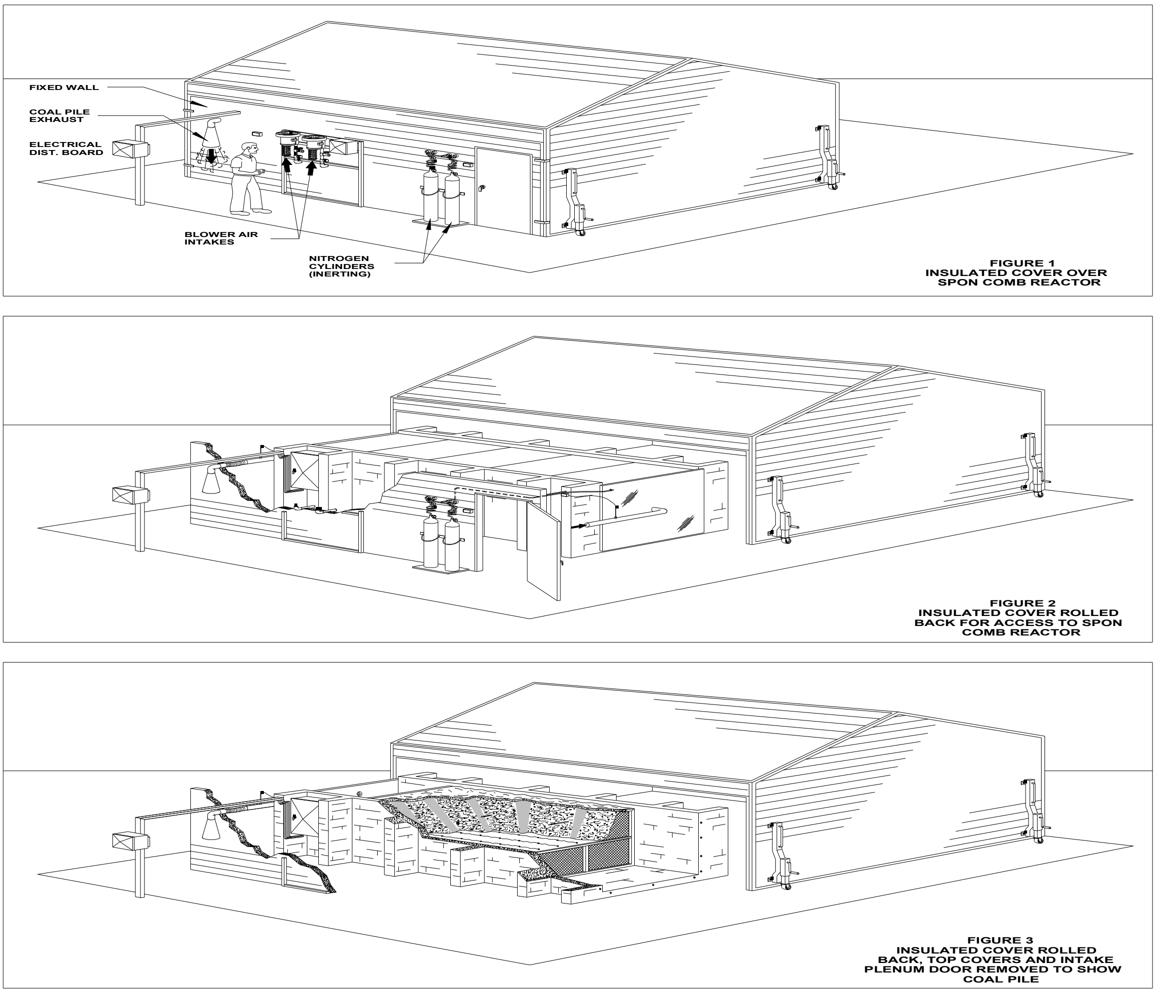


Figure 1. Large Scale test apparatus

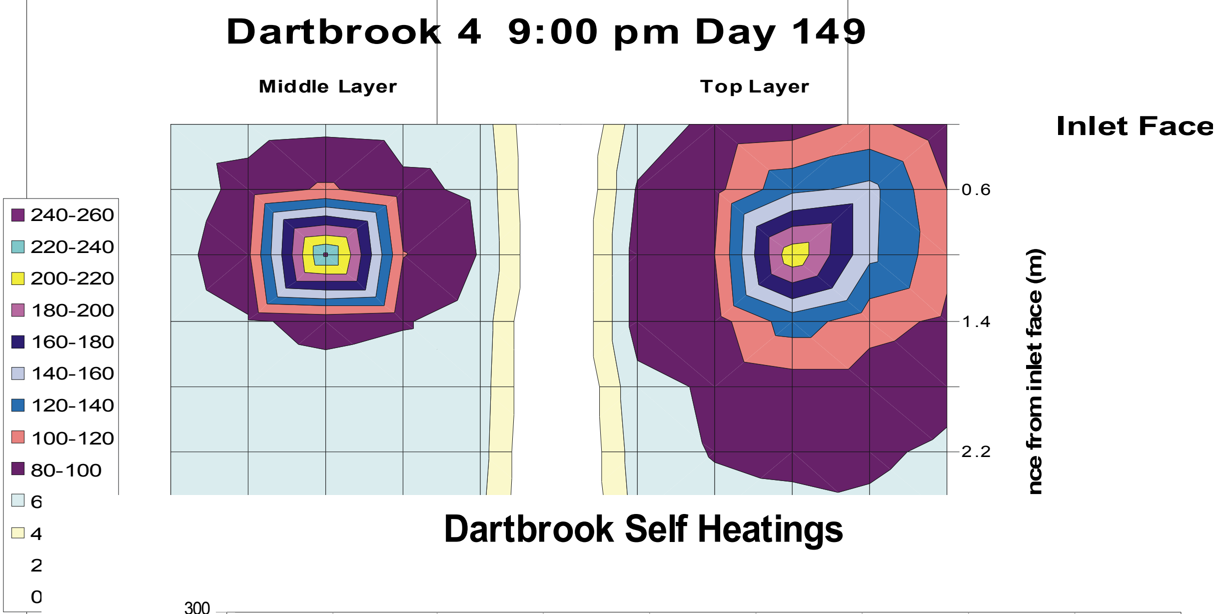
Results And Discussion:

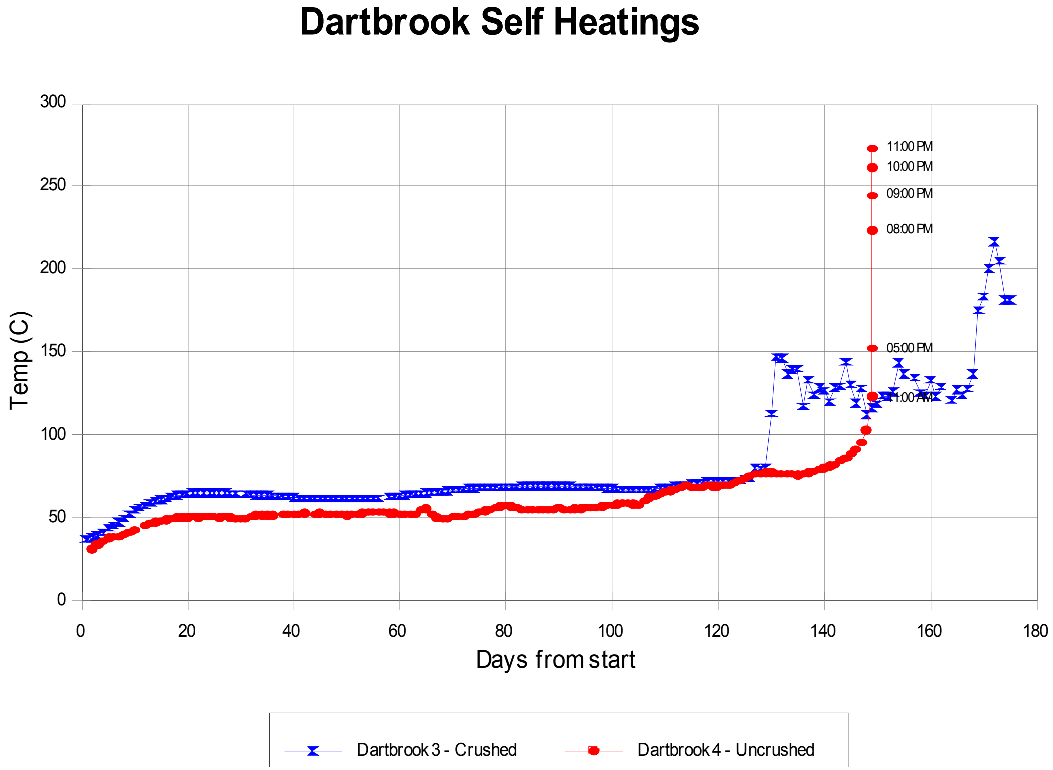
Self heating was obtained for two samples of Dartbrook coal, one crushed and one run of mine. They exhibited a number of differences in behaviour and a number of similarities.

The crushed sample of coal packed much more tightly into the reactor and air flow through the reactor was restricted, with a pressure difference across the reactor of up to 100 pa. In contrast the run of mine sample had no detectable pressure difference across it. This meant that the residence time of air inside the crushed coal was much greater than for the run of mine sample, facilitating secondary reactions and reabsorption of gases into the coal.

Both samples were observed to self heat once the coal had dewatered, indeed it was concluded that the process of removing the water from the coal was the rate limiting step in the self heating process.

The key observations from the self heating experiments are summarised in the table below.

<table>
<tr><td>Observation</td><td>Crushed sample</td><td>Run of Mine sample</td></tr>
<tr><td colspan="3">




</td></tr>
<tr><td>Exponential Temperature rise</td><td>No - stepped increase initially</td><td>Yes</td></tr>
</table>

Time taken to initiate self heating	172 days to thermal runaway	149 days to thermal runaway
Comparison with small scale tests	Good at Hot spot deviation further away	Excellent correlation
Odour	Consistent with small scale tests - not fire stink	Consistent with small scale tests - not fire stink
CO Make	Est 0.5 l/min at 120 °C Est 1.0 l/min at 200 °C	1.2 l/min at 120 °C 2.5 l/min at 250 °C
Size of hot spot	Approx 400 mm radius	Approx 400 mm radius
Location of hot spot	Middle layer, at inlet face	Middle layer 1 m from inlet

Table 1. Summary of experimental results

The temperature time profiles for both experiments are shown in the figure below.

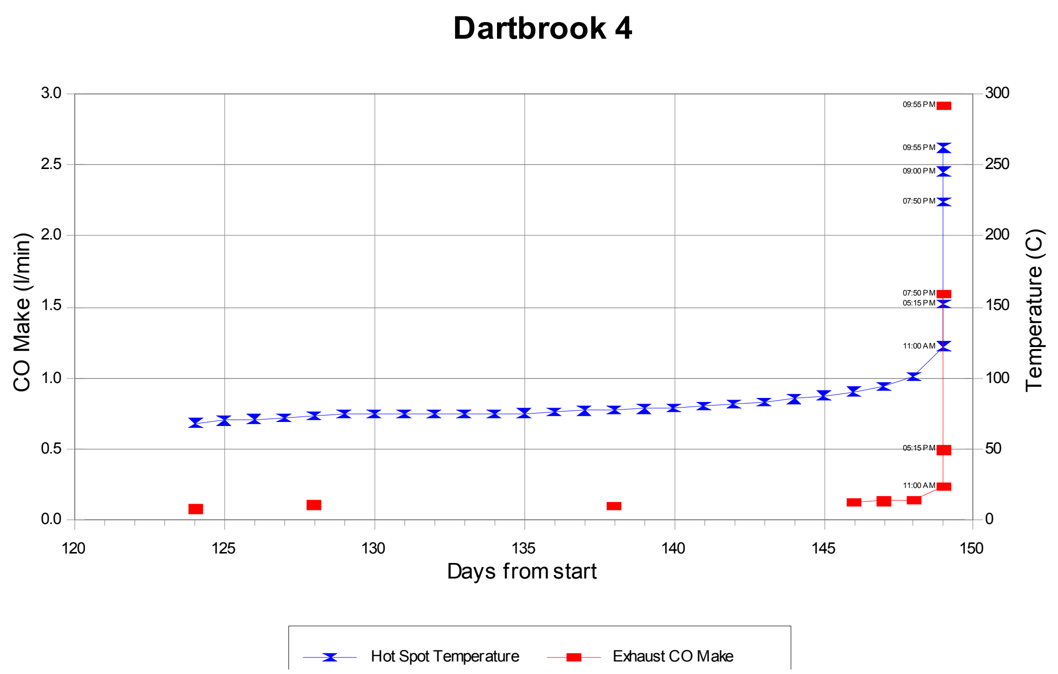
Figure 2. Hot spot temperature vs time plots

The hot spot size is illustrated by the figure below taken from the final Dartbrook uncrushed sample test.

Figure 3. Temperature distance plots for hot spot

The exit CO make as a function of time for the last experiment is shown in Figure 4 below.

Figure 4. Temperature and CO profiles for the hot spot.



Conclusions:

Spontaneous combustion is essentially a simple process. Coal reacts with oxygen to produce water vapour, carbon dioxide and carbon monoxide and heat. If the heat is retained by the coal then it gets hotter and reacts faster , ultimately reaching open flame - a self heating. If the heat produced is completely removed then no self heating occurs. In the case of the above experiments the heat loss process, although including losses due to radiation, convection and conduction, was dominated by evaporation of moisture in the coal until all moisture in the coal was driven off. Once this heat sink was removed self heating began. Indeed the moisture content of the coal determined the "incubation" period for the sample, rather than the inherent reactivity of the coal.

If the data from the large scale tests is extrapolated to goafs and pillars, then it is clear that heatings can develop and worsen in confined areas involving a few tonnes of coal. This heating will be difficult to identify remotely due to dilution effects. If monitoring can be carried out on top of a heating then the data consistent with that obtained from the small scale laboratory tests will be found. The further away from the heating monitoring is carried out and the more dilute or mixed with other gas streams the products of oxidation become, the less reliable is the ability to identify the heating as illustrated below in figure 5 below.

Monitoring in roadways with 10's of m^3 per sec of air flowing in them will make detection very difficult until the heating has reached a very advanced stage. However in this situation, as the heating will be near the surface, heat detection equipment, such as infrared cameras, and thermography may be effective.

Using standard CO make values is meaningless. As soon as the CO make significantly exceeds the background level investigation should begin, not at 10 or 15 or 20 l/min. CO Make is still a better indicator than CO concentration as it
removes the vagaries of dilution.

Graham's ratio is unlikely to be of any practical value in roadways as the heating products will be swamped with fresh air until it is a major heating, thus giving no meaningful oxygen deficiency.

It has been shown before that the detection of significant concentrations of hydrogen or higher hydrocarbons is consistent with a high temperature heating (Cliff and Bofinger, 1999).

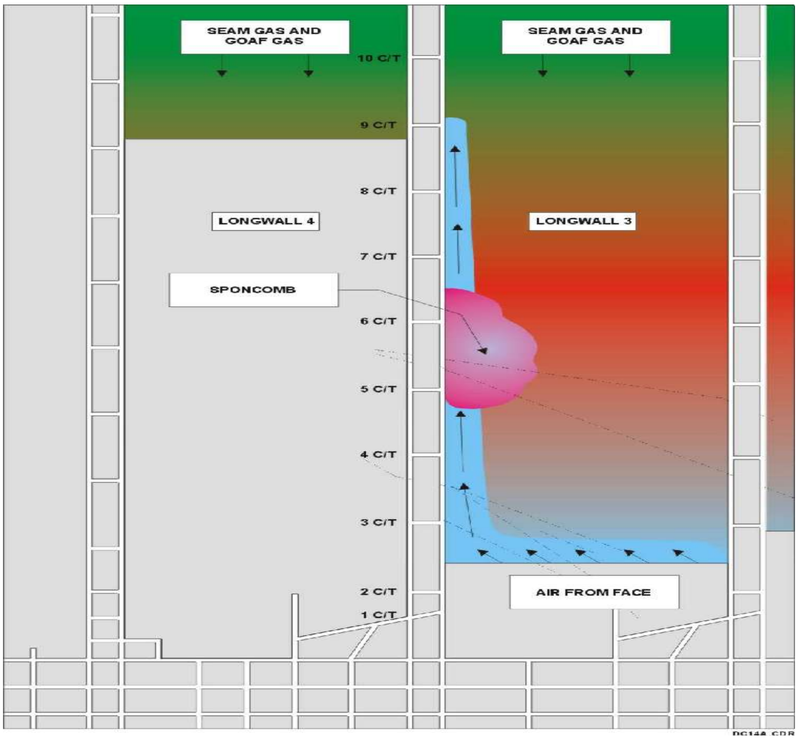
Smell may be of use as human beings are sensitive to very low concentrations of odiferous gases, however care must be taken to identify the smell and allow for the difference in sensitivity between individuals and the subjective description of the smells. The future use of electronic noses holds promise but much work needs to be done in making them routine analysis devices and improving their sensitivity and ability to discriminate between similar smells. Such devices would have to be calibrated with individual coals as the cocktail of hydrocarbon fragments evolved from the coals varies greatly from one coal to the next, depending on the sulphur and oxygen concentration.

Goaf monitoring needs to be carried out as comprehensively as time and resources allow so that all potential heating sites are monitored as close to them as possible. Heatings can only occur where there is sufficient oxygen, the coal is dry and the heat balance is in favour of heat retention.

This last statement is amply validated by the history of heatings at Dartbrook, North Goonyella and Moura No.2 mines.

Other events with high concentrations of CO have demonstrated that high CO can be obtained without any real heating occurring if there is an absence of ventilation - eg Dartbrook and Crinum.

Figure 5. Remote detection of a goaf heating



Recommendations

1. Detection future

- Effort should be put into making electronic noses more robust and discriminatory.

- For pillar heatings infrared cameras and thermography may be more sensitive than gas detection.

- To minimise difficulties in sampling - care should be taken to allow for remoteness of sampling and the inherent limits to reliability and detectability

- Analysis must allow for contributions from seam gases, goaf gases etc.

- The best indicators of spontaneous combustion are those that are independent of the air flow i.e. CO make and Graham's ratio, but even these have grave limitations and we should be aware of what they can and cannot do. We need to specify accuracy limits

- We need to recognise the common fallacy about CO make and not rely on text book values, which as described above, would trigger a response corresponding to a large scale heating, often too late to treat.

- Where airflow is limited gas evolution may be valuable, but there is substantial variation between coals and we need to know individual coal behaviour - for CO, H2 and CO2 particularly.

2. Education

- The workforce needs to be educated to look for the signs of spontaneous combustion and the potential for spontaneous combustion, not just gases, but the other indicators and the potential for unwanted and unintended airflows.

- Much more attention needs to be placed on sampling reliability and representation.

3. Prevention

Most importantly, prevention is far more important than cure so more emphasis needs to be placed on:

- The use of inertisation technology to remove oxygen and thus the potential for spontaneous combustion of goafs should be encouraged. This requires suitable monitoring protocols to be developed to ensure that the inertisation is effective, and remains effective.

- Where pillar heatings are possible, the use of sealants to eliminate oxygen coal interaction should be undertaken.

- Ventilation must be better controlled to minimise leakage. This requires good seals and proper ventilation monitoring, both pressure quantity surveys on a regular basis and inspections of seals. The use of continuous velocity or pressure sensors is desirable. This would enable abnormal airflows to be identified before they are a problem.

- Mine design should include the evaluation of pressure differences and leakage to identify the potential hotspots for spontaneous combustion.

4. Incubation period

- The concept of incubation period should be completely disregarded as these large scale tests have shown that reactivity can lie dormant for long periods only to be activated after the coal dries and finds oxygen. This is also consistent with recent experience at Laleham No.1 colliery.

Future research

- Further research should be carried out in the large reactor to establish whether or not it is a capable of determining the relative propensity of a coal to spontaneously combust.

- Research in the reactor should be extended to study the effects of the different types of inertisation, including sealants, water and self inertisation, on spontaneous combustion, thus optimising the ability to control heatings. This research could should also look at the ability of coal to retain heat, and the time taken for the coal to become unreactive rather than just starved of oxygen.

- Gas chromatographs should be developed with increased detection sensitivity for hydrogen, and the higher hydrocarbons. Rapid analysis has been shown to dramatically improve the quality and quantity of data available to allow interpretation of spontaneous combustion events and treatments.

- The role of water in the chemistry of spontaneous combustion merits more study.

- Research in small scale studies does not allow for the complexities of the real world such as heat loss, and water content. It is only good for inherent reactivity measurement, gas evolution behaviour.

- Electronic nose technology, including neural networks and multifactorial analysis, should be utilised in monitoring systems, once it has demonstrated sensitivity, discrimination and robustness.

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ACARP

Australian Coal Association Research Program

Report

INERTISATION AND MINE FIRE SIMULATION USING COMPUTER SOFTWARE

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ABSTRACT

Inertisation is a technique that has been used around the world to enhance the safety of underground coal mine areas either to avoid the potential for a combustion event or to stabilise a situation after an ignition, fire or heating. The term normally refers to the fact that the atmosphere in the area is such that it cannot sustain combustion, including ignitions, and is therefore "inert".

The primary objective of the study is to review coal mine inertisation in Australia. In particular it is to focus on use of the Polish mine fire simulation software "VENTGRAPH" to gain better understanding of how inertisation (GAG, Mineshield, Nitrogen Pressure Swing Adsorption (Floxal) and Tomlinson Boiler) units interact with the complex ventilation behaviour underground during a substantial fire. Most emphasis has been given to understanding behaviour of the GAG unit because of its high capacity output. Critical aspects targeted for examination under the project grant include location of the unit for high priority fire positions, size of borehole or pipe range required, time required for inertisation output to interact with and extinguish a fire, effects of seam gases on fire behaviour with inertisation present and main fan management. The project aims to increase understanding of behaviour of mine fires in modern mine ventilation networks with the addition of inert gas streams.

A second major aim of the project has been to take findings from the simulation exercises tied to the above objectives to develop inertisation related modifications to the program in conjunction with the Polish program authors.

Computer simulation of mine fires and effects on ventilation networks has been introduced in recent years to the industry with considerable interest and success. This has already put a significant number of mines in an improved position in their understandings of mine fires and the use of modern advances to preplan for mine fires and the handling of possible emergency incidents. Mine exercises have been built around the use of the fire simulation computer program "VENTGRAPH" and modelling of fire scenarios across the mine layouts. A coding system has been developed to assist interpretation within the audit exercises.

Simulation software has the great advantage that underground mine fire scenarios can be analysed and visualised. The software provides a dynamic representation of a fire's progress in real time and utilises a colour-graphic visualisation of the spread of combustion products, O_2 and temperature throughout the ventilation system. During the simulation session the user can interact with the ventilation system (e.g. hang brattice or check curtains, breach stoppings, introduce inert gases and change fan characteristics). These changes can be simulated quickly allowing for the testing of various fire control and suppression strategies.

Inertisation has been accepted to have an important place in Australian mining emergency preparedness. The two jet engine exhaust GAG units purchased from Poland by the

Queensland government in the late 1990s for the Queensland Mines Rescue Service have been tested and developed and mines made ready for their use in emergency and training exercises. Their use in real and trial mine fire incidents has underlined the need for more information on their application.

The NSW Mineshield (liquefied nitrogen) apparatus dates to the 1980s and has been actively used a number of times particular in goaf heating incidents. The Tomlinson (diesel exhaust) boiler has been purchased by a number of mines and is regularly used as a routine production tool to reduce the time in which a newly sealed goaf has an atmosphere "within the explosive range" and for goaf spontaneous combustion heatings. Nitrogen Pressure Swing Adsorption (Floxal) units are available and in use both for reducing time in which goafs are "within the explosive range" and for goaf spontaneous combustion heatings. Each of these facilities puts out very different flow rates of inert gases. Each is broadly designed for a different application although there is some overlap in potential usages.

Case studies have been developed to examine usage of the GAG inertisation unit. One section examined seam gas emissions in the face area; addition of the inert gas stream adds another level of complexity to the already complicated interrelationships between the mine ventilation system, the presence of seam gases and a mine fire. Another section has focused on selection of the surface portal location for placement of the GAG for effective fire suppression. The difficulties that some current approaches present are highlighted.

Priority fire locations at a wide selection of mines with a developed and current Ventgraph simulation model have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation). The conclusion is that the current situation is not well placed to effectively inert most colliery priority fires.

These simulation exercises undertaken with a wide range of Australian mines focused attention to the situation that many potential underground mine fire sources cannot be successfully inertised with the GAG docked at the current specified point. This inability to deliver GAG output is particularly so for fires in extended areas of workings or in panels. Two important conclusions are

- Successful delivery of GAG output from units on the surface must consider other (that is alternative to Mains Travel or Conveyor Heading portals) delivery conduits directly into workings near the fire through existing or purpose drilled boreholes.
- During a fire the stopping of the main surface fan or fans will lead to rebalancing of pit ventilation and in some cases potential explosions through air reversals bringing poorly diluted explosible seam gases or fire products across the fire site.

Another section has looked at inertisation and dilution issues in Mains headings. These

present a complex ventilation network and with additional interference from a fire, maintaining control of the movement of inert gas is more difficult than elsewhere in the mine. Even good quality segregation stoppings allow significant dilution of inertisation flows over relatively short distances. There is a section that has examined considerations presented by "punch" mines layouts. A number of recent punch longwall mines are accessed off highwalls. These mines have some provision for GAG docking from within the highwall pit but all have put down boreholes to workings which enable the GAG team to operate the engine from the surface.

A calibration exercise on the VENTGRAPH software has occurred in two parts. Back analysis of the gas monitoring data during a fire at the US Pattiki Mine showed that a VENTGRAPH model could be established to simulate satisfactorily this incident. The inertisation exercise during part sealing of the Newlands South highlighted a number of findings. The GAG quantity measured exhausting from the mine area being sealed was at first considered to be unrealistically low. However further analysis, as detailed in Chapter 10 of this report, indicates that accounting for temperature and moisture mass changes explains any differences. The hypothesis that some of the GAG exhaust, with diurnal pressure changes, will flow into and out of goafs is of interest and needs to be accounted for. Further monitoring of mine site GAG exercises are warranted to give greater understanding to this complex system.

A brief overview of the VENTGRAPH simulation software is given. It has highlighted the new features that have been added to the software as a consequence of this inertisation project and in particular the ability to use up to four different types of inertisation gases (at varying flow rates) across a mine layout simultaneously and the ability to include carbon dioxide and nitrogen seam gases as well as methane.

Exercises based on Oaky North and Oaky No 1 mines have involved "evaluation or auditing" of ability to deliver inert gases generated from GAG units to high priority underground fire locations. These exercises have been built around modelling of fire scenarios across the mine layouts. A coding system, A to E, has been developed to assist interpretation within the audit exercises. The principal sections focused on the development of scenarios for examining priority 15 fire locations across both mines and firstly their effect on the mine ventilation system and secondly the influence of introducing inertisation gases to stabilise the fire. Inertisation outcomes in all case scenarios have been examined through introduction through the mine's present docking point. Each scenario has then been re-examined one or more times to establish if a different docking point, altered underground ventilation segregation or other approach would be more effective in stabilising the simulated fire.

Five major case study scenarios based on the modelling of fires with introduced inertisation in a number of high priority different points geographically spread within the Oaky North longwall mine layout have been discussed. Possible alternative strategies for successfully

inerting the fires have been examined and conclusions drawn to the success or otherwise of these approaches. Approaches focus on use of alternative portal docking points, increased underground segregation and possible use of boreholes to delivery GAG exhaust directly to the fire seat.

These fire simulation exercises have shown that some priority fires at Oaky North and Oaky No 1 mines can be stabilised through GAG inertisation strategies. One scenario goaf fire strategy developed is a case in point where use of a panel borehole with careful segregation allowed a relatively fast outcome to be achieved. Another scenario development heading fire was similar in that a borehole GAG delivery gave the best outcome. Both these were achieved with one surface fan operating and maintaining minimum pit ventilation and seam methane dilution. A third scenario fire, a Mains belt fire, utilised the GAG positively through use of an alternative Portal for docking. These examples showed that the audit was a success in that it highlighted successful approaches to use of inertisation where the previous approach was inadequate.

On the other hand Mains belt and Development heading belt scenario fires were placed such that alternative approaches to inertisation were ineffective because pit layout means excess dilution affects the GAG exhaust quality which can be brought to the fire.

Recommendations arising from the Oaky North and Oaky No 1 mines exercises were as follows:

1. GAG docking stations should be fabricated for all ventilation intake openings to both mines. The existing facilities should be supplemented by docking points at all Highwall or Drift portals, any pit boreholes of appropriate diameter and future main shafts. In effect each docking point can deliver to a restricted geographic zone within the pit; multiple points allow the appropriate point to be utilised.
2. Segregation strategies have shown that distribution of inert gases to separate Mains headings can be improved. Current segregation is less effective for fires located a long way inbye the mine and in the longwall production and development panels (due to increasing dilution through stoppings).
3. Borehole with a diameter of at least 1 m should be considered at the beginning of each panel for delivering inert gases to each longwall production or development face. These boreholes can also be used for other purposes such as delivery of ballast or emergency extraction of people out of the mine. They may be used for other services. Incorporation of remote controlled doors should be considered to give control over which gateroad should be used to carry the inert gases into the panel.
4. Scenarios in which no satisfactory inertisation strategy was apparent should be further examined to determine the merits of locating a borehole or shaft in the vicinity of the fire to enable satisfactory outcomes.

The fire simulation exercises at Oaky North and Oaky No 1 mines demonstrated that it is possible to efficiently evaluate possible inertisation strategies appropriate to a complex mine layout extracting a gassy seam and determine which approach strategy (if any) can be used to stabilise a mine in a timely fashion.

A final chapter has focused on borehole design parameters. Analyses have been established applicable to Australian conditions based on the complex fluid flow theory that describes the dynamic, hot, pressurised exhaust carrying a superheated vapour. Determinations have been made of the relationships between borehole back pressure and GAG thrust relationships and the best approach to vary the jet engine thrust to overcome this bask pressure. These mathematical relationships can now be applied to investigate the possibility of using GAG in small diameter boreholes for either production inertisation or fire fighting purposes. This would be a verification exercise taking the equations describing GAG exhaust fluid behaviour based on the steady flow energy equation and comparing the theoretical predictions of GAG exhaust fluid behaviour with actual measurements of pressure, quantity and temperature at various locations downstream from GAG exhaust trials proposed.

To support the report's main findings some concluding discussions on borehole delivery of inert gases and aspects of Mains segregation have been included. Some considerations for selecting the best surface portal location placement for the inertisation unit for most efficient suppression of a fire have been examined. There is a brief examination of the possibility of a wider and proactive application of GAG in Australian mines responding to or recovering from mine fires or spontaneous combustion heatings or elimination of the potential explosibility of newly sealed goafs is examined. The primary focus here is on systems involving delivery of GAG exhaust through docking to surface boreholes connecting into underground workings. Attainable designs for panel boreholes and how GAG docking to boreholes can improve delivery of GAG exhaust are discussed. Introduction of inert gases can present difficult emergency management decision making. Should the main mine fans be turned off to reduce dilution of the inert gas, or will this action cause, in conjunction with fire induced buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

The report has recommended some addition studies that may be undertaken based on the findings from this project. It is proposed that a study on production or proactive use of inertisation and particularly the GAG inertisation unit should be undertaken. The study should aim to examine the possibility of a wider and proactive application of GAG in Australian mines responding to or recovering from mine fires or inertisation of sealed mine workings or spontaneous combustion heatings or elimination of the potential explosibility of newly sealed goafs.

In conclusion the main conclusions from this project are:

- Positioning of the GAG inertisation units is a major determinant of potential success for most efficient suppression of a specific fire. Studies undertaken with most Australian underground collieries have concluded that the current situation is not well placed to effectively inert most colliery priority fires.

- There is a need to examine attainable designs for GAG inerting using panel boreholes under Australian conditions with current drilling technology. Part of this is to calculate design considerations to overcome back pressure. There is a limit to the ability of the GAG jet engine to deliver exhaust down smaller dimension borehole. The objective will be to define the
    - Hole designs (diameters and depths) that can deliver directly without assistance of any fan,
    - Hole designs that can deliver with modifications to the jet engine to improve thrust to overcome back pressure required for this delivery to be attained, and
    - Specifications of boreholes design parameters that cannot achieve delivery even with full GAG jet thrust.
- There is a need to examine the use of the GAG for production or proactive uses in a wider application in Australian mines responding to recovering from mine fires, spontaneous combustion heatings, elimination of the potential explosibility of newly sealed goafs or inert mines or mine sections on closure. Some of the current uses of low flow inertisation facilities could be more effectively undertaken with the GAG unit.

Mine fires and heatings are recognised across the world as a major hazard issue. New approaches allowing improvement in understanding their use of inertisation techniques have been examined. The outcome of the project is that the mining industry is in an improved position in their understanding of mine fires, use of inertisation and the use of modern advances to preplan for the handling of possible emergency incidents.

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TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

1. INTRODUCTION

Inertisation is a technique that has been used around the world to enhance the safety of underground coal mine areas either to avoid the potential for a combustion event or to stabilise a situation after an ignition, fire or heating. The term normally refers to the fact that the atmosphere in the area is such that it cannot sustain combustion, including ignitions, and is therefore "inert".

This can be accomplished by reducing the oxygen component of the atmosphere to a level that will not sustain combustion of a gas or of a solid, such as coal, or by increasing the amount of an existing flammable gas, such as methane, to an atmospheric concentration above which it becomes non-flammable relative to the oxygen level of the atmosphere. The oxygen level can be lowered to levels below that of normal air through consumption by slow or fast oxidation processes or by the addition of inert gases such as nitrogen or carbon dioxide which do not participate in the oxidation or combustion processes. The technique enhances safety to the extent that should an ignition source be present in an inert atmosphere, combustion would not occur. Additionally, as will be discussed here, creating such an inert atmosphere in an area of a coal mine where combustion is ongoing, can extinguish the combustion process.

Use of inertisation techniques is very common in coal mining regions around the world with the use of bleederless ventilation systems in active areas of coal mines. Bleederless ventilation is generally used where the prevention of spontaneous combustion is a key parameter for the ventilation design of an active panel. Bleederless ventilation is an attempt to render the goaf inert in that it permits the accumulation of methane and other non-flammable gases, while limiting the introduction of oxygen. This creates an inert atmosphere that will not sustain the self combustion of coal and, therefore, limits the potential for these types of hazardous fire events or as a potential ignition source for an explosion.

Seals are ventilation structures that are designed to prohibit, or at least greatly minimise, the exchange of atmosphere between an abandoned area with any adjacent ventilated areas and are therefore often a key component of inertisation methods. Well constructed seals are also designed to limit the potential that an explosion in a sealed area could impact the active mine areas or the safety of the mine workforce. Seals, Even those seals that are virtually airtight, do not ensure that the entire atmosphere in the sealed area is inert. Research and investigations have shown that the surrounding strata can permit an exchange of atmospheres between the sealed area and adjacent ventilated areas of a mine (Garcia, et al. 1995). Recent case study modelling work by Gale (2005) has likewise shown that strata interaction to mining can produce hydraulic conductivity changes in the strata and provides insight to the mechanism for these mine atmosphere exchanges. The sealed area most susceptible to not being inert is the periphery of areas adjacent to ventilated areas. The concern from the inertisation standpoint is mainly that oxygen from the ventilated area will enter the sealed area and make the atmosphere flammable or capable of sustaining combustion, although the reverse flow of methane or oxygen depleted air from the sealed area can also be a safety concern.

The primary objective of the study is to review coal mine inertisation in Australia. In particular it is to focus on use of the Polish mine fire simulation software “VENTGRAPH” to gain better understanding of how inertisation (GAG, Mineshield, Nitrogen Pressure Swing Adsorption (Floxal) and Tomlinson Boiler) units interact with the complex ventilation behaviour underground during a substantial fire. Most emphasis has been given to understanding behaviour of the GAG unit because of its high capacity output. Critical aspects targeted for examination under the project grant include location of the unit for high priority fire positions, size of borehole or pipe range required, time required for inertisation output to interact with and extinguish a fire, effects of seam gas on fire behaviour with inertisation present and main fan management.

A second major aim of the project has been to take findings from the exercises tied to the above objectives to develop inertisation related modifications to the program in conjunction with the Polish program authors.

Inertisation systems for handling underground fires, use in sealing old mines or mine sections, spontaneous combustion heatings and elimination of the potential explosibility of newly sealed goafs have been accepted as important safety approaches within the Australian industry. Computer simulation of mine fires and effects on ventilation networks has been introduced to the industry over the last few years and particularly under ACARP grant 12026 . This has already put about 20 Australian underground coal mines in an improved position in their understanding of mine fires and the use of modern advances to preplan for mine fires and the handling of possible emergency incidents. The fire program VENTGRAPH allows simulation of the introduction of an inertisation gas stream to the ventilation network and understanding of its effect in fire suppression. This study has investigated application of the program in relation to the utilization of available inertisation units. The interaction of inertisation with a mine's ventilation system during an underground fire requires further investigation and the program simulator has capability to assist mining personnel to understand the critical issues.

The theory of fire behaviour and fire control in the underground mine environment is complex. Application of the simulation software package to the changing mine layouts requires experience to achieve realistic outcomes. A comprehensive research project into mine fires study applying the Polish derived VENTGRAPH mine fire simulation software, preplanning of escape scenarios and general interaction with rescue responses was undertaken in 2003 and 2004 following the awarding of an ACARP grant entitled “Mine Fire Simulation in Australian Mines using Computer Software”. The approach has been introduced to the majority of Australian mines in New South Wales and Queensland mines through on site development of fire scenarios, escape strategies and recovery planning. Initial work under this grant involved acquiring the program and development of a bridging sub-program to allow conversion of VENTSIM mine network lay-outs to the mine layout required in VENTGRAPH. Other preparatory work included reviews of fire thermodynamics, escape approaches and discussions with mine rescue management on applications of the program. Two Polish or Polish/American people with experience in the development and use of the VENTGRAPH program, Dr Waclaw Dziurzynski of the Polish Academy of Sciences, Krakow, Poland and Dr Andrzej Wala of the University of Kentucky, USA have visited Australia during the project and given support. Recommendations have been made to the Polish software authors on improvements to the program to make it easier to set up mine simulations, mine model editing, tracing of other gases and related issues. Many or these were adopted and this cooperation was greatly appreciated.

To ensure credibility the work program then turned to implementation of the approach to mines across Queensland and NSW and on-site use of the program at individual mine sites. Inspectorates in both states have been very supportive of use of the simulation approach to improve understanding in this very important area. The Queensland Mine Rescue Service purchased the VENTGRAPH program and have had training undertaken with all their permanent managerial staff. Furthermore the researchers have been asked by both the Inspectorate and a number of operating mines to assist in the design and implementation of Level 1, 2, 3 and 4 emergency exercises. These exercises have allowed a large number of people to become familiar with some of the capabilities of this approach to fire simulation. Invitations were received to speak on the results of the project to groups such as the Queensland Chief Inspector's CEO's meeting, Regional Inspectors' meetings, the Mine Managers Association of Australia, the annual GAG Inertisation seminar in Queensland and a number of professional institution conferences in Australia and overseas. A number of refereed papers have been published in international journals and conference proceedings.

Simulation software has the great advantage that underground mine fire scenarios can be analysed and visualised. The software provides a dynamic representation of a fire's progress in real time and utilizes a colour-graphic visualization of the spread of combustion products, O_2 and temperature throughout the ventilation system. During the simulation session the user can interact with the ventilation system (e.g. hang brattice or check curtains, breach stoppings, introduce inert gases and change fan characteristics). These changes can be simulated quickly allowing for the testing of various fire control and suppression strategies.

A number of the mine site fire scenario exercises undertaken have addressed the issue of mine recovery. Simulated introduction of the GAG or other inertisation apparatus has indicated that there is a substantial lack of knowledge on the interaction of these facilities with the mine ventilation system. This question formed the principal basis for this inertisation project.

The Queensland GAG unit was purchased in the late 1990s following a recommendation from the Moura Number 2 mine disaster. It was first used actively in 1999 at the Blair Athol mine to handle a spontaneous combustion issue in old underground working that were about to be mined by surface extraction. The Queensland GAG unit was subsequently used successful in an underground mine fire incident in the Loveridge mine, West Virginia in early 2003. On this occasion the GAG ran for approximately 240 hours over 13 days and was successful in stabilising the mine so that rescue teams could enter the mine and seal and fully extinguish the fire-affected zone. Much was learnt about the ventilation network behaviour and the need to have an upcast shaft open. Observations were made on the effects of natural ventilation pressure, barometric changes and rock falls on the backpressure experienced by the operating GAG.

A fire at the Pinnacle mine, also in West Virginia in October 2003 attempted to use a Polish owned GAG unit without success. Following these experiences the US Micon company has purchased a GAG unit and is developing a commercial mine emergency and recovery business. A fire in the Dotiki mine, Kentucky in early 2004 was stabilised using a Nitrogen Pressure Swing Adsorption unit. The Queensland GAG unit was called to the Southland, NSW mine fire at the end of 2003 but not utilised in full.

The primary objective of the study is to use the Polish mine fire simulation software VENTGRAPH to gain better understanding of how inertisation particularly that generated by a GAG unit interact with the complex ventilation behaviour underground during a substantial fire.

An introductory section examines different available mine inertisation sources. Some considerations for selecting the best surface portal location placement for the inertisation unit for most efficient suppression of a fire have been examined. Introduction of inert gases can present difficult emergency management decision making. Should the main mine fans be turned off to reduce dilution of the inert gas, or will this action cause, in conjunction with fire induced buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

This section also examines the possibility of a wider and proactive application of GAG in Australian mines responding to or recovering from mine fires or spontaneous combustion heatings or elimination of the potential explosibility of newly sealed goafs is examined. The primary focus here is on systems involving delivery of GAG exhaust through docking to surface boreholes connecting into underground workings. Attainable designs for panel boreholes and how GAG docking to boreholes can improve delivery of GAG exhaust are discussed.

The section also examines the vital aspect of airway segregation and stopping leakage. Mains headings present a complex ventilation network with often numerous parallel headings, hundreds of cut throughs and a variety of ventilation control devices. In such a complex system (with additional interference from a fire), maintaining control of the movement of inert gas is more difficult than elsewhere in the mine. Some illustrations of this issue are given.

Another section examines attempts to examine calibration exercises that have been undertaken to verify the ability of VENTGRAPH to accurately simulate mine fire situations over time and the impact of introduction of inertisation gases ion the mine ventilation system.

A chapter discusses the VENTGRAPH fire simulation software and applications. This includes discussion on recent additions to the original VENTGRAPH software to allow incorporation of a variety of inertisation unit types and a greater variety of seam gas types.

This major section of the study is devoted to examination of the effects of fires and introduced inertisation on the Oaky North Mine and Oaky No1 Mine ventilation systems using fire simulation software VENTGRAPH. Fifteen major case study scenarios based on the modelling of fires with introduced inertisation at a number of high fire priority different points geographically spread within the longwall mine layouts are discussed. Inertisation outcomes in all case scenarios have been examined through introduction through the mine’s present docking point at the Transport Drift or the highwall down cast main shaft. Each scenario has then been re-examined one or more times to establish if a different docking point, altered underground ventilation segregation, use of boreholes to delivery GAG exhaust directly to the fire seat or other approaches would be more effective in stabilising the simulated fire.

Results have been analyses in detail, conclusions drawn and recommendations made. The fire simulation exercises have demonstrated that it is possible to efficiently evaluate possible inertisation strategies appropriate to a complex mine layout extracting a gassy seam and determine which approach strategy (if any) can be used to stabilise a mine in a timely fashion.

The outcome of the project will be that the Australian mining industry is in an improved position in their understanding of mine fires and the use of inertisation units following the very substantial work already undertaken and built around the introduction of the modern fire simulation computer program VENTGRAPH and the consequent modelling of fire scenarios at a substantial number of mines in Queensland and NSW.

Simulation software has the great advantage that underground mine fire scenarios can be analysed and visualised and actions planned to control fire contaminants, maintain safe escapeways and develop approaches to recovery. The VENTGRAPH software provides a dynamic representation of the fire's progress (in real-time) and utilises a colour-graphic visualisation of the spread of combustion products, oxygen, and temperature throughout the ventilation system. During the simulation session the user can interact with the ventilation system (e.g., hang brattice or check curtains, breach stoppings, introduce inert gases such as those generated by a GAG and other units and change fan characteristics). These changes can be simulated quickly allowing for the testing of various fire control and suppression strategies.

Because of complex interrelationships between the mine ventilation system and a mine fire it is difficult to predict the pressure unbalance and leakage created by a mine fire. Depending on the rate and direction of dip of incline of the entries (dip or rise), reversal or recirculation of the airflow could occur because of convection currents (buoyancy effects) and constrictions (throttling effects) caused by the fire. This reversal jeopardizes the functioning and stability of the ventilation system. Addition of the gas stream from the inertisation unit adds another level of complexity to the underground atmosphere behaviour. Should the main mine fans be turned off so as not to dilute the inert gas or will this action cause, in conjunction with buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

The project has increased understanding of these and other questions in the use of inertisation units. It also has reviewed in detail all types of inertisation units available in Australia and discussed how each can be utilised in a fire emergency. Simulations have been undertaken of the effects of common fire causes and fire progress rates. Inertisation units have been simulated at more than one mine “docking” surface point to help mines decide on optimal portal placement.

The Polish program authors have accepted recommendations for changes and have undertaken inertisation related modification to the VENTGRAPH software from the project findings. These improvements will be available free of charge or at cost price to all Australian mines that have already purchased the VENTGRAPH program as part of earlier mine site fire scenario exercises.

The exercise to introduce simulation of fires and their effects on mine ventilation networks supported by ACARP in 2003 and 2004 has been highly successful. Technology transfer from this project has occurred both to individual mines and in a large number of industry and professional forums. It is considered that the outcomes from this complimentary simulation project aiming to understand inertisation will be similarly received and be considered to be of great benefit to the mining industry.

Successful attainment of goals involved commitment of a number of parties. The research program led by a team consisting of experienced researchers with significant mine ventilation experience. The support of operating mines to allow examples of testing of fire scenarios incorporating inertisation in various mine layouts has been essential. Linkage with Mines Rescue Bodies and related parties was also essential. Some of the investigations were undertaken on site at mines. The research program involved a number of interlinked stages.

2. INERTISATION IN AUSTRALIAN COAL MINES

2.1. Overview of Mine Inertisation Systems

Successful underground mine inertisation is fundamentally dependent on being able to dilute or displace oxygen in the presence of an inerting agent to less than combustible levels.

A number of factors contribute to the success of underground inertisation:

- Flowrate of the inert gas
- Pressure of the inert gas
- Density of the inert gas
- Continuity of the inert gas supply

Low flow inertisation systems have been successful in the proactive inertisation of goaf areas and have the ability for total mine inertisation. A substantial period of time is required due to their low flow rates. To put it simply, large-volume units take less time to achieve the results of the smaller capacity systems, but consideration must be given to relative cost factors. Experience to date has shown that, where large volumes of inert gases are required, the GAG 3A system can deliver these large volumes on a lower cost per cubic metre of inert gas than many of the low-flow methods for which total cost information is available.

Each inertisation system has an optimum application dependent on the site-specific variables existing at a mine at the time of a combustion event. Experience has indicated that a risk based logic approach will aid in the selection and determination of the appropriate system for a particular application. To assist with selection of an inertisation system of choice Table 2.1 indicates both positive and negative variables for consideration prior to application.

Table 2.1 Comparison of inertisation methods (after Mucho et al, 2006)

Inertisation Methods	Advantages	Disadvantages
GAG 3A Jet Engine Inertisation System	▪ Large Volume ▪ Low cost per m^3 ▪ Mobility ▪ Access to the mine ventilation system ▪ Self Contained	▪ Manpower required ▪ Support Materials/Supplies ▪ Transport and Availability ▪ Training ▪ Higher O_2 (than CO_2 and N_2) ▪ Fire gas ratios unstable (due to CO & H_2 production)
Tomlinson Boiler	▪ Versatility ▪ Manpower (2 people/24 hrs) ▪ Portability	▪ Low Flow ▪ Time Duration ▪ High Maintenance

	▪ Minimal support materials/supplies	▪ Fire gas ratios unusable
CO_2 Liquid and/or Gaseous	▪ Cool ▪ Denser than air (can be advantage application dependent) ▪ Ease of movement ▪ Detection relatively easy	▪ Low Flow ▪ Method of application ▪ Transport and Availability ▪ Fire gas ratios unusable
N_2 Liquid and/or Gaseous	▪ Cool ▪ Lighter than air (can be advantage; application dependent) ▪ Non-toxic ▪ Injection ability ▪ Operational logistics relatively simple	▪ Low Flow ▪ Method of application ▪ Transport and Availability ▪ Fire gas ratios unusable

This section reviews the principal categories of inertisation systems in use in Australian coal mines. It also gives some application of the use of each approach.

2.2. Flue Gas Generator (Tomlinson Boiler)

The Tomlinson Inert Gas Generator has been developed primarily for use within the coal mining industry. It grew from large scale hot water heater systems used in institutions such as hospitals. The main application is to inertise underground mined areas to sealing from the mine ventilation system to minimise the possibility of methane gas explosions.

Figure 2.1 Photographic Views of Tomlinson Inert Gas Generator (after Tomlinson Boilers, 2004)

Figure 2.2 Tomlinson unit on mine site

2.2.1. Specification and current applications

Typical inert gas specifications produced by a Tomlinson Inert Gas Generator are:

- Gas volume - 1,800 m^3/hour or 0.5 m^3/s
- Delivery pressure - 100 kPa
- Oxygen content (O_2) - less than 2%
- Nitrogen content (N_2) - 75%
- Carbon dioxide (CO_2) - 12.5%
- Gas temperature - ambient + 20°C

Other possible applications for the Tomlinson Inert Gas Generator include:

- Purging LP gas storage vessels prior to internal inspections
- Purging sewerage digesters prior to internal maintenance
- Purging fuel storage tanks prior to internal inspections
- Dilution of process gas streams to adjust calorific value or chemical composition

Current users include many underground coal mines in the Queensland Bowen Basin.

2.2.2. Tomlinson application - ACARP Project C6002

ACARP funded research project C6002 titled "Sealing, Monitoring & Low Flow Inertisation of a Goaf" undertaken by Cook Resources Mining Ltd (CRM) tested the feasibility of this low flow inertisation concept and successfully demonstrated that the Tomlinson Inert Gas

Generator has considerable potential for the elimination of potential explosion hazards which may exist in some circumstances when areas of a mine containing flammable gases are sealed.

The aims and objectives of this project were:

- To develop, test and refine a sealing management plan.
- To identify gas monitoring protocols which occur in a goaf, before, during and after sealing.
- To demonstrate that low flow inertisation provided by a Tomlinson Inert Gas Generator of a sealed area will:
  - prevent the atmosphere behind the seals from entering the explosive range;
  - be achieved without interruption to the normal production cycle;
  - be cost effective.
- To develop a computer model which enabled future predictions for the inertisation of mine areas using either external inert gas generation, natural processes or a combination of both.

The low flow inertisation concept was successfully demonstrated at Cook Colliery during May/June 1997 and the overall result was very encouraging. In the project CRM confirmed that the Tomlinson Inert Gas Generator is capable of eliminating potential explosion hazards and possible business interruptions when areas of a mine containing flammable gas are sealed.

It must be recognised that the make or volume of flammable gas found in the 9 West Waste Workings at Cook and the 2 South District at Laleham Collieries prior to the inertisation trials was relatively low. However in the case of Cook the make or liberation of methane into the workings should have been sufficient for an explosive mixture to occur before the zone could self inert.

In both cases, the make or liberation of methane was much lower than expected and this raised a number of interesting points. The Tomlinson Inert Gas Generator produced a positive pressure in the sealed area of about 300 Pa at Cook and about 650 Pa at the seals for the 3 South District at Laleham.

It was noticed that this pressure increase was maintained regardless of diurnal variations in the barometer of up to 900 Pa and it would appear that provided the ventilation pressure across the seals has been balanced effectively the barometric pressure has little if any effect.

There was no evidence to support a hypothesis that this overpressure suppressed the release or desorption of seam gas and in particular methane.

There is also no evidence to support a hypothesis that the methane was in fact, displaced. Rescue Team personnel could find no evidence of layering or stratification of gases and this was confirmed by numerous spot tests and bag samples taken from the floor, roof and mid seam heights in a number of roadways.

2.2.2.1. Sealing issues

Good sealing practice dictates that ventilation should be maintained throughout the panel until the intake and return airways are blocked off or sealed simultaneously. In the past the mines had attempted to achieve this by sealing non-critical intake and return airways first and then coordinate the sealing of the main intake and return airways.

When two seals are to be erected simultaneously major challenges are faced. Seals require some time to cure in addition to the timing and resource problems before they are subjected to the low pressure produced by an Inert Gas Generator. The opinion was formulated that for the inertisation process, in any form, to be successful, maintenance of a ventilation circuit until the latest possible time was necessary. This may be some time after the actual inertisation process or injection of inert gas has commenced.

There needs to be an ability to close off the mine ventilation rapidly and to remove the mine's ventilation pressure from the seals.

The inertisation of CRM's 9 West Waste Workings at Cook Colliery was a success, in that:

- the oxygen in the sealed area atmosphere was reduced to a level below 12%;
- the methane level in the sealed area atmosphere did not reach the lower explosive limit for that gas.

There is no doubt that this project and in particular the inertisation trials broke new ground and the overall results were very encouraging. CRM is confident that low flow inertisation has considerable potential for the elimination of potential explosion hazards and business interruptions in some mines which contain flammable gas.

The trial at Cook Colliery lasted about 236 hours with 181 hours of effective pumping or inertisation time and this equates to an overall unit efficiency of about 77%. It should be recognised that the Inert Gas Generator was new and in fact commissioned on site during the early days of the trial. The methods employed were new and the operators were to a degree, self trained on the job.

2.2.3. Advantages and disadvantages

Advantages of Tomlinson use

- Well known to industry
- Inert gas is generated continuously

Disadvantages

- Requires a continuous feed of fuel and water
- Requires operator supervision
- Delivers inert gas at low pressure
- Combustion flue gas used for inerting is acidic
- Nameplate performance very difficult to verify on site
- No capable of large peak flows of inert gas
- High initial capital outlay

2.3. Mineshield Liquid Nitrogen System

The Mineshield system was developed by the NSW Mines Rescue Board and gas providers CIG in 1985 in response to the frequency of heatings in underground mines and the Appin explosion in 1979. The system works by 'boiling off' liquid nitrogen to generate inert gas.

The unit can be operational within 4 hours from the initial call. Liquid nitrogen which is supplied by BOC gases is delivered by tanker from their facilities in Wollongong, Newcastle and Brisbane and stored onsite tanks to be used between deliveries (Mines Rescue, NSW 2007).

The plant heats liquid nitrogen to convert it to nitrogen vapour which is then introduced into the problem area by a bore hole. Up to 17 tonne per hour of liquid nitrogen per hour can be used to lower the oxygen content of the problem atmosphere to less than 2%.

Figure 2.3 Mineshield inertisation unit on mine site

Figure 2.4 Mineshield on mine site, pump and vaporisation unit

2.3.1. Specification and current applications

The components of the Mineshield unit are:

- 40 tonne Storage Tanker
- Vaporizer trailer
- Pump trailer
- LPG supply tanker

Additional requirements are:

- Site pad capable of bearing 5.7 t/m^2
- Water (10,000L at start-up, plus ongoing supply)
- A 400 kVA power supply; provided by grid or generator
- Site lighting
- Phone lines
- Four operators (provided by BOC)
- Road access and turning facility for B-double tankers.

An example site layout for Mineshield is shown in Figure 2.5. The unit consumes between 1t and 17t of liquid nitrogen per hour. One tonne of liquid nitrogen equates to 860m^3 of inert nitrogen gas. Thus the unit can produce up to 4 m^3/s of inert gas. It is believed that an output of only 2 m^3/s can be sustained over a long period. Current supplies of liquid nitrogen are limited to 400 t/day at Port Kembla and 100 t/day at Brisbane. Transport of liquid nitrogen involves significant logistical difficulties due to restrictions on movement of dangerous goods and the limited number of suitable rigs.

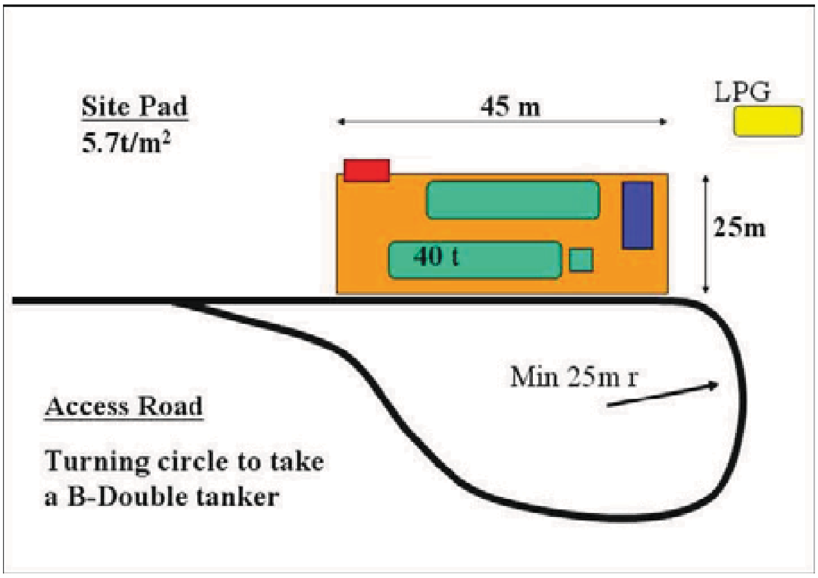


Figure 2.5 Mineshield plant layout.

Liquid carbon dioxide can be used in place of nitrogen but is only suited to a limited number of scenarios as it is heavier than air and tends to sink to the bottom of the mine. This greater density also results in a lower volume of gas per tonne.

Advantages of the Mineshield unit over the GAG are:

- Greater control of low oxygen content in output.
- Nitrogen from Mineshield can be distributed to boreholes using flexible hosing. Thus multiple holes can be injected at once, or injection points can be changed by simply reconnecting hoses. Hoses can also be used to distribute the inert gas underground.
- Inert gas from Mineshield also has a cooling effect.

Disadvantages are:

- Higher operating cost
- Lower flow rates
- Greater demands on infrastructure

Mineshield has been used at five mines in NSW and at Moura No. 4 in Queensland. The most recent operation was at Dartbrook in the Hunter Valley during 2002 when inert gas was injected into the goaf behind the longwall face over five months (Coal Service, 2003). This case was a success with production maintained and all face equipment safely recovered.

2.3.2. Mineshield application - 1986 Moura No. 4 underground mine inertisation

Shortly after the disaster borehole drilling was commenced to facilitate further sampling, water injection into the goaf and nitrogen injection into the workings. At approximately 8:00 a.m., Sunday 20th July, the Mineshield equipment, technical personnel and four tankers containing a total of 64 tonnes of liquid nitrogen arrived on site (Lynn, 1986). However, the propane gas tanker which contained fuel for the vaporising unit was delayed.

Attempts were made to inject liquid nitrogen directly down two boreholes. These attempts proved unsuccessful as back pressure in the bore holes caused the liquid nitrogen to force its way back to the surface via cracks in the subsoil. This eventually froze the ground and blocked the borehole. Water injection was also proving difficult through blockages in the uncased boreholes. Both operations were abandoned and recovery of the blocked holes by reaming and casing was commenced in anticipation of the nitrogen vaporising unit becoming operational and the arrival of further quantities of liquid nitrogen.

Bore hole recovery and mine atmosphere sampling continued into Monday 21st July, when sample results determined at 10:00 a.m. from bore holes indicated the mine atmosphere about the Main Dips Section was not explosive. Further samples 1 hour later provided similar information.

As a result, rescue team 9 accompanied by a District Union Inspector, The Mines Rescue Superintendent and the Government Mines Inspector entered the mine. During this inspection concern arose about the accuracy of sample results received up to that time because a thick bluish smoke and a "fire stink" were detected. These signs indicated the existence of an active fire inbye of 22 Cut Through (Lynn, 1986).

Further exploration attempts were suspended and attempts were made to inject nitrogen gas. The first significant injection rate of 5 tonnes per hour was achieved at approximately 6:00 p.m. This rate was increased gradually to 14 tonnes per hour at 8:00 p.m. causing the oxygen levels to be slightly reduced. However, this rate could not be maintained due to the difficulties of getting sufficient nitrogen to the site. It was evident that the natural ventilation flow in the unsealed panel was diluting the nitrogen and it was calculated that to reduce the atmosphere to 12% oxygen would require an injection rate of 18 tonnes per hour which could not be guaranteed.

On Tuesday, 22nd July, water injection to the goaf area was recommenced to reduce the area to be inertised by nitrogen.

Rescue teams 10 and 11 entered the mine to locate the source of smoke and to erect brattice seals to reduce the quantity of airflow in the panel. While these teams were underground, the nitrogen injection rate was set at 10 tonnes per hour.

A large area of smouldering floor coal as well as evidence of burnt out props was discovered in 24 Cut Through between 2 and 3 Headings. A new sample tube point was established inbye and all roads were sealed by brattice.

Drilling of a borehole was commenced directly over the heating to allow the injection of nitrogen vapour into the area. On Wednesday, 23rd July, at 8:20 a.m., the hole was completed and nitrogen at the rate of 3 tonnes per hour was pumped through the drill stem. With sufficient quantities of nitrogen on site and additional supplies in transit, it was decided to attempt to recover the bodies.

The nitrogen injection rate was increased and five rescue teams were prepared for the recovery operation. By 1:00 p.m., oxygen levels had been reduced sufficiently to allow the operation to commence. Rescue teams 12, 13, 14, 15 and 16 were to prepare and remove the bodies to the fresh air base.

Physical conditions were extremely arduous with high temperature and humidity, very poor visibility and extensive blast debris. However, in spite of these conditions all of the bodies which had been previously located were recovered together with the two bodies which had not previously been located. One of these was located wedged beneath the outbye section of Shuttle Car No. 31.

The last of the bodies was transported to the surface by 5:15 p.m. The Mineshield equipment was shut down at approximately 5:30 p.m. It appeared that inertisation of the sealed area had been successful in that the oxygen level had remained outside the explosive range (Lynn, 1986).

2.3.3. Mineshield application - 2002 Dartbrook mine heating inertisation

On the 16th May 2002 the Hunter Valley Station responded to a spontaneous heating in the longwall goaf at Dartbrook Colliery (Coal Services, 2003). The Mineshield Inertisation Plant was activated to pump an average of 4 tonne/hour of liquid nitrogen into the area. The longwall equipment had been removed and, by 30 September, the goaf area sealed and inerted. During the whole operation approximately 10,500 tonnes of liquid nitrogen had been used. The Plant remained on standby at the mine until 8th October 2002.

This protracted utilisation of the Mineshield Plant put a strain on the pumps and the electrical systems which were only designed for short intense usage of up to 18 tonne/hour of liquid nitrogen. Following a review of the performance of the Mineshield Plant, it was decided to undertake a capital upgrade of the plant to ensure it remained operative and effective for the next 15 years. The upgrade was completed late in 2003.

2.3.4. Advantages and disadvantages of Mineshield

Advantages:

- Utility requirements are relatively minor
- Can deliver large amounts of inert gas in a short time

- Can deliver inert gas at pressure
- Gas 100% inert
- Good solution for limited use applications

Disadvantages:

- High set up cost
- Requires set-up of liquid nitrogen vessel on-site
- Requires a continuous fleet of liquid nitrogen tankers to maintain inert gas supply
- High specific cost of inert gas

2.4. GAG 3A Jet Engine Inert Gas Generator

The GAG-3A Jet Engine Inert Gas Generator consists of a modified jet engine which no longer produces thrust. The engine was originally built for use in a Polish military training aircraft and has subsequently been adapted to generate inert combustion products (Prebble and Self, 2000). The stages within the engine are:

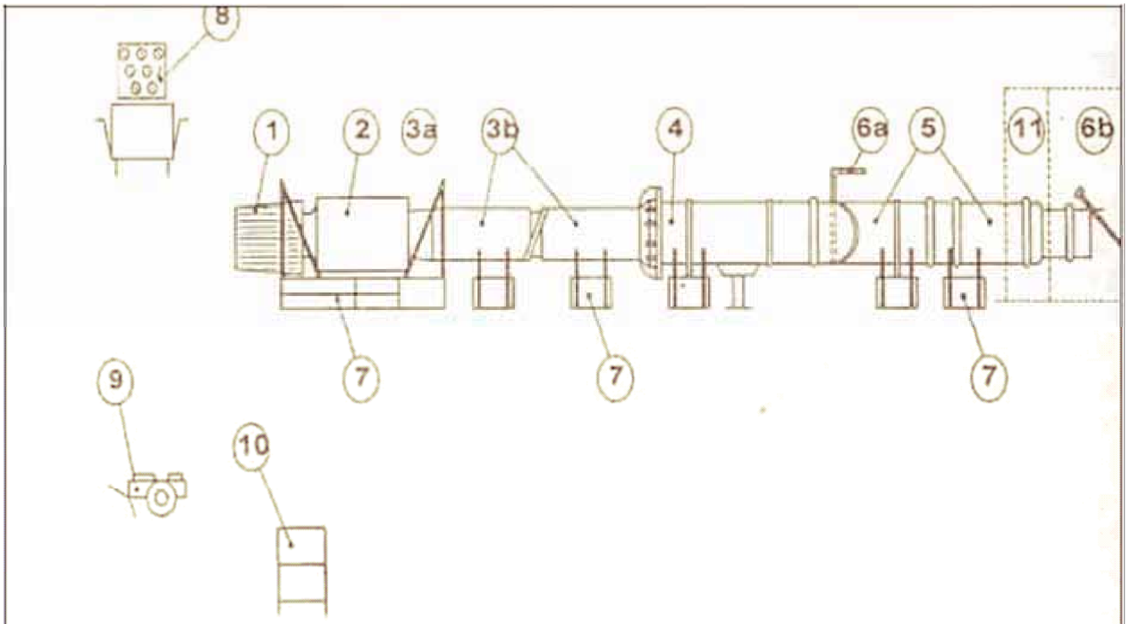
- Compressor
- Combustion chamber
- Turbine
- Afterburner and mixing chamber
- Cooling

A photo of the assembled unit is shown in Figure 2.6. The length of the unit is approximately 12m (Parkin, 2005).

Figure 2.6 One of the Queensland Mines Rescue Service GAG-3A inertisation units

The engine can operate at speeds up to 11,000 rpm, with a standard operating speed of 8000 rpm. The unit produces approximately 25 m^3/s of moisture saturated gas at 85°C, equivalent to 10 m^3/s dry gas. Claims of the oxygen content in the exhaust gas vary from 0.1% to 5% and 2-3% is a realistic target while the unit is running smoothly. There may be a relationship between oxygen in the combustion products and other operational factors. The unit requires up to 2000 L/hr jet fuel and 40,000 L/hr of water for cooling during operation.

The Queensland Mines Rescue Service owns and operates two GAG units that were purchased in 1998 (Parkin, 2005). These units are currently stationed in the Bowen Basin. The unit is transported by truck and setup time for operation is approximately three hours once equipment is on site. The GAG inert gas generator has also been used in Poland, the US, Kuwait (for oil well fires), the Czech Republic and South African gold mines (Page, 2003 and Parkin, 2005).



Item	Description	Item	Description
1	Air Filter (1m x 1.2m)	6b	Fire Dam Door (Shown Open)
2	Engine (1.1m x 1.6m)		
3a	After-burner Fuel Injector Chamber	7	Skids
3b	Water-jacketed Cooling Tubes (2)	8	Control Console
4	Cooling Chamber (1m x 2.3m)	9	Battery Cart
5	Ductwork (1m x 3.2m)	10	Fuel Tank and Pump
6a	Butterfly Valve		

Figure 2.7 Schematic of GAG-3A inert gas generator.

Figure 2.8 Assembled GAG engine (after Romanski, 2004).

2.4.1. Specification and current applications

Some operational parameters are given in the following table.

Table 2.2 Operation Parameters of the GAG-3A Inert Gas Generator.

	Parameter	Unit	RPM	
			7200	11000
1	Flow	m^3/s	13.95	33.25
2	Mass of inhaled air	kg/s	5.48	10.76
3	Fuel Consumption	litres/min	approx. 17	approx 32.5
4	Oil use	litres/hour	1.3	1.3
5	Cooling Water for afterburner at 70kPa Range 60kPa to 90kPa	litres/s	5	5
6	Water cooling system exhaust gas cooling at 350kPa Range 200kPa to 450kPa	litres/s	7.5	7.5
7	Inert gas temperature on GAG-3A exit	°C	approx. 85	approx. 85
8	Min. water pressure Afterburner Tubes Diffusive Cooler	kPa	70 350	70 350
9	Approx gas make: Oxygen Carbon Dioxide Nitrogen Carbon Monoxide	% % % ppm	0 to 0.5 13 to 16 80 to 85 approx. 3	0.5 to 2 13 to 16 80 to 85 approx. 3

2.4.2. GAG-3A application - ACARP Project C6019 and Collinsville mine trial

Surface and underground trials of the GAG-3A jet inertisation device were held at the Collinsville No 2 underground coal mine from 7th to 18th April 1997 (Bell et al, 1997).

The selection criteria for the trial developed by the Moura related Task Group 5 Committee were met with the exception that output flow rates were slightly below the levels predicted

(19 m^3/s against an expected 20-25 m^3/s). This diminution in flow rates was attributed to higher ambient air and water temperatures.

The unit operated safely during all aspects of the trial and no mechanical problems were encountered. Over 100 industry stakeholders visited the demonstration and feedback questionnaires were generally positive. The demonstration supported the view that the device was suitable for coal mine use. No external flame was visible on the device.

The unit produced noise levels in excess of 124 dB(A) (when measured 1 m from the jet) in both surface and underground operations. Environmental noise levels measured 2.3 km from the GAG-3A were not impacted by the operation of the unit. The limited stratification experiment conducted indicated that the gas produced by the GAG-3A tended to move closer to the roof than the floor (Bell et al, 1997).

The trial demonstrated that the GAG-3A device has applications in underground coal mines and that it outperformed all other available technologies with respect to volume of inert gas produced. It is clear that the GAG-3A produces a low oxygen level output and has a wide range of applications. The device produces a large volumes of low oxygen inert gas which can be used to replace a potentially explosive atmosphere in an underground coal mine.

The GAG-3A inert gas generating device was developed in Poland in the early 1970's and has been used extensively in Poland, Czech Republic, CIS, China, and more recently, to combat frequent and extensive gold mine fires in South Africa. A variation of this device was used, mounted on a remotely controlled tank, to extinguish the oil well fires in Kuwait following the Gulf War. The GAG-3A has been used for tens of thousands of operational hours with no serious accidents reported to date. The device was brought to Australia by the Polish Mines Rescue Service with SIMTARS providing operational support for this ACARP and industry-funded project.

Following the explosion at the Moura No 2 coal mine in 1994, the subsequent inquiry recommended that various forms of inertisation be investigated with regard to their suitability for use in Queensland coal mines.

Task Group 5 under the auspices of the Moura Implementation Committee was formed with two main foci, inertisation and the suitability of the current sealing strategies in use in underground coal mines. This project focussed on the demonstration of one particular inertisation strategy

The GAG-3A jet inertisation device was trialled under a variety of circumstances at the Collinsville No 2 Mine. The device produced large volumes of inert gas and complied with the criteria set down by Task Group 5 with the exception that due to site-specific conditions at

Collinsville No 2 Coal Mine, relating to water and ambient temperatures the flow rate of inert gas was 19 m^3/s rather than 20 m^3/s nominated in the Task Group 5 selection criteria.

This compares very favourably to the only other significant trial of inertisation in Queensland, at Moura No 4 in 1986, where 700 tonnes of liquid nitrogen were injected into a mine area over a period of 5 days to produce an oxygen level of less than 10%.

The GAG-3A achieved similar results in 6 hours at a fraction of the cost - $600,000 liquid nitrogen versus $4,500 Jet A fuel (Bell, et al, 1997).

In the mine the device operated faultlessly although there was one minor stoppage due to dirty fuel filter problems. The jet was re-started in less than 10 minutes. The operation of the device should be supervised by a competent ventilation engineer. It is clear that the GAG-3A produces lowered oxygen levels over a wide range of excess air conditions and therefore has a wide range of applicability.

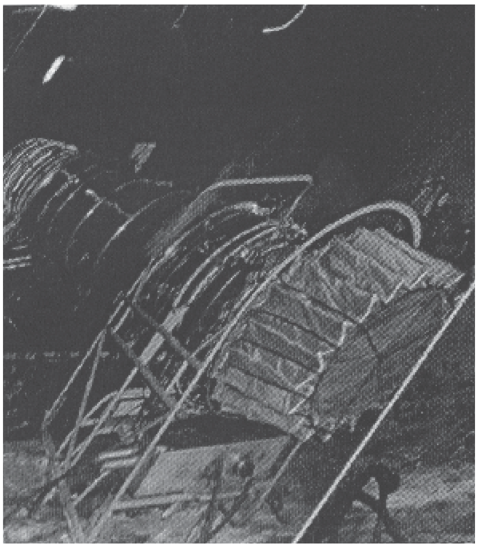


Figure 2.9 Demonstration of the GAG-3A jet inertisation device at the Collinsville No. 2 Coal Mine April 1997 ACARP Project 6019

On the basis of the trials conducted at Collinsville No 2 Coal Mine the GAG-3A device was concluded to be is applicable with respect to conducting relatively high quantity inertisation in underground coal mines. It was also concluded that each usage of the device would be dependent upon site-specific factors and the GAG-3A may not be suitable for every mine situation (Bell et al, 1997).

2.4.3. GAG-3A application - Blair Athol open cut mine, 1999

Blair Athol is an open cut mine in Queensland's Bowen Basin, 25km north west of the town of Clermont. The Number 3 seam mined at Blair Athol contains old underground workings from several collieries originally mined between the 1890s and 1960. As the old workings were exposed there had been a history of heatings but all had been easily treated (Prebble and Self, 2000).

However in July 1999 a new strip was commenced and the problems encountered were much greater than previously experienced. Coal in the strip rapidly heated and open fires formed in many areas. The propensity of this coal to self-heat presented two major risks for mining of the strip:

1. Smoke coming out of the old workings contained toxic levels of carbon monoxide, up to 1200ppm.
2. The possibility of an explosion in the old workings, fuelled by hydrogen, methane and carbon monoxide produced by oxidation of the coal.

Eight boreholes were drilled into the old workings to determine the atmosphere within the abandoned colliery. Results showed high concentrations of explosive gases and low oxygen, typically 10% CO, 12% hydrogen, 4% methane, and less than 1% oxygen (Prebble and Self, 2000).

To disperse this mixture of explosive gases it was decided to displace the explosive gases with inert gas. The GAG-3A inert gas generator was chosen to flush out the workings and then a Tomlinson boiler and/or Floxal nitrogen unit was used to maintain the inert state in the colliery. To provide access for the GAG into the old workings a 900mm borehole was sunk (to approximately 50m depth). Typical running times for the GAG at Blair Athol were 1-4 hours and in total five campaigns were run. Experiences stowed that it was highly effective in flushing out explosive gases although some teething issues with equipment and labour problems occurred.

2.4.4. GAG-3A application - Loveridge Mine, West Virginia, 2003

The Loveridge No.22 mine is operated by Consol Energy Inc. Following a fire in February 2003 the mine was evacuated and sealed. Shortly afterwards Consol contacted QMRS about the possibility of using one of the GAG units to extinguish the fire as shown Figure 2.10.

Two teams of operators were invited to assist Loveridge. The GAG ran for approximately 240 hours over 13 days and was successful in inertising the fire area. Following this success the mine could be re-entered by rescue teams and the mine could be unsealed.

Parkin (2005) also observed the effects of natural ventilation pressures, barometric changes, and rock falls on the backpressure experienced by the GAG. This was a key issue as high ventilation system backpressure resulted in a number of operational delays.

Figure 2.10 GAG-3A inertisation unit in use at the Loveridge mine, 2003

On February 13, 2003 a fire began in a trash car near the bottom of the slope in the Sugar Run area of this longwall mine, The mine was evacuated after some direct fire fighting attempts, the mine openings sealed and six boreholes drilled in and around the fire area for monitoring and water pumping.

By March 2003, attempts to suppress the fire with water pumped from the surface had not been fully successful. A decision was then made to attempt to inert the fire area using the GAG 3A jet engine technology. The QMRS was contacted to deploy an engine and operating personnel in a joint collaboration with Consol Energy (Parkin, 2005).

A ventilation simulation of the inertisation situation at Loveridge Mine was done by the operator, which concluded that it was feasible to inert the whole mine. On March 8, 2003, preliminary plans for a means (a docking facility) to connect the GAG 3A system to the existing slope entrance structure were discussed and the design of the necessary components was begun. The slope, initially sealed with a make-shift seal, would permit the inert gases to travel to the fire area near the slope bottom and continue through the main entries of the mine to the other shaft areas.

On April 4, 2003, the GAG 3A system had arrived at the Loveridge Mine site and, after some maintenance to the jet engine and its components; the system was commissioned for inertisation operations. Following the first 12 hours of engine operation, the unit was shut down due to overheating.

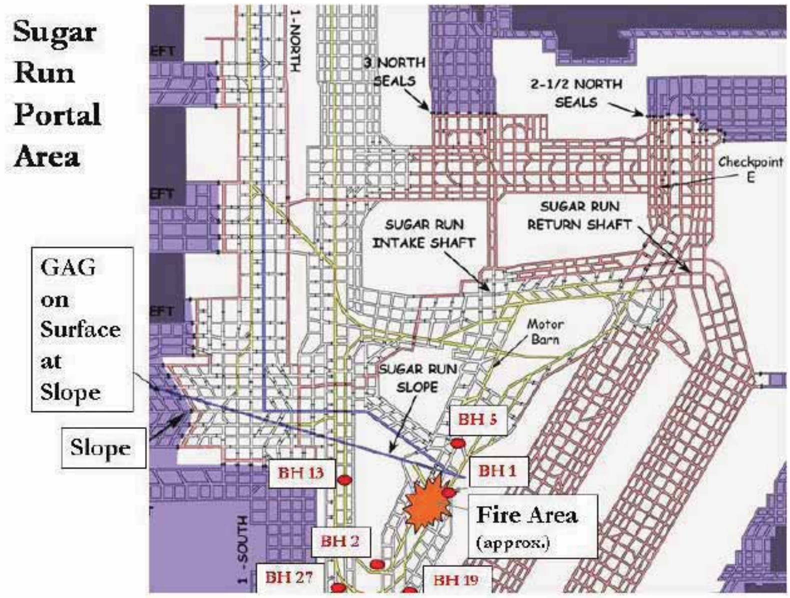


Figure 2.11 Fire area at Loveridge Mine (after Mucho et al, 2005).

On April 5, 2003, after 16 hours of run time, the plywood slope seal was leaking steam and the outer connection to the slope as shown in Figure 2.12 developed gaps and was leaking badly. The engine was shut down while repairs were made to the slope seal. Leaks in the ductwork attached to the slope were sealed with polyurethane foam and sand bags.

Figure 2.12 GAG 3A jet engine system interface with Sugar Run Slope. Note steam indicating exhaust gas leakage.

By April 9, 2003, boreholes and shafts in the Sugar Run area were out-gassing and readings indicated the presence of the exhaust gases. Examples of the presence and effectiveness of the engine exhaust gases in the Sugar Run bottom area, indicated by the presence of exhaust gases (CO_2 as the identifier) and the decrease in O_2 are shown in figures 2.13 and 2.14.

To facilitate movement of the exhaust gases throughout the entire mine, the seal of the St Leo Return Shaft, located at the opposite end of the mine was breached. After 5 days of operation, the jet exhaust gases had reached St Leo shaft and by April 13, 2003, St Leo and Miracle Run fans were first started, which helped to pull inert gases through the mine from the Sugar Run bottom area.

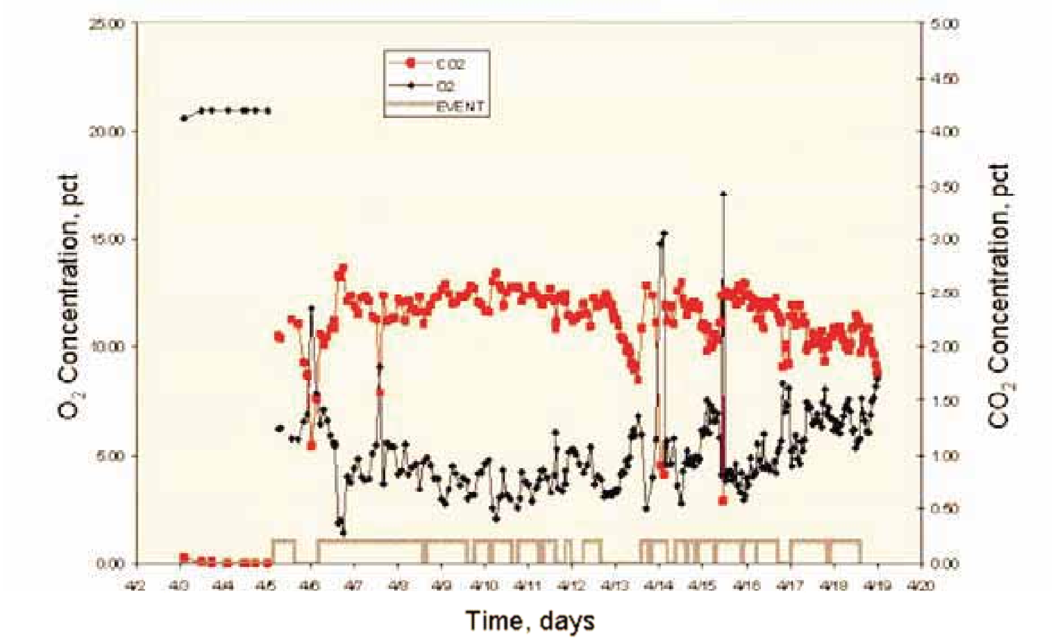


Figure 2.13 CO2 and O_2 concentrations in Sugar Run Slope, Loveridge Mine. "Event" as shown on the graph indicates GAG 3A operation (on/off) (after Mucho et al, 2005).

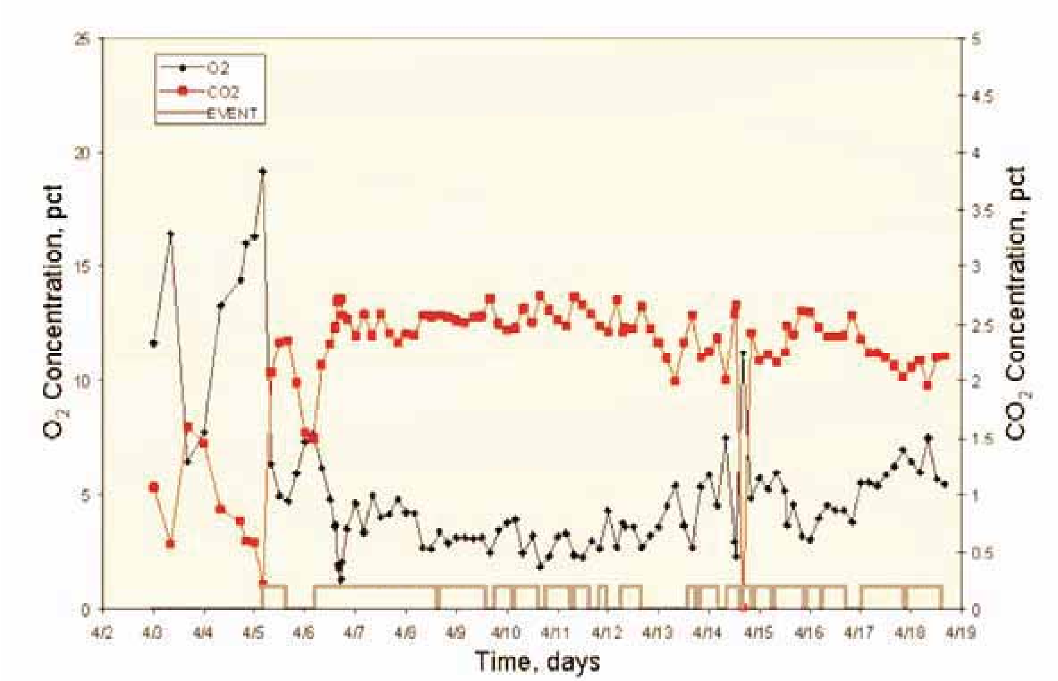


Figure 2.14 CO_2 and O_2 concentrations at borehole #1, Loveridge Mine. "Event" as shown on the graph indicates GAG 3A operation (on/off) (after Mucho et al, 2005).

At 8 pm on April 15th, 2003, the first rescue team re-entered Loveridge, 61 days after the mine was sealed and 10.5 days after the GAG engine was started. The GAG 3A system was operated intermittently during the mine rescue re-entry operations until April 21, 2003, when operations were terminated. In total, the engine operated for 270 hours with minor servicing and replacement of consumable parts and the engine consumed an average of 1,600 litres of fuel per hour and 18,500 litres of water per hour.

During its 17 days of operation at the Loveridge Mine, the GAG 3A system was able to render inert the Sugar Run bottom area and the over 9.5 miles of passageways at the Loveridge Mine by reducing the O_2 concentrations (Conti and Lazzara, 2003). As a result, the mine rescue teams were able to safely re-enter the mine to explore and ultimately isolate the fire area, which still had indications of active combustion, and was further inerted with N_2 injection and permanently sealed. At the time, this was the longest that the jet engine had operated for a mine inertisation application and some system components failed, but these occurrences were handled without major impact to the overall inertisation process. The potential benefit of more positive sealing of connections, ports, and mine seals was also recognized. Finally, the Loveridge Mine experience also provided a learning process for those involved and demonstrated that the GAG 3A system would be a valuable tool for fighting mine fires in the U. S.

2.4.5. GAG-3A application - Pinnacle Mine, West Virginia, 2003 and 2004

A series of four explosions between August 31 and September 16 2003 occurred at the Pinnacle Mine, Pineville, WV, in the active #8 longwall district shown in Figure 2.15. Mine gas readings from the various monitoring boreholes indicated that there was active combustion ongoing at an unknown location in the longwall district. The operator began drilling additional boreholes into the longwall gateroads to detect the heat source. Phoenix First Response was contacted by the operator and arrangements were made to utilize the GAG 3A jet engine in an attempt to inert the approximately 3 km by 3 km longwall district to extinguish the fire. Arrangements were also made to have trained GAG operators from Poland man the operation of the jet engine (Mucho et al, 2005).

By October 1st, the engine had been set up at the 8A bleeder shaft and the operators had arrived. The engine was started late in the day after a crane had removed the bleeder fan elbow conduit from the shaft and replaced it with a specially designed GAG docking hood that was then fastened to the shaft coping. This system had considerably less leakage issues than the Loveridge slope structure and temporary seal which were not as amenable to a pressurized, leak-proof connection.

The GAG 3A system ran successfully through October 7th with only occasional operational or maintenance issues. S. Fork fan was operated and ventilation adjustments made to assist in drawing the inert exhaust gases toward the active longwall. Even so, as occurred at the Loveridge Mine, there were periods when the engine would see more or less backpressure from the mine. Theories as to why this was occurring abounded and included flow restrictions, barometric pressure influences, and an "air bubble". Obviously, the exact cause could not be determined, but perseverance in terms of continuous operation of the engine seemed to overcome the variable backpressure problem.

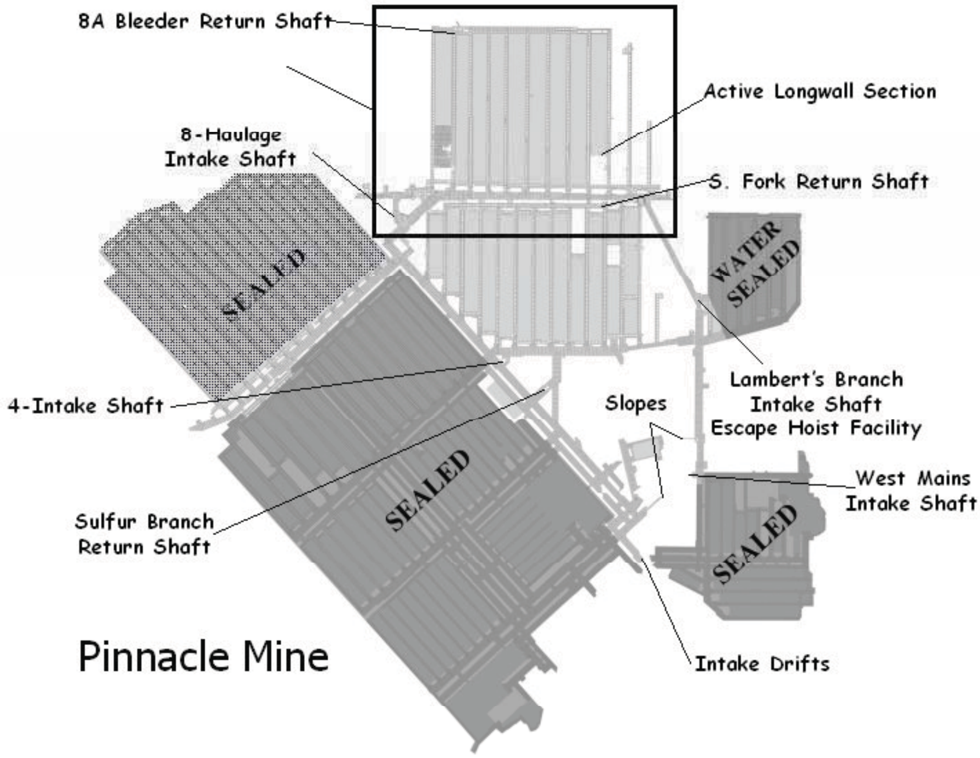


Figure 2.15 Pinnacle Mine showing #8 Longwall District, 8A Bleeder fan, and active longwall.

Tracking the underground movement of the exhaust gases via the monitoring boreholes in the #8 longwall district indicated that the exhaust gases initial migration was generally down-dip from the 8A shaft bottom, i.e., the structurally low northwest corner of the district inerted first and then the inerted zone gradually moved up-dip. Gravity (their higher density) may have been a reason for this, although ventilation (the operating S. Fork fan), ventilation controls, water accumulations, or goaf resistance could also have been reasons in whole or in combination (Mucho et al, 2005).

By October 8th, the inert gas front was approaching the active 8I longwall area and the five pressure monitoring borehole sensors measured a sudden pressure increase attributed to a gas ignition or explosion. The time to initially inert the desired area of the mine took approximately 7 days, which is the length of time that the mine had originally estimated.

The GAG 3A engine continued to be operated through October 19th in an attempt to maintain the inert area near the active longwall face. During this time, the pressure transducers measured another, much lower magnitude and less sudden, pressure increase on October 14th, indicating a possible explosion. Also during this time period, the mine operator, using a compressor, brought inert gases out of a borehole in the lower elevation area of the longwall district, transported the gases overland via a pipeline, and pumped the inert gases into boreholes closer to the area of the suspected ignition source in the goaf behind the active longwall face (Mucho et al, 2005).

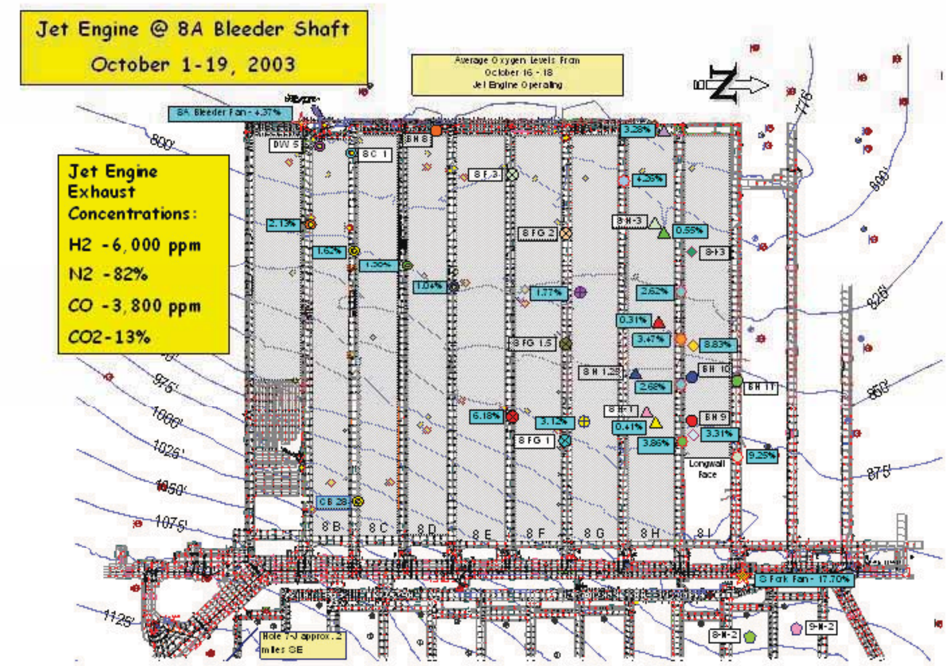


Figure 2.16 Pinnacle #8 Longwall District depicting bottom of coal elevation contours, 8I active longwall face, monitoring boreholes and GAG exhaust gas levels.

In the time period following the inertisation process, a more extensive array of pressure monitoring was installed in the longwall district and the mine was alternately ventilated using different ventilation scenarios. This ventilation process was an attempt to determine whether the ongoing ignition source had been successfully and completely extinguished by the inertisation process. While gas readings did not show conclusively that combustion was ongoing in the area, concerns about the presence and interrelationship of relatively small amounts of H_2, CO, and CH_4 delayed re-entry until a localised inertisation plan was instituted early in 2004. This re-entry plan is presented by Smith (Smith et al., 2005). The #8 longwall district was first temporally sealed and then permanently sealed in February and March of 2004, permitting continuous miner production to resume on April 7, 2004. Following re-ventilation of the 8I longwall panel in May, longwall production resumed on May 17, 2004.

2.4.6. GAG-3A application - Southland colliery December, 2003

Southland Colliery is located near Cessnock in the Hunter Valley. The mine, which was owned by Gympie Gold Ltd, was sealed following the fire but has since been reopened. It was concluded that the fire was started by spontaneous combustion in the goaf adjacent to the longwall face, and was unable to be contained.

Romanski (2004) gives an account of the use of the GAG at Southland. The initial callout came on the evening of the 24th of December 2003. The GAG unit and operators were mobilized on the 25th, and all necessary work before operation could commence was completed on the 26th. Operation commenced on the 27th and the engine ran almost continuously for over 43 hours.

Figure 2.17 GAG 3A Jet Engine set up at Southland Colliery, 2003

On the 29th Gympie Gold elected to halt the inertisation operation and the GAG was dismantled. During the operation the GAG unit consumed 65,000l of jet fuel, equating to approximately 1.4 million m^3 of dry inert gas. The effect of the inert gas on the fire was difficult to judge as the injection point was a long way from the fire site causing a long delay before inert gas could have reached the fire, and the lack of information available about the state of the mine following sealing.

2.4.7. Global Steamexfire Inertisation Services (GSIS) jet unit

The Global Steamexfire Inertisation Services unit operates in a very similar way to the GAG-3A jet engine and is purpose designed for mine fire inertisation. Owned by a Dutch company, GSIS launched its technology in 2005 and has been involved in the successful stabilisation of a coal mine fires at the Anglo Coal Goedehoop Colliery in South Africa in April in 2005 and the Svea Nord longwall coal mine in Spitzbergen, Norway in September 2005 (GSIS, 2007).

The company is reported to be working on offering more advanced technology. One of their upcoming releases will be a software inertisation program that will allow mines to run experimental exercises in real time, such as deciding the best possible inert site for a mine fire, goaf inertisation during longwall changeouts and experiments showing contaminate distribution underground and Steamexfire's inertisation effects.

2.4.8. Advantages and disadvantages of the GAG-3A

Advantages

- Well known to industry
- High volume inert gas generation

Disadvantages

- High set-up cost
- Requires a continuous feed of fuel and water
- Requires intense operator supervision
- Deliver inert gas at low pressure

2.5. Membrane Nitrogen Filter Units

The nitrogen production from an on-site generator is derived through a hollow fibre membrane separation process. It is the property of the membrane fibre that certain gases pass more quickly through the wall of the fibre than others. Filtered compressed air is introduced at the inlet of the bundle. As air (approximately 78% nitrogen, 21% oxygen, 1% other) passes through the hollow fibres, water, carbon dioxide, and oxygen molecules permeate through the wall of the fibre more quickly than nitrogen molecules. In this way, atmospheric nitrogen is concentrated as it passes along the membrane fibre. The process is continuous – this means there are no pressure swings. The inherent design of the membrane system means that systems are modular (allowing expansion of existing systems for more capacity as opposed to installing a larger unit), and that nitrogen production turndown is possible on larger systems.

A typical membrane system is comprised of:

- a 13 or 15 bar (g) single stage lubricated rotary screw air compressor/compressors with integral air drier,
- a membrane cabinet which includes compressed air filtration and membrane bundles,
- a mass flow meter and totaliser for performance monitoring and complete transparency,
- a downstream receiver to act as nitrogen buffer and source of peak nitrogen gas.

The entire system can be run automatically and remotely and is self monitoring. Similar units are available from other manufacturers in Australia.

2.5.1. Specification and current applications of the Floxal AMSA units

The AMSA's systems for underground inerting have been purpose built for the Australian coal mining industry. They have been designed as a complete, portable, skid mounted package. The primary components of the AMSA system are (Sajimon, 2005):

- Lubricated screw air compressor/compressors
- Air drying and filtration skid
- Nitrogen membrane air separation modules
- Nitrogen buffer/receiver
- Transportation skid/frame
- PLC control system and TeleFlo wireless communications computer

Figure 2.18 Floxal membrane inertisation unit at mine site

Atmospheric air is filtered and compressed in a standard air-cooled single stage lubricated screw air compressor/compressors. Compressed air is cooled with an air-to-air heat exchanger and dried with a refrigerated air drier. A condensate drain and two coalescing filters remove the liquid carryover entrained in the compressed air. An activated carbon filter removes hydrocarbons that may carryover past the filters. The clean dry air is then heated to ensure a uniform feed air temperature into the nitrogen membrane modules. As air passes through the membrane modules, oxygen and remaining water vapour are vented (discharged through the waste gas header) and nitrogen gas is concentrated. An oxygen analyser continuously monitors the produced nitrogen to ensure that oxygen levels are maintained at all times. Nitrogen gas is discharged from the AMSA at 9 bar pressure (Sajimon, 2005).

The operation of the AMSA system is monitored and controlled by a PLC. Interfacing with the PLC is TeleFlo – Air Liquide's proprietary telemetry and communications computer. Teleflo is Air Liquide's facility management system based on a robust combination of industrial PC hardware and software specific to Air Liquide's Floxal systems.

The TeleFlo system includes a GSM wireless link to interrogate the Floxal system at any moment and observe all process parameters. Conversely the Floxal system is able to 'call out' to signal routine maintenance for example or to issue alarms if necessary.

Technical staffs in all regions carry pagers which the Floxal units TeleFlo system can 'call' day and night. On being paged they are able to connect to the site with portable computers, and to investigate the nature of the problem. The TeleFlo system is powered by an un-interruptible power supply so that it can proceed with calling out to signal alarms as necessary and may still be interrogated remotely.

Floxal AMSA for goaf inertisation performance specifications (AMSA Floxal Unit, 2006):

- Nitrogen flow: 1934 m^3/hr
- Nitrogen gas pressure: 9 bar
- Residual oxygen: 3%
- Design availability: 98%/annum

Floxal AMSA utility requirements:

- Water: none
- Fuel oil: none
- Operators: none
- Electricity: 3 phase 415VAC, 805 kW, 981 kVA
- Surface Preparation: Level packed earth; sleepers may be used to distribute skid load and aid levelling.
- Telemetry connection: phone line, reliable GSM network reception, or satellite phone connection.

Footprint and weight:

- A standard AMSA 3000 series (with 14 x 12''Modules) nitrogen generation unit with after cooler and refrigerated drier, skid mounted. (L: 14.5m x W: 3.5m, Wt: 20 T)
- A Compressor skid having 3 x Kaeser ESD 441 air compressors with MCC. (L: 14.5m x W: 3.5m, Wt: 25 T)
- One 8 KL nitrogen buffer vessel mounted on the AMSA Skid.
- The ancillary equipment necessary to ensure the stable and reliable operation (control panel, PLC, Teleflo, interconnecting piping and wiring) of the Floxal and compressor.

2.5.2. Advantages and disadvantages

Advantages

- No CO, therefore no masking of spontaneous combustion heatings
- No toxic gas introduced
- Low set-up cost and easy set-up
- Can deliver gas at pressure
- Possible of centralized "fixed" positioning of the unit
- Inert gas generated continuously
- No need for operator supervision

- No requirement for fuel or water
- System and Performance is monitored
- 1934 m^3/hr of nitrogen from a single AMSA system can be delivered over 12.5 km through a 4" pipeline.

Disadvantages

- Requires a continuous supply of 415 VAC electricity
- High cost for electricity usage
- Relative new to the industry

2.6. Summary of Inertisation Systems

Inertisation has been accepted to have an important place in Australian mining emergency preparedness. The two jet engine exhaust GAG units purchased from Poland by the Queensland government in the late 1990s for the Queensland Mines Rescue Service have been tested and developed and mines made ready for their use in emergency and training exercises. Their use in real and trial mine fire incidents has underlined the need for more information on their application.

The NSW Mineshield (liquefied nitrogen) apparatus dates to the 1980s and has been actively used a number of times particular in goaf heating incidents. The Tomlinson (diesel exhaust) boiler has been purchased by a number of mines and is regularly used as a routine production tool to reduce the time in which a newly sealed goaf has an atmosphere "within the explosive range" and for goaf spontaneous combustion heatings.

Nitrogen Pressure Swing Adsorption (Floxal) units are available and in use both for reducing time in which goafs are "within the explosive range" and for goaf spontaneous combustion heatings. Each of these facilities puts out very different flow rates of inert gases. Each is broadly designed for a different application although there is some overlap in potential usages.

Table 2.3 examines some typical simplified characteristics of the outlet flow of examples of these four units.

Some recent Australian incidents have utilised more than one form of inertisation to stabilise an incident. Table 2.4 lists the systems used at the Dartbrook Colliery 2006 goaf heating.

Table 2.3 Characteristics in simplified form of the outlet flow of the GAG-3A, Mineshield, Tomlinson and Floxal inertisation units.

	Flue Gas[1] Generator (Tomlinson Boiler)	Mineshield[2] Liquid Nitrogen System	GAG unit[3]	Membrane[4] System (AMSA Floxal Unit)
Inert Output Range, m^3/s	0.5	0.2 – 4.0	14 – 25	0.12 – 0.7
Default Quantity, m^3/s	0.5	2.0	20	0.5
Delivery Temperature, ^oC	54	Atmospheric	85	20
Oxygen, %	2	0	0.5	3
Nitrogen, %	81.5	100	80 – 85	97
Carbon Dioxide, %	15.3	-	13 – 16	-
Carbon Monoxide, ppm	0	-	3	-
Water Vapour, %	1.2	-	some	-
Water droplets			significant	

Table 2.4 Systems used at the Dartbrook Colliery 2006 goaf heating (after Sykes and Packham, 2006)

System	Capacity l/s	Installed	Removed
Floxal AMSA 16	120	Pre 19/01/2006	
Floxal AMSA 17	120	Pre 19/01/2006	
Tomlinson 1	300	Pre 19/01/2006	11/02/2006
Tomlinson 2	500	24/01/2006	
Tomlinson 3	500	11/02/2006	
Air Liquide nitrogen	300 (up to 500)	12/02/2006	
BOC nitrogen	300 (up to 1200)	18/02/2006	

1 Tomlinson Boilers, 2004

2 Mines Rescue, NSW, 2007

3 Bell, et al, 1997

4 Sajimon, J. 2005 and AMSA Floxal Unit, 2006

2.7. CONCLUSION

The chapter has examined types of inertisation systems currently available and in use in Australian coal mines for sealing mines or mine sections, for elimination of the potential explosibility of newly sealed goafs, for combating goaf spontaneous combustion heatings or for stabilising fires in high priority locations. Systems have been compared in a number of tables to allow evaluation of the advantages and disadvantages of each approach.

3. SOME ISSUES OF IMPORTANCE IN MINE INERTISATION

3.1. Introduction

Underground mine fires lead to complex interrelationships with airflow in the mine ventilation system. Addition of the gas stream from an inertisation unit adds another level of complexity to the underground atmosphere behaviour. Important questions are raised such as should the main mine fans be turned off so as not to dilute the inert gas or will this action cause, in conjunction with buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion? This chapter examines simulation exercises on mine examples of inertisation usage to focus on a number of issues raised in introduction of the additional gas flow to the mine ventilation network.

3.2. The GAG and the Mine Ventilation Systems

Simulation exercises on the introduction of the GAG or other apparatus to a mine ventilation system have indicated that there is a substantial lack of knowledge on use of these facilities. The Queensland GAG units were first used actively in 1999 at the Blair Athol mine to handle a spontaneous combustion issue in old underground workings that were about to be mined by surface techniques as described by Prebble and Self, 2000. The GAG unit was subsequently used successfully in an underground mine fire at the Loveridge mine, West Virginia in early 2003 (Urosek et al, 2004). On this occasion the GAG ran for approximately 240 hours over 13 days and was successful in stabilising the mine so that rescue teams could enter the mine and seal and fully extinguish the fire affected zone. Much was learnt about the ventilation network behaviour and the need to have an upcast shaft open. Observations were made on the effects of natural ventilation pressure, barometric changes and rock falls on the backpressure experienced by the operating GAG.

A fire which was suspected to have been caused by lightening strike at the Pinnacle mine, also in West Virginia, was out of control from October 2003 to May 2004. A Polish owned GAG unit was successfully used to stabilise the situation although there were a number of underground gas explosions during the course of the incident (Campbell, 2004). Following these experiences the US Micon company has purchased GAG units and has developed a commercial mine emergency and recovery business.

New and innovative approaches to mine recovery are occurring. In the US an equipment unit fire in the Dotiki mine, Kentucky, in early 2004 was stabilised using a Nitrogen and Carbon Dioxide (Wesley et al., 2006). Also in early 2004 carbon dioxide was used to stabilize a goaf spontaneous combustion heating in the West Ridge mine in Utah (Stoltz et al., 2006).

Simulations using the fire simulation software VENTGRAPH can be undertaken to gain better understanding of how inertisation units or systems interact with the complex ventilation behaviour underground during a substantial fire or hating. Aspects worthy of examination include:

- Location of the introduction point for inert gases for high priority fire positions; e.g. portal docking position, special boreholes;
- Size (diameter) of borehole or pipe range required to deliver inert gases and back pressure issues;
- Time required for inertisation output to interact with and extinguish a fire;
- Effects of seam gas on fire behaviour with inertisation present;
- Changes which can be safely made to the ventilation system during inertisation including switching off some or all fans;
- Need for remote controlled underground doors to channel inert gases to the fire location;
- Complications caused by underground booster fans; and
- Spontaneous combustion issues.

3.3. Effects of Positioning of Inertisation Units on Fires and the Mine Ventilation System

An ACARP research project entitled "Mine Fire Simulation in Australian Mines using Computer Software" incorporating a number of mine site exercises to reduce the effects of fire incidents and possible consequent health and safety hazards has been undertaken focused on the application of mine fire simulation software packages for contaminate tracing and fire modelling in coal and metalliferous mines (Gillies, Wala and Wu, 2004). Broad conclusions from work undertaken at individual Australian coal mines are discussed as examples. The effort is built around the introduction of the fire simulation computer program 'VENTGRAPH' to the Australian mining industry and the consequent modelling of fire scenarios in selected different mine layouts.

Generic case studies have been developed to examine usage of inertisation units and particularly application of the GAG jet engine unit. One example has focused on selection of the best surface portal location for placement of the GAG for most efficient suppression of a fire. A second has examined a situation with significant seam gas being emitted on the face. This has shown that under certain face dip angles stopping the mine surface fan to reduce dilution of GAG exhaust gases will cause reversal of face air and consequent mine explosion as gas laden air is drawn across a fire. A third examines inertisation and dilution issues in mains headings. Mains headings present a complex ventilation network with often numerous parallel headings, hundreds of cut-throughs and a variety of ventilation control devices. In such a complex system (with additional interference from a fire), maintaining control of the movement of inert gas is more difficult than elsewhere in the mine. Some illustrations of this issue are given.

3.3.1. Positioning of inertisation units

Studies were carried out to examine usage of interisation tools and particularly application of the GAG jet engine. The best surface portal location placement for the GAG for most efficient suppression of a fire has been examined. Case studies of the typical Australian longwall examples in previous section were modified. A generalised longwall mine layout was used with the length of Mains was set at 2 or 4km. A 1 m diameter borehole was connected to the back of longwall panel about 400m from the longwall panel.

Two GAG jet engine positions were investigated. The first position is at the portal B heading and the second position is at the top of the borehole located at the back of the longwall panel. A diesel fire with a 30m length of fire zone, a fire intensity of 10 and a time constant of 120 seconds is started 50m outbye of the current longwall face was simulated.

Procedures to implement the GAG for both positions are as follows.

1. Start the simulation and let the fire run for 1 hour.
2. Start the GAG after 1 hour and close the emergency door at portal B Heading just outbye the GAG.
3. Shut off the fan and close off the other two emergency doors located at C and D heading.
4. Let the GAG run till the heat production from fire is minimal and the fuel temperature is less than 250^{o}C.

It was found that it made no difference for the second case study GAG position whether the emergency doors at the portal was closed or not.

When the length of the Mains is 2 km, the time it takes to have the GAG put the fire out was similar whether the GAG unit is at the Mains portal or at the top of longwall back borehole. However, when the length of the Mains is increased to 4 km, it was found that a GAG unit located at the back borehole has significant advantage in terms of time in reducing the fire to significantly reduced state (see Figures 3.1 and 3.2).

It should be noted that the advantages can be gained from use of various GAG positions depends on a number of considerations including the location of the fire, the relative distance from the GAG placement portal location and the attributes and complexity of the mine ventilation network Operation of a GAG unit requires preplanning in terms of infrastructure requirements for a GAG surface portal docking station and access for operating personnel, jet fuel, water and other operating requirements.

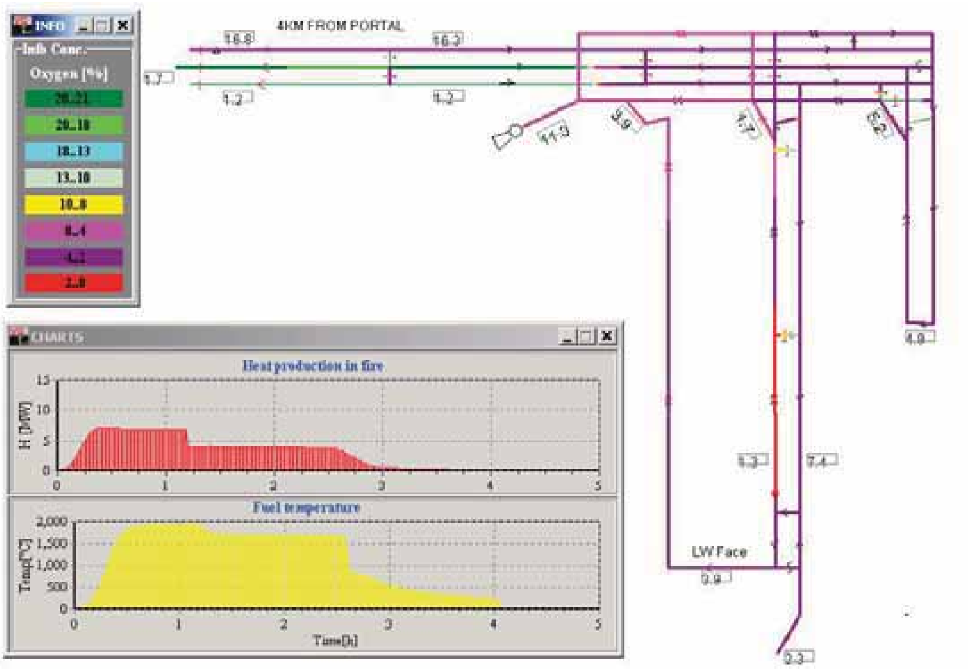


Figure 3.1 GAG position at the portal B heading for 4km Mains length

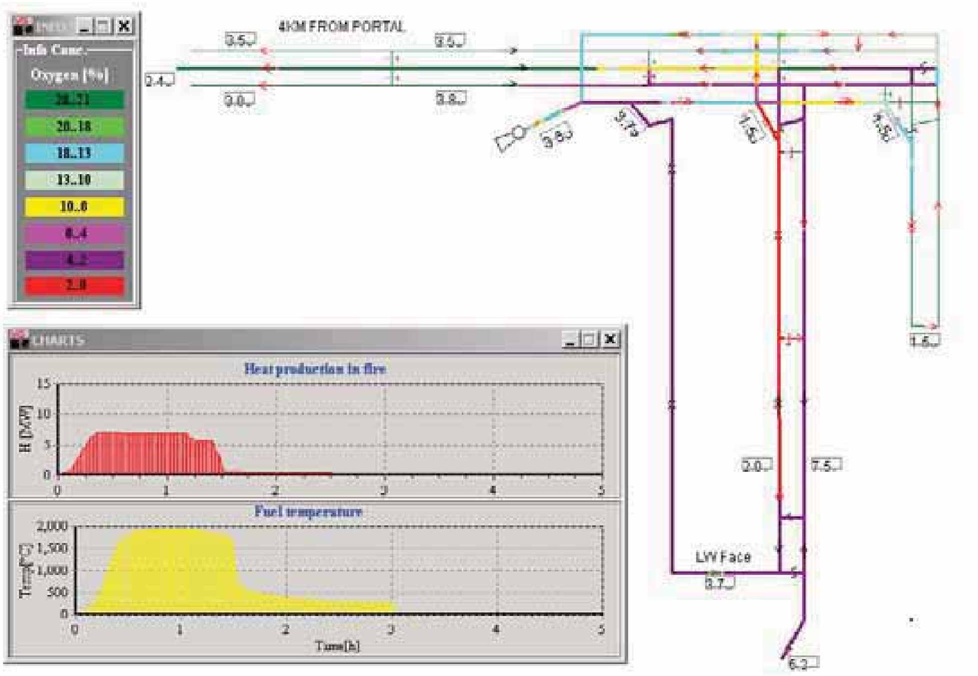


Figure 3.2 GAG position at the top of back longwall borehole for 4km Mains length.

The same conclusion from GAG studies also applies to use of other tools such as the Nitrogen Shield in New South Wales. Any evaluation of this kind requires a detailed study of each mine's ventilation and fire simulation model to identify optimum unit position placement for various fire locations.

3.3.2. Fire with high gas level at face

Investigations were also carried out to examine usage of interisation tools and particularly application of the GAG jet engine in a mine with high gas emission level at the longwall face. Case studies of the typical Australian longwall examples used in previous section were modified. A seam gas face source of CH_4 of 400litres/s was introduced in the middle of the longwall face line in the model to simulate this case. This gives a CH_4 concentration level of about 1% on the return side of the longwall face. In the simulation a diesel fire of 10m length of fire zone, a fire intensity of 10 and a time constant of 120 seconds was started 50m inbye the maingate end of the current longwall face.

The longwall face was examined under two situations of dip angles of 2.5% and 5% (-6 and -12m respectively on a longwall face 240m long) down from maingate to tailgate. This gives descentional ventilation effects as discussed earlier in the paper. The fire in this situation will work against the main ventilation direction along the longwall face. The GAG unit is positioned at the Mains travel road portal B heading.

Procedures to implement the GAG for both positions were as follows.

1. Start the simulation and let the fire run for 1 hour.
2. Start the GAG after 1 hour and close the emergency door at portal B Heading just outbye the GAG.
3. Close off the emergency door located at C, Shut off the fan and then close off the emergency door located at D heading.
4. Let the GAG run till the heat production from fire is minimal and the fuel temperature is less than $250^{o}C$.

It was found that when the longwall is dipping at 2.5%, the GAG unit is successful in reducing the fire to minimal heat production and fuel temperature of less than $500^{o}C$ around 4 hours after the GAG was started as indicated in Figure 3.3. No airflow reversal was observed at the longwall face.

However, when the dipping angle increased to 5% for the same fire situation, as soon as the fan is turned off, the airflow on the longwall face reversed. This leads to the high concentration of face CH_4 flowing back across the fire with high likelihood of an explosion occurring as shown in Figure 3.4. A sharp drop of the heat produced from the fire is observed.

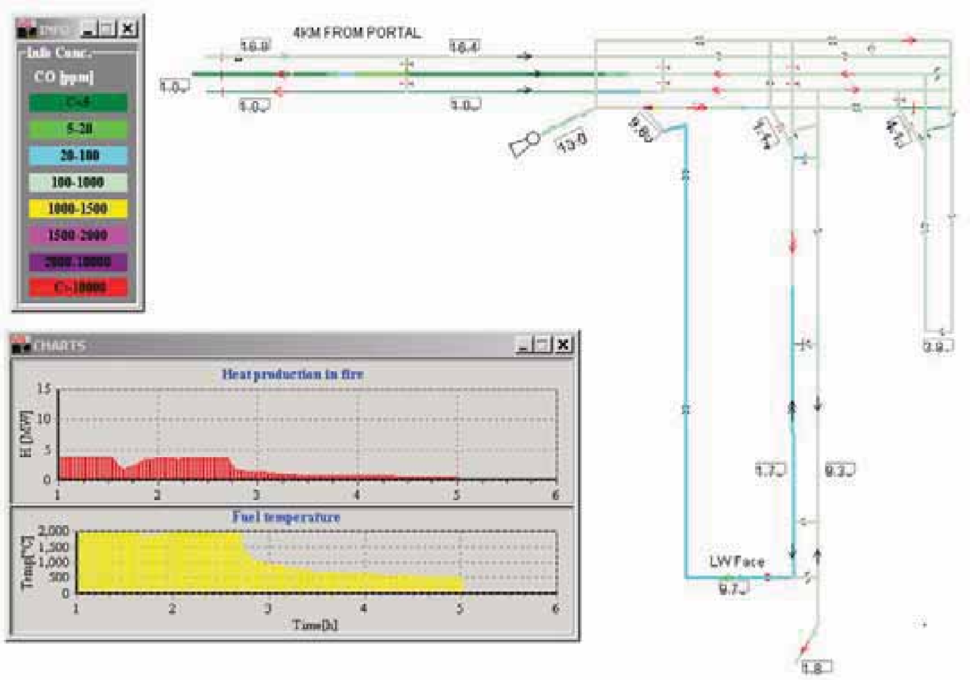


Figure 3.3 Gassy longwall dipping at 2.5% from Maingate to Tailgate.

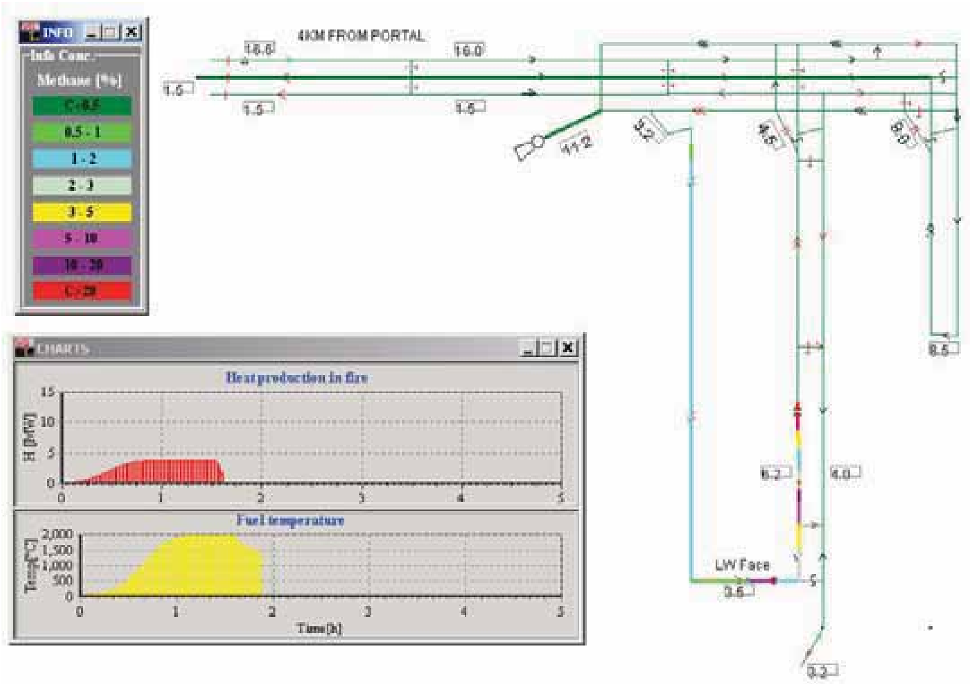


Figure 3.4 Gassy longwall dipping at 5% from Maingate to Tailgate.

As soon as an explosion "occurs" in the VENTGRAPH simulation program, the program will no longer simulate the heat production from fire.

Addition of the inert gas stream adds another level of complexity to the already complicated interrelationships between the mine ventilation system, the presence of seam gases and a mine fire. Should the main mine fans be turned off to reduce dilution of the inert gas, or will this action cause, in conjunction with fire induced buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

3.4. Effective Docking Positioning of Inertisation Units

Positioning of the inertisation units is a major determinant of potential success for most efficient suppression of a specific fire. Traditionally in Queensland docking points have been placed on intake ventilation headings (either travel or conveyor belt roads). Some mines have prepared docking points on boreholes of about 1.0 to 2.0m diameter placed at the back of longwall panels.

The advantages that can be gained from use of various inertisation docking positions depends on a number of considerations including the location of the fire, the relative distance from the inertisation docking portal location and the attributes and complexity of the mine ventilation network. Operation of a GAG unit requires preplanning in terms of infrastructure requirements for a GAG surface portal docking station and access for operating personnel, fuel, water and other operating requirements.

Priority fire locations at mines with VENTGRAPH simulation models developed in an ACARP research project entitled "Mine Fire Simulation in Australian Mines using Computer Software" have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation; in most mines this is at the portal of a Mains travel or belt heading).

A system was derived for categorising mines' principal inertisation docking points as to their ability to inert a priority fire location as set down.

- Category A covers fire in which the inertisation product is directed fully over the fire. No mine priority fire examined achieved the situation in which the simulated fire is directly stabilised to aid recovery in a timely manner.
- Category B covers situations in which the inertisation product goes straight to the fire but there is significant dilution from other ventilation air or leakage through stoppings. Because of dilution stabilisation of a fire through inertisation can only be achieved with some main surface fan changes. 20 percent of mines are in this category and under these situations the fire should, over time, be abated or stabilised to a point where conventional recovery approaches can be initiated.
- Category C covers priority fires in which the GAG output will never reach the fire location without stopping of one or more main surface fans to rebalance ventilation within the pit. In many of these cases requiring fan changes to put GAG output across the fire location

effective ventilation air velocity has been reduced to the extent that local reversal across the fire occurs and fire fumes are pulled across the fire. This is an unsatisfactory situation as fire smoke and fumes can carry combustible products. This situation broadly prevails for 46 percent of the cases examined

- Category D covers priority fires in which the GAG output will never reach the fire location even if surface main fans are altered. These are fire locations within panel sections in which either the fire behaviour stops normal intake ventilation flow into the section headings or the GAG docking point is in an airway that is isolated from the section. This situation is seen in 14 percent of the cases examined.
- Category E covers priority fires in gassy mines in which section production gas make has been included in the simulation modelling. GAG exhaust will never reach the fire location without stopping of one or more main surface fans to rebalance ventilation within the pit. However this change in ventilation causes working section methane and ventilation air (incl. fire fumes) to reverse across the fire. This is clearly a potentially dangerous situation. This situation was found in 20 percent of the cases examined.

A total of 71 potential priority mine fire locations that have had scenarios simulated were reviewed. From these 35 scenarios were considered worthy of incorporating utilisation of the GAG as an exercise in recovery of a mine following a major fire. Table 3.1 shows results of the outcome of the 35 scenarios from the study.

These simulation exercises undertaken with a wide range of Australian mines focuses attention to the situation that many potential underground mine fire sources cannot be successfully inertised with the GAG docked at the current specified point.

This inability to deliver GAG output is particularly so for fires in extended areas of workings or in panels. Two important conclusions are

- Successful delivery of GAG output from units on the surface must consider other (that is alternative to Mains Travel or Conveyor Heading portals) delivery conduits directly into workings near the fire through existing or purpose drilled boreholes.
- During a fire the stopping of the main surface fan or fans will lead to rebalancing of pit ventilation and in some cases potential explosions through air reversals bringing poorly diluted explosible seam gases or fire products across the fire site.

The next section examines some considerations in use of boreholes for delivery of inertisation products.

Table 3.1 Effectiveness of GAG delivery

Code	Description	Results out of 35 scenarios simulated	Percentage %
A	GAG exhaust delivered efficiently (without significant dilution) to fire.	0	0
B	GAG exhaust reaches fire but diluted and not fully effective. Fan change needed to allow inertisation stabilisation of fire.	7	20
C	GAG exhaust reaches fire only after fan change and potentially effective after local reversal of ventilation air (incl. fire fumes) across fire.	16	46
D	GAG exhaust will never reach fire even with fan changes.	5	14
E	GAG exhaust only reaches fire after fan change. Reversal of working section methane and ventilation air (incl. fire fumes) across fire.	7	20

3.5. Inertisation Effectiveness in Mains Heading Fires

Mains headings present a complex ventilation network with five or more parallel headings, numerous cut-throughs and a variety of ventilation control devices. In such a complex system with additional interference from a fire, maintaining control of the movement of inert gas is more difficult than elsewhere in the mine. There is added emphasis in Queensland where most mines have inertisation injection portals (docking stations) connected to Mains entries. At present most Australian collieries have limited control over flow of air in Mains intakes. The quality of segregation stoppings and doors varies greatly between sites. Some states have legislative requirements regarding segregation.

Causes of fires in mains headings include:

- Belt fires (including transfer points and motors)
- Vehicle fires
- Spontaneous combustion in pillars (particularly pillars with large pressures differences across them)

It is not always practical, or safe, to turn off the main fans and flush the mine with inert gas in the event of a fire. Given this limitation, use of segregation can allow fans to be kept on while inert gas is delivered to a particular fire site without dilution and without losing inertising gases in the other airways. On the other hand, without adequate segregation inert gas will spread between all intake airways and be diluted by fresh air. It will also leak to returns.

To determine the impact of the quality of segregation (stopping resistance) on GAG effectiveness in a quantitative manner, Hosking (2004) undertook VENTGRAPH simulations using a fully segregated belt heading with a range of segregation stopping resistance values. The belt way carries intake air and had a regulator placed outbye to reduce airflow and cause leakage flow into it from adjacent intake headings A GAG unit was connected to the beltway drift and run at 11 000 rev/min to give an exhaust stream with an oxygen level less than 5 per cent. The oxygen level found at each cut-through was then measured for each stopping resistance. To keep the scenario simple no doors were included and no fire was actually placed in the drive. The mine fans were kept on throughout the simulations. Existing overcasts in the model were retained and cut-throughs were spaced at about 50 m intervals.

Figure 3.5 shows the results as a set of contour lines for oxygen concentrations. It can be seen that on a log-log plot the dilution rates form a clear relationship with stopping resistance and distance. As would be expected higher resistance segregation stoppings will maintain a reduced atmospheric oxygen content at fixed sensor points in the belt road, and the oxygen content increases with distance from the drift (as the number of leakage paths increases). As the pressures across stoppings are lower further inbye, the leakage rate drops and the contours become steeper.

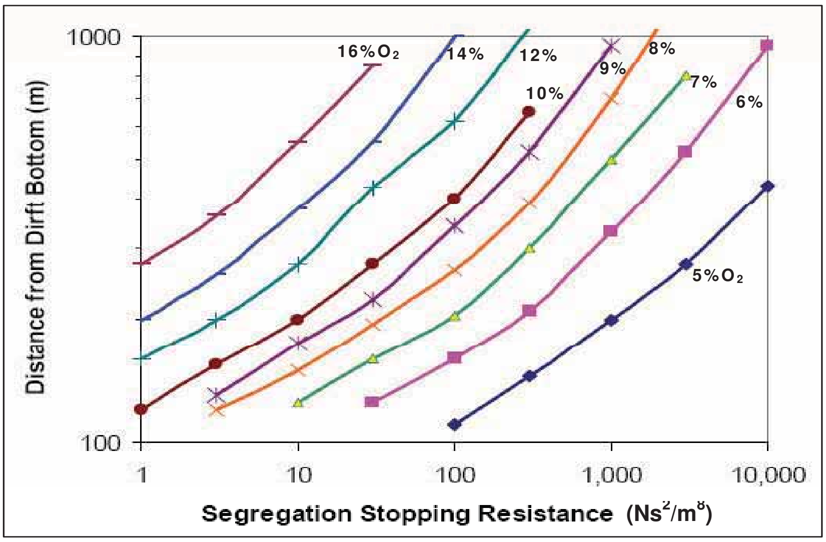


Figure 3.5 Dilution of inert gas at varying segregation qualities and distance

Considering the contour for 10 per cent oxygen (a level below which open flames will not occur), the quality of segregation has a dramatic effect on the range of the inert gas. If flaps/used conveyor belt are used for segregation (resistance less than10 Ns^2/m^8) this concentration of inert gas will only travel 200 m – the first four cut-throughs after the drift bottom. On the other hand a quality stopping that is well maintained (resistance of 100 Ns^2/m^8) will keep the oxygen level at 10 per cent for the first 400 m of the mains.

Figure 3.6 illustrates how effectively the ventilation network can deliver inert gas to a fire at 1.0 km distance. Stopping resistances less than 10 Ns^2/m^8 are unable to stop dilution of the heading air at this distance. Above 10 Ns^2/m^8, the oxygen content steadily declines with higher quality segregation.

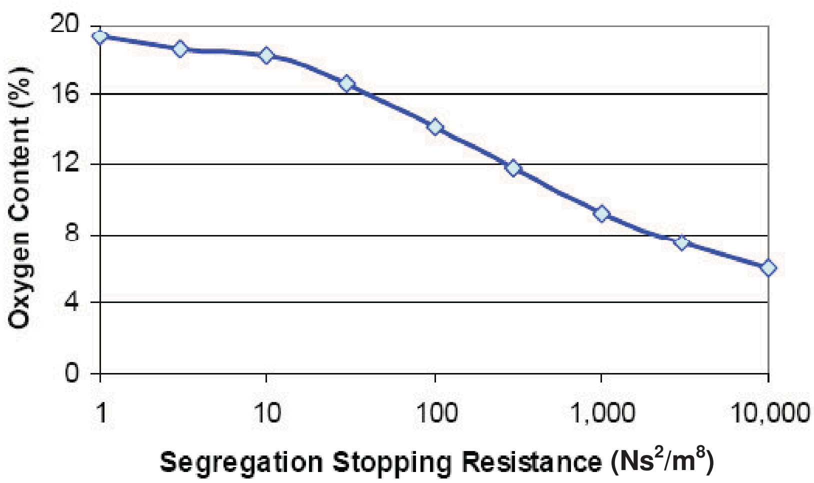


Figure 3.6 Dilution of inert gas at 1.0 km from drift bottom.

These plots are relatively simple to generate for a ventilation network once the model exists. While it may be technically unrealistic or impractical to consider changes to segregation stoppings in an existing mine, diagrams of this form are useful as a planning tool for future developments. Good quality segregation restricts the spread of contaminants (heat, dust and gas on a routine basis and smoke and fire products in an emergency) in addition to assisting the movement of inert gases.

3.6. Inertisation of Highwall Punch Mines and Use of Boreholes

A number of Australian mines have adopted “punch” mine layouts with access to workings through the highwall of a box cut. Many of these have no conventional Mains. Practical options for inertisation of punch mine longwall workings are required. Borehole docking and delivery of inert gases may be required for fires in some sections of the mine if the open cut is not available for GAG action because of

- Geometry of the open cut,
- Open cut road access issues,
- Open cut roads pass in front of open portals,
- Potential of fume build up in the box cut

There is a debate on whether a borehole into punch mine workings should be placed near the front of the mine workings (close to and within a few hundred metres of the highwall portal) or at the back of the mine behind the longwall installation road. These alternatives of use of a front or back borehole have various advantaged and disadvantages that are often mine layout specific.

The use of a back borehole to conventional longwall panels in Queensland is becoming very common. Examples are for instance in use or planned at Crinum, North Goonyella, Moranbah North, Grasstree, Oaky Creek No 1, and Kestrel collieries. Other mines such as Bundoora and Aquila have put in boreholes for potential inertisation use. The punch mines of Newlands North, Broadmeadow and Crinum East have examined the competing merits of use of different location boreholes for inertisation use.

A back borehole in a punch mine can be useful for the following.

- Borehole downcast air can be used at start of extraction of LW panels to ventilate Main Gates if development slows over life of mine and there is no hole through to the next planned panel. It provides a form of ventilation insurance.
- Borehole downcasts clean air that provides some additional ventilation throughout LW panel lives.
- Chilled air can be downcast through the borehole throughout LW panel lives with positional advantage for delivery when longwall face is farthest inbye and often at greatest depth.
- Borehole can be used for services and communication links.
- Borehole delivery of GAG inerts is generally equal to or advantageous (in terms of GAG operating time to inert a fire) for back half of mine compared with docking at front boreholes or highwall portals.
- Borehole can be used for emergency man escape if it is considered too far to walk from back of LW panels to open cut portals.

Front boreholes can be developed earlier than back boreholes. However they do not generally have the positional advantage in relation to providing extra production face air (chilled or normal) or emergency man escape. Efficiency of inert gas delivery through a front borehole will partly depend on mine layout and whether the mine longwall panels are progressing from right to left or left to right. Extra overcasts or remote operation of a VCD door or regulator may be required to direct borehole inert gases to the fire site.

The use of highwall portals for delivery of GAG inerts to longwall panels is the simplest and most direct approach. No extra development of borehole drilling is needed. All new development immediately inbye a new Portal requires this approach for delivery of inert gases until a borehole (if one exists) is holed into. The docking approach is essential for the first part of any new Development headings.

Many punch mines are currently being developed with provision for inertisation docking at both the highwall and boreholes to allow efficient inertisation of fires across a variety of priority locations.

3.7. Conclusions

The potential for simulation of the effects of inertisation on fires within a mine ventilation network was examined. The project involved applying the VENTGRAPH mine fire simulation software to preplan for mine fires. Work undertaken to date at some Australian coal mines is discussed as examples. The effort has been built around the modelling of fire scenarios in selected different mine layouts.

Case studies have been developed to examine usage of the GAG inertisation unit. One section examined seam gas emissions in the face area; addition of the inert gas stream adds another level of complexity to the already complicated interrelationships between the mine ventilation system, the presence of seam gases and a mine fire. Should the main mine fans be turned off to reduce dilution of the inert gas, or will this action cause, in conjunction with fire induced buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

Another section has focused on selection of the surface portal location for placement of the GAG for effective fire suppression. The difficulties that some current approaches present are highlighted. The advantages that can be gained from use of various inertisation docking positions depends on a number of considerations including the location of the fire, the relative distance from the inertisation docking portal location and the attributes and complexity of the mine ventilation network. Operation of a GAG unit requires preplanning in terms of infrastructure requirements for a GAG surface portal docking station and access for operating personnel, fuel, water and other operating requirements.

Priority fire locations at a wide selection of mines with a developed and current Ventgraph simulation model have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation). Many mine layouts were reviewed and from these 35 scenarios were considered appropriate for use of the GAG. These fires were categorised A to E in terms of ability of the GAG exhaust to effectively stabilise and extinguish the fire. As examples of results no fires met the category A description, 14 percent met category D and 20 percent met category E. The conclusion is that the current situation is not well placed to effectively inert most colliery priority fires.

These simulation exercises undertaken with a wide range of Australian mines focused attention to the situation that many potential underground mine fire sources cannot be successfully inertised with the GAG docked at the current specified point. This inability to deliver GAG output is particularly so for fires in extended areas of workings or in panels. Two important conclusions are

- Successful delivery of GAG output from units on the surface must consider other (that is alternative to Mains Travel or Conveyor Heading portals) delivery conduits directly into workings near the fire through existing or purpose drilled boreholes.
- During a fire the stopping of the main surface fan or fans will lead to rebalancing of pit ventilation and in some cases potential explosions through air reversals bringing poorly diluted explosible seam gases or fire products across the fire site.

Another section has looked at inertisation and dilution issues in Mains headings. These present a complex ventilation network and with additional interference from a fire, maintaining control of the movement of inert gas is more difficult than elsewhere in the mine. Even good quality segregation stoppings allow significant dilution of inertisation flows over relatively short distances

A final section has examined considerations presented by “punch” mines layouts. A number of recent punch longwall mines are accessed off highwalls including Broadmeadow, Carborough Downs, Newlands North and Crinum East. These mines have some provision for GAG docking from within the highwall pit but all have put down boreholes to workings which enable the GAG team to operate the engine from the safety of the surface.

4. GAG-3A INERT GAS GENERATOR CALIBRATION EXERCISES

4.1. Introduction

Two validation studies of the mine fire simulation program VENTGRAPH using data gathered from an actual mine fire or other real exercises have been examined. One involved examination of data from a large mine fire and the other use of the GAG-3A inertisation unit as an exercise in making safe old mine workings on mine closure.

4.2. Validation Study of the Mine Fire Simulation Model against Pattiki Mine, Kentucky 1991

Validation studies of the mine fire simulation program VENTGRAPH using data gathered from an actual mine fire which occurred in November 1991 at the Pattiki Mine, Kentucky in US were undertaken by Wala et al (1995). The study evaluated the suitability of the simulation software for modelling an underground fire.

The study simulated a fire as written up in the major event publication listed in the MSHA District #8 Accident Report, 1992. Throughout the fire CO concentration at the mine exhaust fan station was continuously measured and recorded by the mine control and monitoring system and this data stream was used as indicator to compare the VENTGRAPH simulated fire with the real fire.

The simulation processes involved the following steps,

- modelling of the ventilation system prior to the fire,
- modelling of the fire source,
- simulation of the actual fire according to the major event record.

A series of simulations were undertaken using a trial and errors method. The reason for this approach was to obtain a match with the CO data recorded at the fan station and to determine a time sequence for the fire area growth and for parameters which characterise dynamics of fire development. A comparison of the simulated and actual CO levels at fan station is shown in Figure 4.1. The thin solid line represents the simulated CO level and the thick line represents the actual CO level recorded. It can be seen that there is a very close correlation between the simulated and actual CO levels. The time axis of the figure presents real time and descriptions of the particular actions that took place during the fire.

The study shows the VENTGRAPH mine fire simulation model is able to simulate the documented scenario of mine fire at the Pattiki Mine with great confidence. Consequently the calibrated model can be used to perform a number of simulation exercises to test different fire frighting and sealing strategies. The study also shows the importance of the real time atmospheric monitoring system for the validation process.

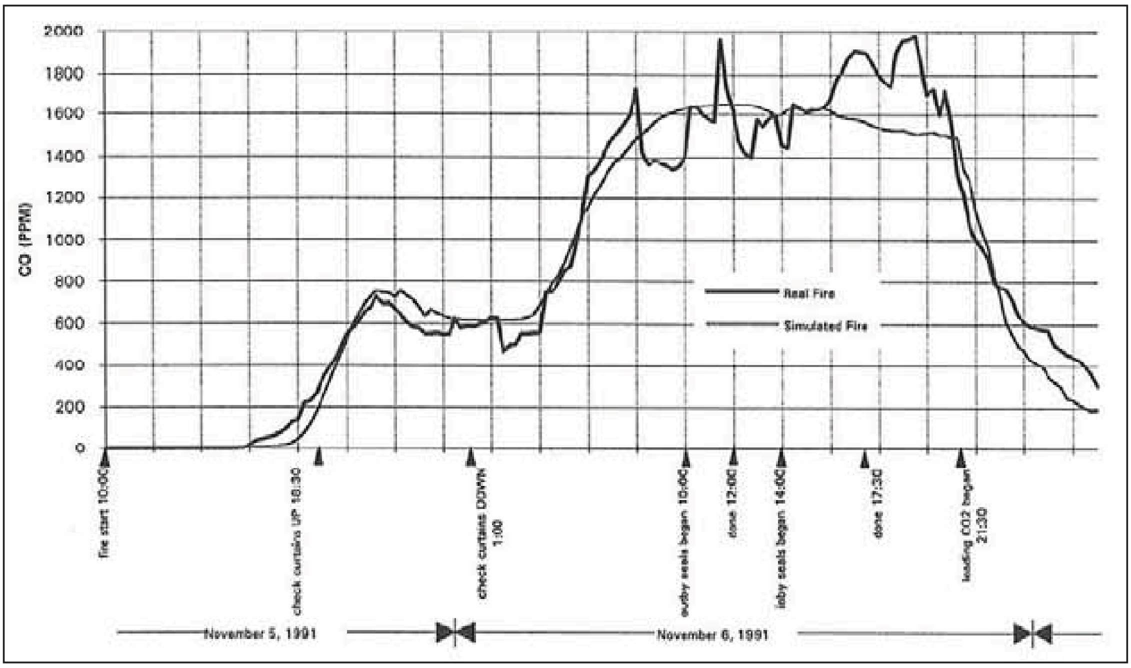


Figure 4.1 Simulated and measured CO levels at Fan Station.

4.3. Validation Study of the Mine Fire Simulation Model against the Newlands Southern Operation, Queensland, 2005

During December 2005 the Xstrata Newlands Southern Underground operation used the Queensland Mine Rescue Service (QMRS) GAG jet engine inert gas generator to inertise its Southern underground workings to reduce the potential risk of an explosive atmosphere after closure of this mine.

The mine operators worked together with the Queensland Mines Rescue Service (QMRS) staff in inertising a section of the mine over a 24 hours period. The trial was also used as a calibration exercise to validate pre-planned VENTGRAPH models and assist with planning for the application of the GAG at the Northern Underground.

The cooperation of Mine Manager Mr David Stone (Stone, 2006) and Mine Technical Services Superintendent Mr John Phillips (Phillips and Hanrahan, 2006) in sharing information and answering questions on the inertisation exercise is acknowledged.

4.3.1. Sealing and re-entry process at Newlands' Southern operation

Figure 4.1 shows the location of the sealing and re-entry strategy used by Newlands' Southern Operation during the inertisation operation.

The strategy focused on reversing mine ventilation by removing the main surface fans and forcing GAG exhaust down what had been the return shaft. Some seals were prepared as shown at the pit bottom area. As the GAG forced air out of the Mains and production panels final seals were completed while airflow and gas concentrations were monitored.

Figure 4.2 Sealing Strategy in Newlands Southern Underground Operation (after Phillips and Hanrahan, 2006).

A policy has been in place of using 350 kPa seals to separate old goafs from Mains, new workings and potential reserves. This had in the past reduced requirements for injecting Tomlinson boiler gas once sealing has been completed. Appropriate risk assessment with sound atmospheric testing and goaf seal integrity testing had allowed the workforce to remain underground at all times.

The system required no major dewatering and U tubes were left open to drain water through the Micon seals and allow flow down dip to a water storage point at the bottom of the mains. This management approach has bee relatively low cost compared to traditional re-enter and re-establish operations with minimal belt, ventilation and other preparation costs relative to recoverable coal reserves.

The arrangement allowed for the retreating salvage of gear required for the start of the new Northern Operation but still provided for the option for the extraction of coal at the Southern in the future.

4.3.2. Start of the Newlands exercise

The main mine fans were turned off at 8:30am on December 15 and ventilation was basically suspended in the mine workings whilst the shaft cap was fitted. The GAG port was fitted to the top of the shaft to allow the GAG hook-up to take place later in the day. The following

Figures 4.2 and 4.3 shows the main fan removal, shaft capping and GAG alignment; as exercise that took less than one hour.

Figure 4.3 Removed Main Fan Dividing Breach on top of shaft.

Figure 4.4 Positioned shaft cap, injection elbow and flexible elbow to GAG and aligning GAG flexible connection.

4.3.3. Seal up strategy

A primary objective was to safely and effectively seal the mine. The GAG provided a means of rapid, safe and effective inertisation. The exercise also provided a valuable training opportunity for Newlands personnel and a commissioning trial for the new GAG trailer without the pressures found in a real emergency.

The pit bottom area was ventilated for this exercise by an 80m^3/s Portal fan located at the top of the old belt drift and the outbye part of the mine was effectively ventilated at the same time. This was used for maintaining ventilation during the sealing process and also enabled continuation of seal construction while the GAG injected exhaust underground.

The mine had VENTSIM and VENTGRAPH models. The Southern Underground's VENTGRAPH model was updated to reflect the most recent VENTSIM model, and scenarios run to simulate the most effective means of inerting the mine. VENTGRAPH modelling indicated it would be possible to reduce the mine's O_2 concentrations to below 8% (an effectively inert atmosphere) in approximately 8 or 9 hours as shown in Figure 4.5.

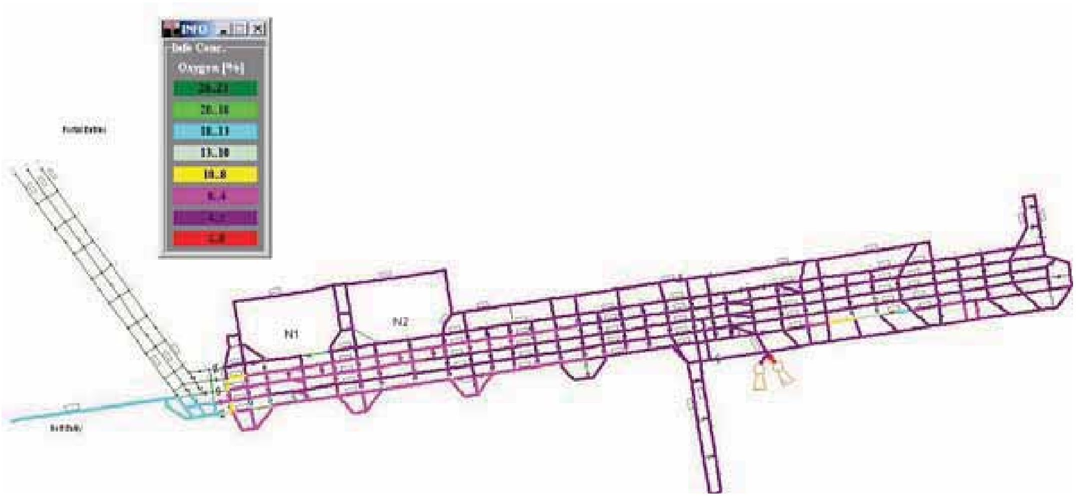


Figure 4.5 Oxygen distribution at 8.5 hours from VENTGRAPH simulation.

A "ventilation change" procedure was prepared with the aim of reversing the mine returns so that the inert gas was directed towards the bottom of the dips (to overcome buoyancy) and then purged out through the mine.

Managing possible CO hazards at the seal construction site was another important consideration. The VENTGRAPH model indicated that the 80m^3/s of auxiliary fan ventilation was capable of safely diluting the expected CO levels as shown in Figure 4.6. Additional gas monitoring and temporary stoppings were installed to manage this risk.

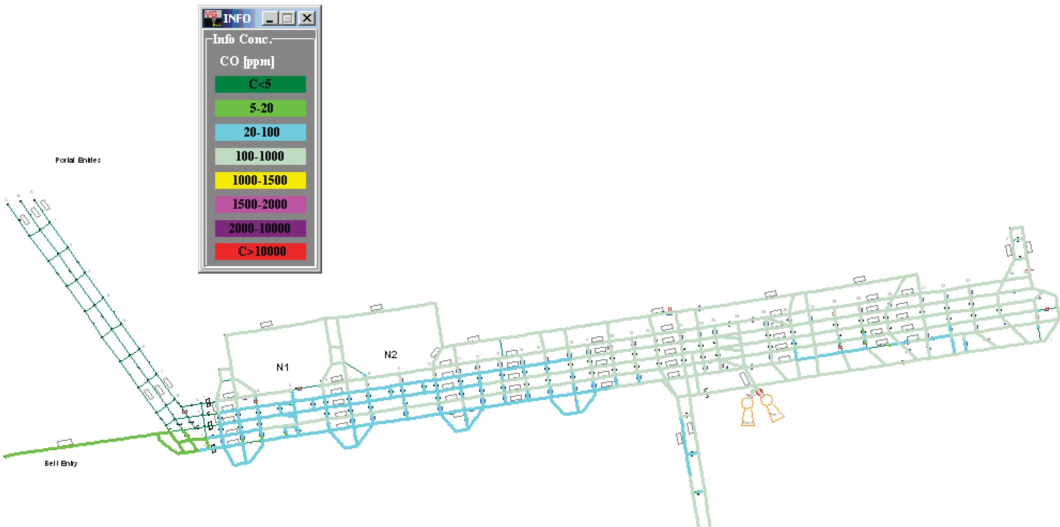


Figure 4.6 CO distribution at 8.5 hours from VENTGRAPH simulation.

4.3.4. GAG operation

The GAG injection process commenced at 4:30pm the same day (December 15) as shown in Figure 4.7 and products of GAG combustion were noted on the mine monitoring system at the bottom of the shaft around 45 minutes later.

Figure 4.7 GAG operation at Newlands South.

Injection continued until around 2:40am on December 16 when the GAG was stopped due to a minor mechanical failure. During GAG injection the quantity of atmosphere exiting the workings at the D heading seal site was measured at around 6.5m^3/s. This perceived low flow was in fact due to a combination of contraction of the exhaust gas due to cooling, the loss of water vapour out of the mixture, the flow of some air and exhaust into the sealed areas due to a barometric high and the effect of turning off the main mine fans. A theoretical analysis of flow behaviour is given in Chapter 10 and data from the Newlands exercise used as a case study.

The door in the temporary seal was closed at 4:30am on December 16 and final sealing continued at that site until the D heading seal was completed at around midday on December 17. At 6:30pm on December 18, the underground atmosphere was showing a gradual increase in methane throughout the workings. At no point had the atmosphere become explosive. The methane concentration had been steadily rising and it appeared if it did become explosive, it might just pass through the bottom corner of the Coward's triangle.

4.3.5. Comparison of predicted and measured GAG outputs

During the operation, underground gas readings were taken with a tube bundle system. They showed that the shaft bottom area inertised within a suitable time frame, but the gas did not migrate throughout the remainder of the mine as predicted (Phillips and Hanrahan, 2006).

Figure 4.8 shows that at tube bundle station 24 at D-E Hdg 24 c/t Shaft Bottom after 8 hrs O_2 level was reduced to 5% compared to the prediction form VENTGRAPH model of 4% and CO level measured was100 ppm compared to predicted levels of 100 to 1000ppm at the same location.

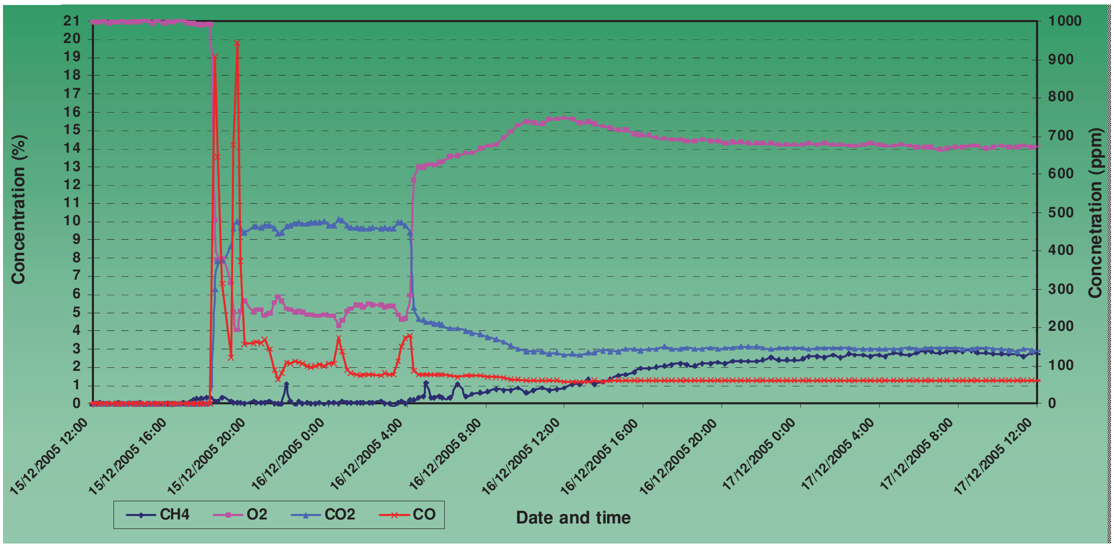


Figure 4.8 GAG outputs from mine monitoring system at D-E Hdg 24 c/t.

After the first two to three hours the gas monitoring system clearly showed that the GAG gas was effectively reporting to the Northern Goafs as shown in the Figure 4.9.

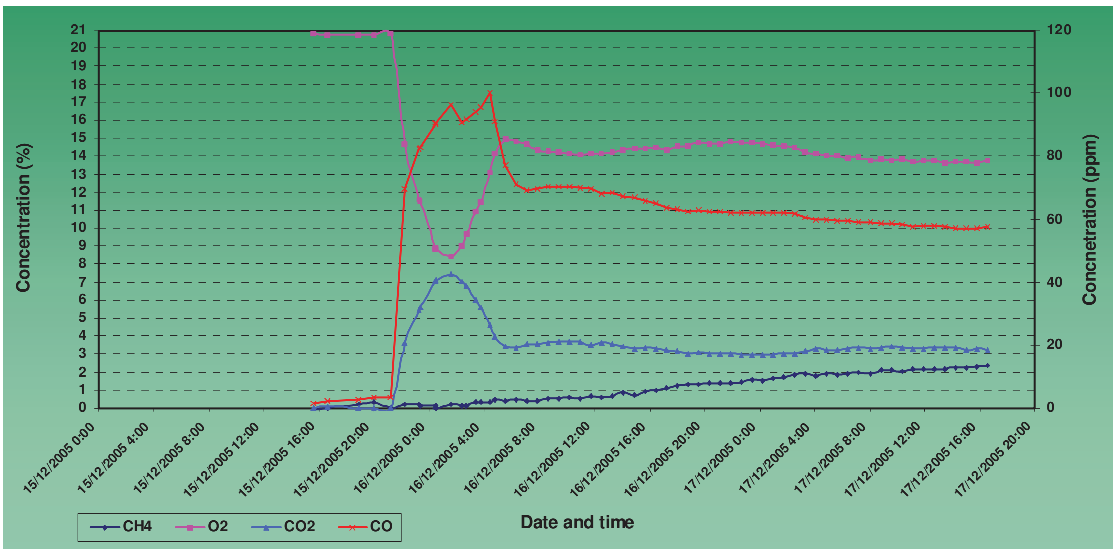


Figure 4.9 GAG outputs from mine monitoring system N1 TG Chute road.

However, the GAG exhaust gas was taking considerable time to report to pit bottom at Mains D Hdg 34 ct as shown in Figure 4.10.

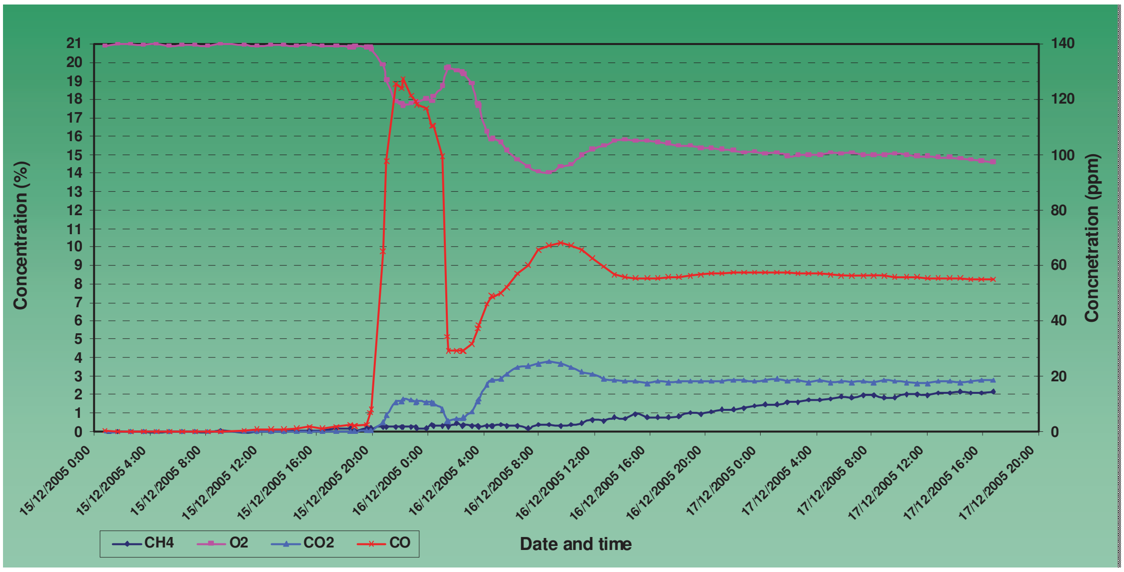


Figure 4.4.10 GAG outputs from mine monitoring system at pit bottom.

Summary of some observations by mine personnel were as follows.

- The resistance of the appliances setup as part of the "ventilation change" could not be measured and validated. Therefore it is difficult to assess the accuracy of the VENTGRAPH model. The injected O_2 levels were 2 to 3 percent higher than the levels modelled in VENTGRAPH.
- At the morning debrief the decision was made to remove a regulator in the northern seal returns as part of the ventilation change. This was a risk-based decision was based on:
  - The barometer was rising steeply and the northern Goafs were prone to oxygen ingress.
  - The Tomlinson boiler could only deliver limited amounts of gas to the area due to requirement to inject elsewhere.
  - The low CH_4 levels in the northern goafs meant the atmosphere could rapidly progress towards an explosive range.
- The product from the GAG did report to the northern returns as expected, however in a higher proportion than expected.
- The output from the GAG was measured a $7m^3/s$ at the Main Dips D reading return. This is inline with conventional literature after taking into account moisture condensation and cooling effects.
- Buoyancy and the small amount of ventilation pressure from the portal fan did act to draw the gas towards the D Heading return as modelled.
- Injecting down the mineshaft did not unduly hinder the GAG.

- The CO levels at the job site were acceptable (despite one scare from a false reading).
- O_2 levels at pit bottom continued to drop after the GAG was switched off and the CO_2 settled to pit bottom.

4.3.6. Issues and findings

The main issues found during the operation identified by Newlands Underground operations manager, Mr Dave Stone (Stone, 2006) were

- cooling of the gas (80^oC to 35^oC) which reduced the quantity of GAG output,
- leakage into the goaf and backpressure from buoyancy of the gas in the shaft.

The manager at this point said he were still undecided whether they would use the GAG method of mine sealing in future. "The use of the GAG appears to be very situational specific. All factors are required to be analysed prior to the use of the GAG. The Northern Underground operation is currently assessing various scenarios for which the GAG would be applicable".

GAG operations are very situation specific. For its use careful consideration of the following is required;

- Time to inertise area,
- Effective dilution rates and flows.

Fighting fires with the GAG may be the most effective usage and best suited to long term resource recovery.

4.4. Conclusions

The back analysis of the gas monitoring data during a fire at the US Pattiki Mine showed that a VENTGRAPH model could be established to simulate satisfactorily this incident. The inertisation exercise during part sealing of the Newlands South workings (without a fire present) highlighted a number of findings.

- The GAG quantity measured exhausting from the mine area being sealed was at first considered to be unrealistically low. However further analysis, as detailed in Chapter 10 of this report, indicates that accounting for temperature and moisture mass changes explains any differences. The GAG jet exhaust (as with any combustion exhaust) puts out a lot of moisture and the cooling water usage adds a lot more. This exhaust product flow mass is lost from the system as it condenses and “wets” the mine workings. Temperature reductions leads to no mass change but “lower” quantity measured.
- The hypothesis that some of the GAG exhaust, with diurnal pressure changes within the workings, will flow into and out of goafs is of interest. This is very likely and

means that both goaf voids should be taken into account in calculating mine excavation volume and that the cyclic pattern of this in and out flow needs to be accounted for.

Further monitoring of mine site GAG exercises are warranted to give greater understanding to this complex system.

5. VENTGRAPH FIRE SIMULATION PACKAGE

5.1. Introduction

The VENGRAPH package is an integrated set of computer programs providing mine ventilation engineers with efficient tools for solution of complex ventilation problems. Its Australian application has been particularly focused to situations with the potential for fires. Difficult and hazardous mine operating conditions necessitate the continuous development of methods of assistance to ventilation service personnel. This contributes to increased work safety and improves economic performance both during normal mining operation and in case of emergency situations such as underground fires or the presence of gas sources (CH_4, CO_2 and N_2) in uncontrolled situations. Interdisciplinary combinations of various branches of science have enabled scientists to formulate a mathematical description of airflow in a complex mining excavation system.

The software focuses on phenomena influencing unsteady processes of flow of air and mixtures of air and CH_4, CO_2 and N_2 or fire gases. The program's graphical capabilities, sufficient computational power and user-friendly interfaces allow a fast and detailed interpretation of calculations.

The concept of the VENTGRAPH ventilation engineering software system was developed in the late 1980s. The presentation method of information about a ventilation network adopted in VENTGRAPH software requires prior preparation of two types of data. These are data about the network structure and the three-dimensional diagram of the ventilation network presented on the screen. This software, which was originally based on a DOS platform, was imported into a WINDOWS environment in 2002.

The VENTGRAPH package is a tool that may assist Ventilation Engineers and others in design and maintenance of ventilation networks including hazard prevention and suppression. Applications include

- Constant control of the non-stationary ventilation process,
- Computer assisted detection of dangers in the mine ventilation network,
- Forecasting of new ventilation arrangements,
- Reconstruction of the non-stationary ventilation process particularly after occurrence of disasters in the mining network (fire, discharge of methane etc).

The package uses opportunities offered by the graphical multitask environment MS WINDOWS, including:

- Enhanced user interface of the software based on system of windows with active boxes which facilitates the operation of this type of programs and is relatively user-friendly in the calculating process,

- Possibilities of using any printers, plotters, monitors and video cards that are WINDOWS compatible,
- Use of full capabilities of editors at any time during the running of a simulation exercises,
- Addition of other newly developed ventilation engineering software that makes use of a common database such as gas sensors in the monitoring system.

5.2. VENTGRAPH Package

The basic VENTGRAPH package consists of the following programs with features described below:

5.2.1. Input database and ventilation network graphic diagram

EDTXT A special full-screen text editor has been developed for inputting data. This enables preparation of data and supports some basic calculations including verification of the correctness of network structures, calculation of resistance values of branches on the basis of measurements conducted in the ventilation network and the approximation of characteristics of fans. This division of data into groups related to branches, measurements at nodes and data for fans allows convenient and fast inputting of large quantities of measurement data.

EDRYS A graphic editor that is designed for drawing three-dimensional diagrams. Knowing the structure of connections between branches, it is possible to draw a 3-D diagram of a multi-level, three-dimensional network. Each branch and node are assigned with selected flow parameters, including air velocity, static pressure, pressure loss and other parameters, such as the node's position with regard to depth, pressure, potential variation from the isentropic pressure distribution. These parameters can be displayed on the screen. Sometimes it is necessary to draw diagrams of selected network regions or simplified diagrams, where a selection of branches is replaced by one equivalent branch. Using the computer keyboard, mouse or digitiser diagrams consisting of branches, nodes, information boxes, symbols of fans, dams, arrows showing the flow direction and boxes with data for individual branches can be drawn.

5.2.2. Simulation software – steady state

Air parameters in the situation of stationary distribution in a ventilation network are constant for a given place in a branch. Taking advantage of this fact, we can restrict the presentation of results on the screen to results in a numerical form displayed next to branch symbols. However, the basic problem is the large amount of results rather than the method of their presentation. An extensive ventilation network in a mine may include several hundred branches; therefore a legible presentation of the whole system on the screen together with results of calculations is not feasible. The solution presented offers a possibility to display the network at any scale and to show any marked part of the network. Calculations of parameters in branches are placed in rectangular boxes adjacent to drawings of branches. It is also possible to present characteristic parameters for network nodes (pressure, elevation, potential). Documentation of the computing conducted is produced both in a traditional tabular form and graphic one (Dziurzyński, et al, 1988).

GRAS GRAS allows for calculation of the steady air distribution in a mining excavation network in normal and emergency conditions. This can be achieved due to the following features of this software:

- The possibility of changing resistance values of selected mining excavations;
- The possibility of building a constriction for a specific resistance value;
- The possibility of introducing air pressure drops due to a fire;
- The possibility of changing the type of branch, including:
  a) a normal branch, without any change in parameters;
  b) a branch with a fan, with the possibility to change the characteristics of the fan;
  c) air supply, change in the flow rate;
  d) methane supply, change in the flow rate;
  e) regulator (fixed flow branch), the calculation of ventilation door resistance or changed operating conditions of a fan.

5.2.3. Simulation software – the unsteady state

Transients generated by fires or outbursts of gas and rocks in a ventilation network lead to a complex distribution of parameters which vary both in time and space. The presentation of this type of distribution requires substantial resources. In simulation software of this type of phenomena a range of 32 colours available in PCs are used. In contrast to computing in stationary states, information about individual colours for presentation is displayed (e.g. distribution of oxygen concentration levels in fire gases). This solution allows the observation of fluctuations in this distribution during simulation. Another advantage is the possibility of obtaining time diagrams of observed parameters in selected network points. The application of a solution with a legible colour screen makes the interpretation of phenomena occurring in the network much easier and assists decision-making during simulated rescue actions.

FIRE FIRE allows the simulations of unsteady distribution of air and gases in a mining excavation network after occurrence of an underground fire. The fire can be simulated by:

- calculating the flow rate of air and fire gases in each ventilation route;
- identification of the current depression of fans, thermal depression and natural depression;
- calculation of temperature as a function of time and location;
- computing the propagation of gases and concentration levels of individual gases as a function of time and location;

- calculation of time and zone where reversion occurs.

The capabilities of this software listed above enable the prediction of possible effects on the use of various fire fighting tactics or elimination of the potential danger might be faced by various working areas. Multi-variant simulations allow responses of the ventilation network during a fire before a real danger occurs. This is the basis of prevention training of mine ventilation service personnel.

5.3. Preparation of Data

The value of a computer model of a ventilation system in a mine is dependent on data about its structure and accuracy of measurements of ventilation parameters in branches and at fans.

5.3.1. Representation of the ventilation network diagram

It is possible to draw a 3-D diagram of a multi-level, three-dimensional network knowing the structure of connections between branches, Each branch and node can be assigned with selected parameters of flow (flow rate, static pressure) or other values such as a node elevation or branch length.

VENTGRAPH system includes EDRYS, a special graphic editor designed for drawing three-dimensional diagrams. It is possible to draw a diagram consisting of branches, nodes, information boxes, symbols of fans, dams, arrows showing the flow's direction and boxes where data for individual branches are displayed. EDRYS offers also a possibility of assigning sensors of the monitoring system to the three-dimensional diagram.

5.3.2. Ventilation measurements

To prepare an accurate model it is necessary to conduct what is generally referred to as a “Pressure Quantity” survey in a mine. The measuring procedure is performed by the specialists teams equipped with instruments for measurement of air velocity, static pressure and temperature measured by a dry and wet bulb thermometer. Accuracy is needed while measuring the barometric pressure and the flow rate (measured as an average flow rate across the mining excavation diameter). It is necessary to know the average cross sectional area of branches, lengths of branches and elevations of nodes.

The results of measurements and data about the structure (i.e. connections between branches) are put in by means of EDTXT, a full-screen network data editor. In addition, this software enables verification of the correctness of the structure and supporting calculations. Other programs of the VENTGRAPH system use databases prepared by EDTXT.

5.4. VentInter Conversion Program

A conversion program, VentInter, has been developed by the authors of this report to convert a VENTSIM data file into 3 VENTGRAPH data files, namely, DT1, DT3 and RS0. The program is written in Visual C^{++} and its main purpose is to build a window interface so that the ventilation data can be imported from VENTSIM into VENTGRAPH. It contains one interface window that requires the user to select the VENTSIM model file and its location as the source and to generate the three VENTGRAPH data files at the same directory at default as shown in Figure 5.1.

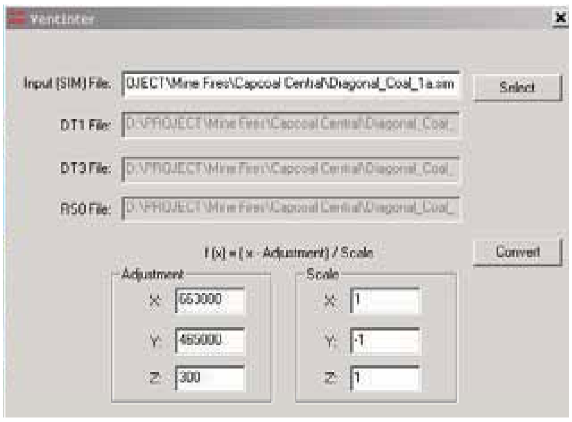


Figure 5.1 Interface window in the VentInter conversion program.

It allows the user to input adjustment factors to convert the mine X, Y and Z coordinates into appropriate screen true coordinates so the network model can be fitted into the VENTGRAPH screen display for easier manipulations. Scales factors are also included in the program so that depending on the network model size; the user can re-size the model to fit the display area. By using negative numbers, the model can be flipped in 3-D so the user can determine the best viewing setting for the model.

5.5. New Inert Unit and Gas Source Functions

Under this ACARP supported project the Polish program authors kindly undertook inertisation related modification to the VENTGRAPH program from the project findings. The modifications are as follows.

1. New inertisation units. The original program only allowed use of a GAG form of inertisation unit within the ventilation network. The modified version includes additional units. An additional pull down menu “Inert Units” has been included. Under the “Inert Units” submenu a selection of inertisation units namely GAG, Mineshield, Tomlinson Boiler and Membrane Filter can be included. Once the unit or system is selected, a pop up window shows up with the operating parameters and outputs parameters displayed. The users can then accept the default parameters or opt for changes if the operating or output parameters are different from the default parameters.

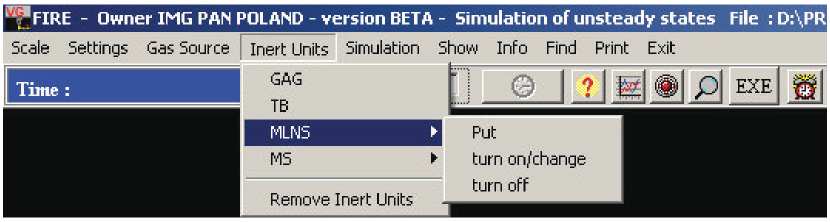


Figure 5.2 New Inert Units Pull down Manu in VENTGRAPH

2. Including extra mine gas sources. The original VENTGRAPH version only allowed the seam gas of CH_4 to be included. The new version allows in addition CO_2 and N_2 to be placed in the model. A new pull down menu facilitates this.

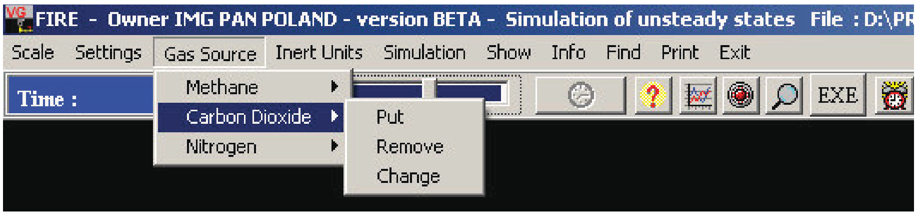


Figure 5.3 New Gas Source Pull down Manu in VENTGRAPH

As part of the ACARP funded project a Beta version of the new form of VENTGRAPH was created and tested. The new additions were trialled and tested on various mines' VENTGRAPM models and as a result and a list of suggested improvements to the Beta version was sent to the authors of VENTGRAPH for consideration. All recommendations were accepted and VENTGRAPH is now significantly more useful for use in a mine inertisation study.

5.6. Application of VENTGRAPH to Mines

The introduction of a fire simulation computer program relies on the modelling of fire scenarios on accurate mine layouts. It is most important for running VENTGRAPH that the mine has an accurate, timely, and calibrated ventilation network model such as the Australian VENTSIM software or equivalent. It is necessary to have the VENTSIM model of the mine being studied in an appropriate state to act as a foundation to build mine fire simulation models. Basically the VENTSIM model must be up to date, calibrated and incorporate depth coordinates. It is recommended that as a minimum the model's calibration be checked. It is assumed that mine planning software data and mine survey measurements are available (AutoCAD style DXF format) in readily transferable form. A small amount of additional extra data for fire simulation modelling also needs to be collected. In particular relevant information on underground heat sources and air temperatures may be needed.

Once an updated VENTSIM model from the mine is established the model can be transferred to VENTGRAPH graphic model. This has been undertaken using the VentInter conversion program developed for transfer and building the five data files required for VENTGRAPH. The major time factor here is constructing and validating the file which controls the program graphics interface. Building the graphics file from scratch would be most inefficient.

Mine fire scenarios based on fire sources common to all mines have been simulated. Ventilation engineers or other engineering and safety staff at each mine involved were given an overview of how to use the VENTGRAPH program. The teaching effort emphasised to mine staff the need to have an accurate VENTSIM model in a form to serve as the base for initiating the fire simulation process. Information was given on how to undertake file transfer to VENTGRAPH so that at a later stage as the mine progresses a new VENTSIM model can be transferred. Experiences were shared on how to undertake fire simulations.

Fire simulation programs rely on the modelling of fire scenarios across actual mine layouts. The exercises undertook simulations of the effects of common open fire causes and fire progress and intensity rates. Examples studied across various locations included fires initiated by for instance vehicles, stationary installations (fuel containers, transformers, equipment containing hydraulic fluids, etc) gas sources and conveyors. Considerable effort went into correct modelling of the thermodynamics characteristics of the various fire sources modelled. Fires from vehicles, fuel, belting coal and so on all develop at different rates and different intensities. Much work had been done to calibrate fire types and this was used to give “best representation”. As far as possible these simulation scenarios were developed working directly alongside mine Ventilation Officers and other technical staff.

Some simulations of safe escape scenarios from a pit affected by fire as part of emergency evacuation were also undertaken. These simulations involved for instance installation of temporary stoppings, removal of stoppings or doors, changes to mine fan duty and possibly use of interisation. These were done with mine staff principally based on the scenarios the mine already has in place in its emergency evacuation plan. Fires and application of inertisation scenarios developed during the studies are described and discussed in the following chapters 6 to 9.

5.7. Conclusions

This section has given a brief overview of the VENTGRAPH simulation software. It has highlighted the new features that have been added to the software as a consequence of this inertisation project and in particular the ability to use up to four different types of inertisation gases (at varying flow rates) across a mine layout simultaneously and the ability to include carbon dioxide and nitrogen seam gases as well as methane.

6. CASE STUDIES OF FIRE SCENARIOS AT OAKY NORTH MINE

6.1. Introduction

Scenarios developed for Oaky North Colliery have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage following a fire.

A total of five scenarios were simulated for Oaky North Colliery based on the mine fire simulation model developed from the ventilation arrangements in July and October 2005 as shown in Figure 6.1. These scenarios are as follows.

1. Belt Fire in Mains C Hdg conveyor at 39ct.
2. LW goaf fire. Spontaneous Combustion in goaf behind South Longwall 3 face currently at 15ct. Spon comb potentially spread over 600m (modelled by development of spider web arrangement.
3. Belt Fire in South LW 4 MG 22CT Tripper drive.
4. Belt tripper drive Mains 11ct C Hdg.
5. Dev in 7 MG at 26 ct (100m pillar). Eimco vehicle fire at 500m outbye of the face. Face 2.2 m^3/tonne CH_4.

Each of these three scenarios are described and discussed in the following section.

6.2. Scenarios for Xstrata – Oaky North Colliery

Exploration at Oaky Creek began in 1977. MIM acquired its majority stake in the Oaky Creek project lease in 1981, becoming project manager, and the mine was officially opened in 1983. Initially an open cut dragline operation, underground mining was later introduced to increase coal production as the open cut mine became deeper and stripping ratios increased.

As part of the Oaky Creek project, the Oaky North underground mine was developed from 1995. Oaky North Longwall underground mine went into production in 1998 at a cost of $213 million. A move to more economical coal recovery through the two underground operations saw the wind up and closure of the open cut operation during 1999. However, changes in exchange rates and coal prices saw the open cut operation become economical again and reopen with two draglines in the high quality Aquila seam in July 2001.

Developed as a major longwall mine, Oaky North is generally a larger scale operation than the Oaky No 1 mine, with a greater seam thickness, wider longwall blocks and bigger more powerful mining equipment. In Oaky North, continuous miners develop the blocks for longwall extraction maintaining minimum developed longwall inventories of at least two blocks at all times. It has generally good working conditions with stable roof, floor and coalface and an above average seam thickness.

Xstrata acquired the MIM operations in mid 2003.

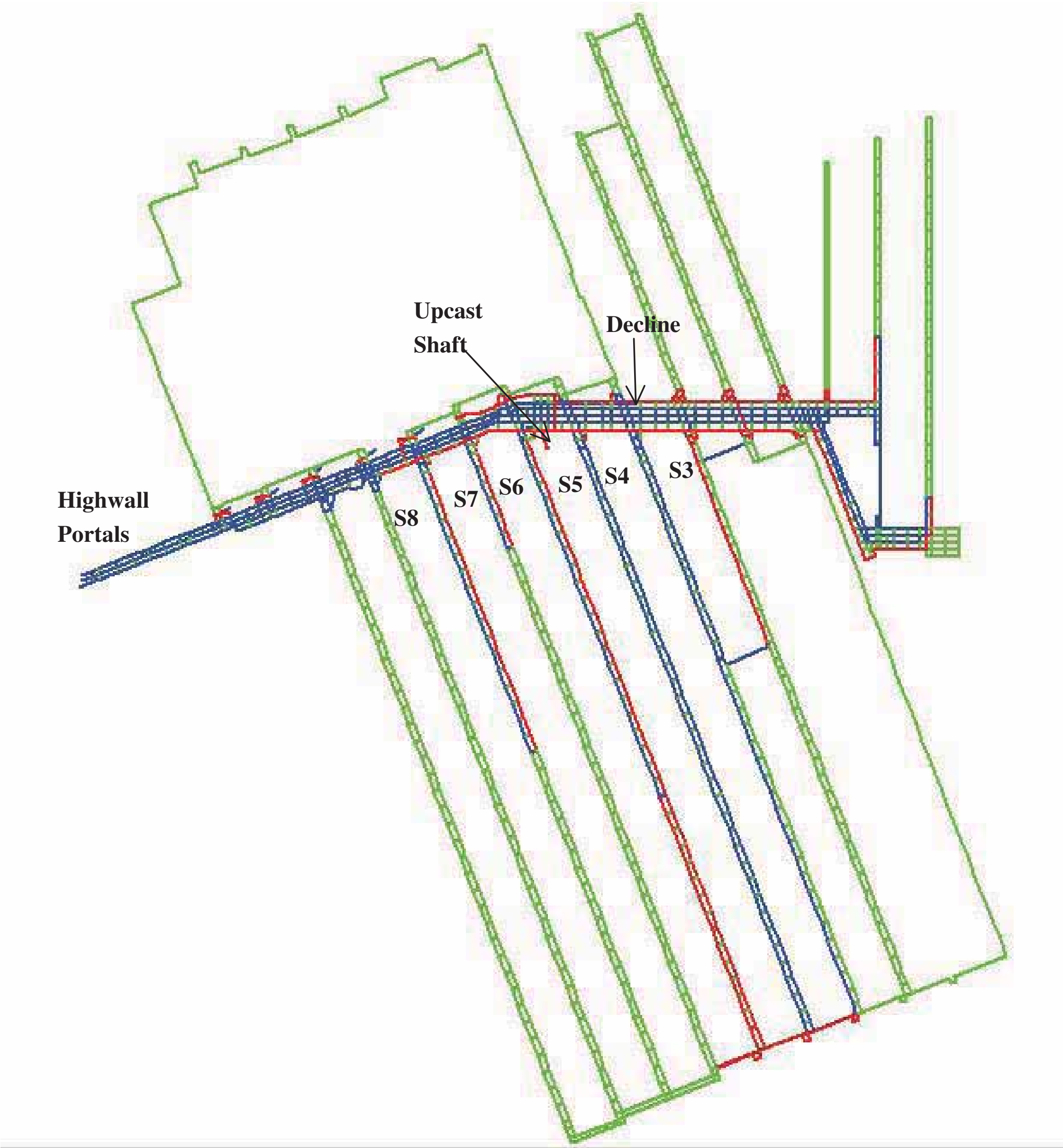


Figure 6.1 Ventilation arrangements at Oaky North Colliery in October 2005

6.3. Oaky North Fire Scenario 1

Scenario: *Belt Fire in Mains C Hdg conveyor at 39ct*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s

Gas emission levels from measurements in mine November 2005.

CH_4 source of 600 litres/s from LW goaf at TG end of face

Prior to running fire simulation pre-enter some of the controls that may be required e.g.

- CO Gas sensors set at points before and after fire, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.
  CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 15 minutes: 30 litres oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10 CO: CO2 = 0.1. (assume H_2 = CO level).

Step 2 Time 15 - 30 minutes: 230 litres oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Step 3 Time 30 – 120 minutes: Oil consumed. 50 m entry length coal develops. time constant 14,400s, intensity 5.

Control: At 120 minutes decision made to introduce high flow inertisation – GAG

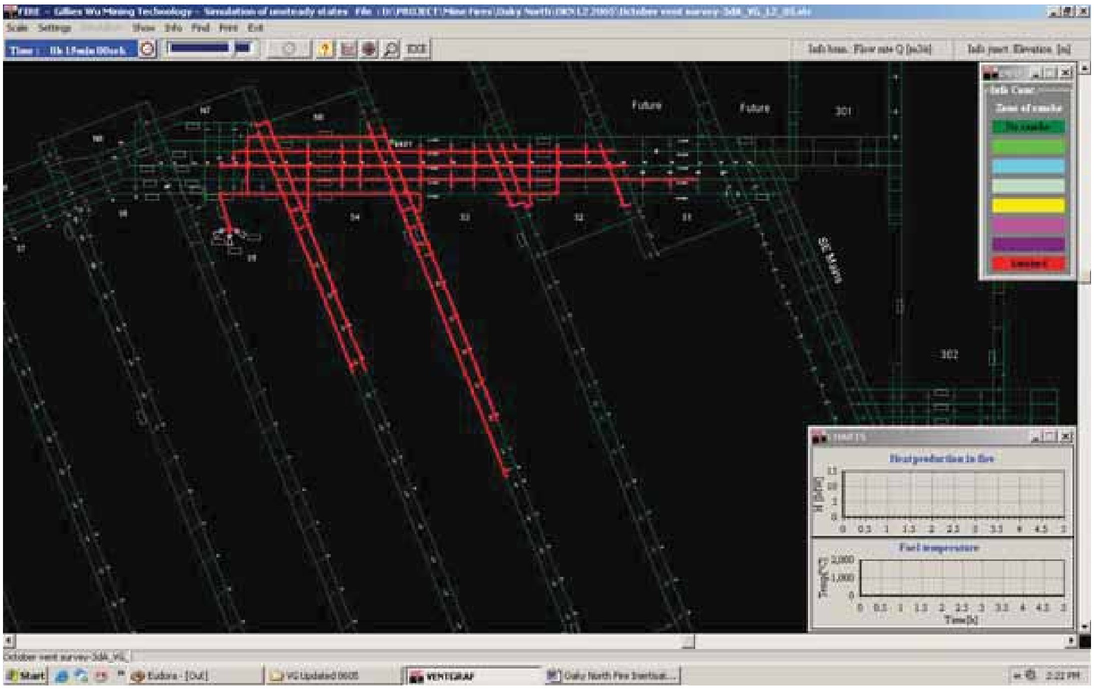


Figure 6.2 Smoke distribution after 15 minutes

Step 4 Time 120 – 300 minutes: 100 m entry length coal fire develops. time constant 14,400s, intensity 5.

Step 5 300 minutes: Segregation devices (e.g. prep seals, brattice, remote controlled doors or manually controlled doors) installed. Segregation for pit bottom required at the following points. For delivering into

B Hdg

- Close machine door at 37ct B – C
- Prep seal B Hdg 35 – 36

C Hdg

- Prep seal at B Hdg 35-36
- Open machine door at 37ct B-C
- Brattice around belt structure C Hdg 36 – 37ct
- Brattice around belt structure 37ct C - D
- Prep seal at B Hdg 37-37A ct

D Hdg Must turn at least one main fan off

- Close B Hdg, C Hdg & D Hdg, 35 – 36ct
- Close B Hdg & C Hdg 37 - 37A ct
- Open machine door at 37ct B-C

Step 6 Time 300 minutes: GAG has been set up at the Intake Drift Close emergency door R=10; Start GAG

Examine all three main fan curve operating points; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 330 minutes: Shut down No 1 fan; fan louvre doors closed R=20
Examine No 2 and No 3 fan curve operating points
Reversal of air in pit has occurred
Fire unstable and erratic local air reversals over fire.

Step 8 390 minutes: install brattice stopping in Belt drift C heading R = 1

Step 9 420 minutes: Shut down No 2 fan; fan louvre doors closed R=20

Examine No 3 fan curve operating point

Step 10 450 minutes: Close Mains Portal D Heading Emergency Door R = 20

Step 11 480 minutes: Shut down No 3 fan

Step 12 540 minutes: Close Main Portal B Heading Emergency Door R = 20

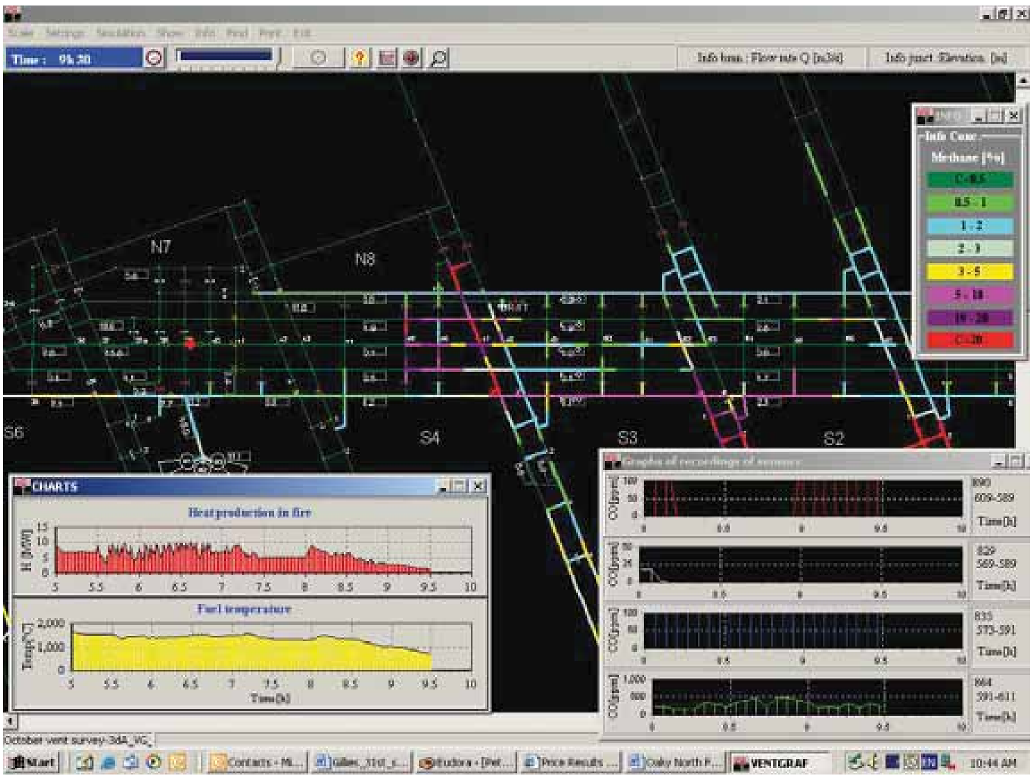


Figure 6.3 Methane distribution after 570 minutes

Summary *With GAG running Fire intensity insignificant at 10 hours and oxygen level outbye fire at less than 2.9 percent.*

6.4. Oaky North Fire Inertisation Scenario 2

Scenario: LW goaf fire - analysis of goaf needing understanding of caved material permeability and methane emissions. Spontaneous combustion in goaf behind South Longwall 3 face currently at 15ct. Spontaneous combustion potentially spread over 600m.

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
- Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO Gas sensors set at points either side of bottom of Ventilation Shaft.
- CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- O_2 sensor in the LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 360 minutes: 1 m entry length coal fuel in 18 c/t MG edge of goaf burning; time constant 14400s, intensity 1 $CO:CO_2$ = 0.1. (assume H_2 = CO level); fire very unstable and not under control.

Step 2 Time 360– 720 minutes: 5 m entry length coal burning with gas continuing to burn; time constant 14400s, intensity 2.

Step 3 Time 720 – 1080 minutes: Continue coal fire 25 m entry length coal burning; time constant 14400s, intensity 4.

Step 4 Time 1080 - 1440 minutes: Continue coal fire 100 m entry length coal burning; time constant 14400s, intensity 8. Fire very unstable and not under control.

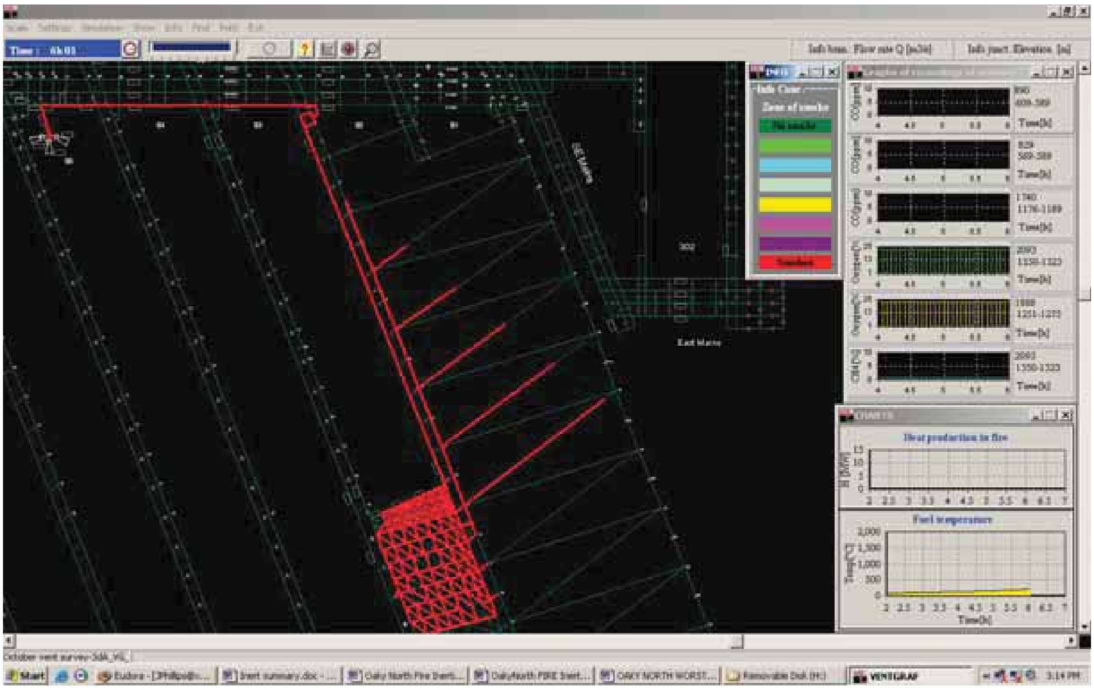


Figure 6.4 Smoke distribution after 360 minutes

CO concentration at 19 hours sets off alarm at bottom of vent shaft.

Step 5 Time 1440 - 1800 minutes: Continue coal fire 200 m entry length coal burning; time constant 14400s, intensity 10.

Step 6 Time 1440 minutes: GAG has been set up at the Intake Drift Close emergency door R=10; Examine all three main fan curve operating points; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 1500 minutes: Shut down No 1 fan; fan louvre doors closed R=20
Examine No 2 and No3 fan curve operating points

Control Assess effectiveness of GAG

Step 8 1560 minutes: Install brattice stoppings Belt drift C heading R = 1

Step 9 1590 minutes: Shut down No 2 fan; fan louvre doors closed R=20
Examine No 3 fan curve operating point

Step 10 1620 minutes: Close Main Portal D Heading Emergency Door R = 20

Step 11 1680 minutes: Shut down No 3 fan. Close main portal B Emergency Door R=20

At 34 hours local reversal occurred to produce a minor methane burn-off.

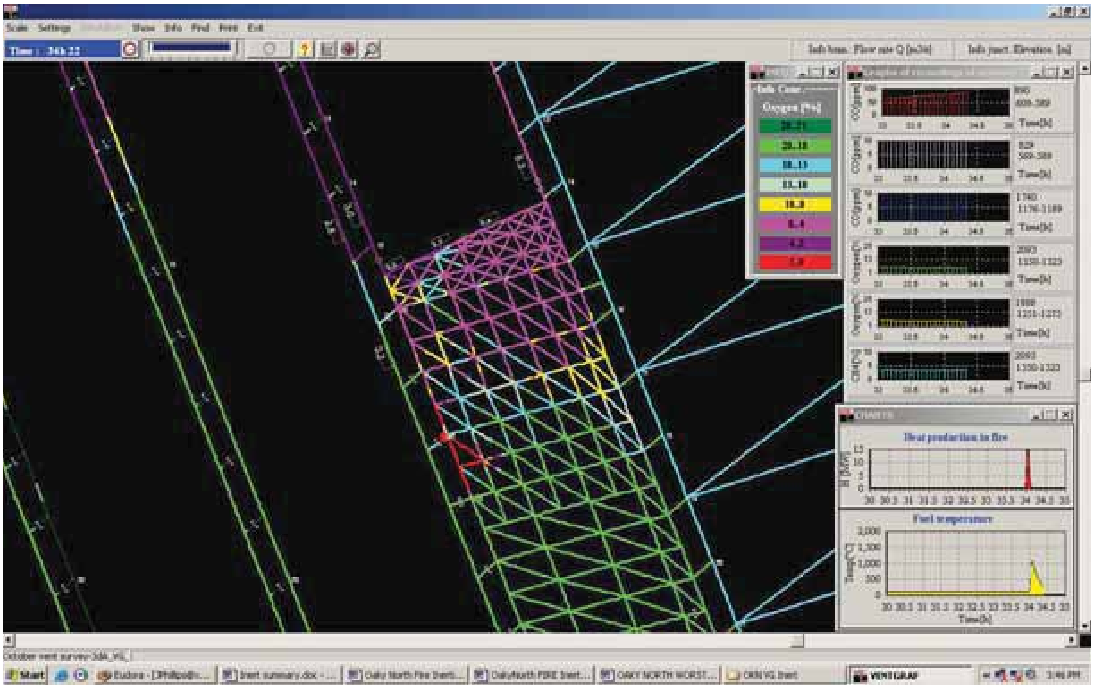


Figure 6.5 Oxygen distribution after 7200 minutes

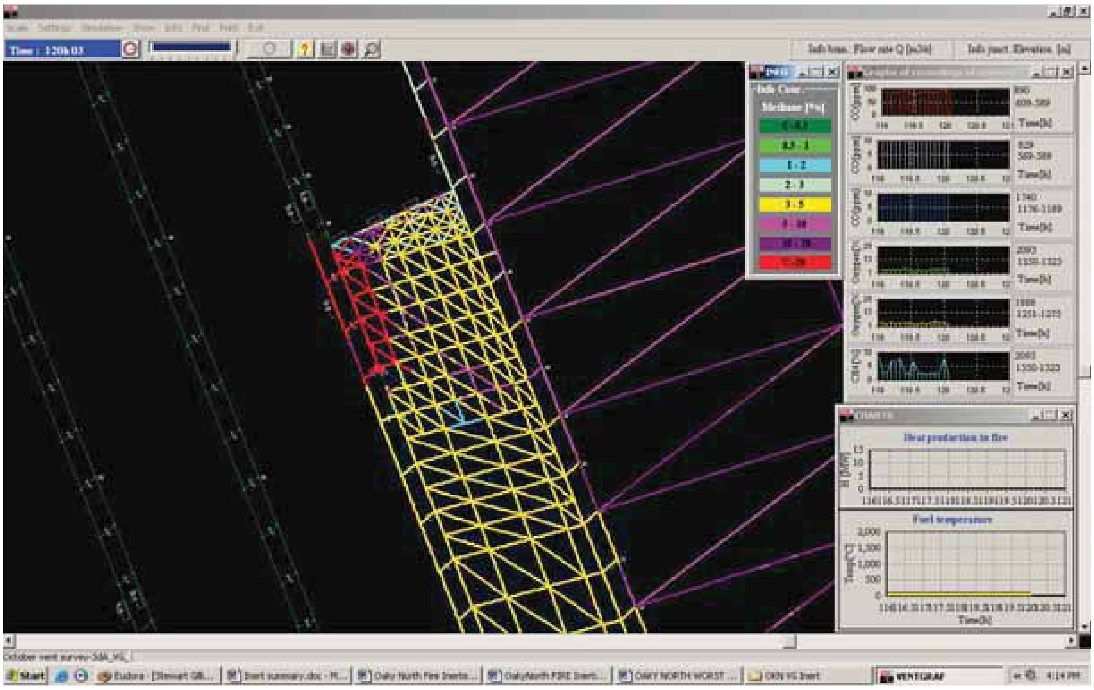


Figure 6.6 Methane distribution after 7200 minutes

Summary *With the GAG running after 5 days there is no significant fire. Outbye the fire oxygen is 0.1 percent.*

6.5. Oaky North Fire Inertisation Scenario 3

Scenario: *Belt Fire in South LW 4 MG 22CT Tripper drive.*

Sections

1. *LW 4 at 37ct 3/06*
2. *Dev 301 MG at 10ct 3/06*
3. *Dev South MG 7 at 28ct 3/06*
4. *Dev Stone Development road header (contractor) MG6 at 8ct at 3/06*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.
- CH4 source of 600 litres/s from LW goaf at TG end of face
- CO Gas sensors set at points before and after fire, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.
  CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 30 minutes, Spillage coal burning. Simulate 1 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control Fire fighting control commences with hoses; ineffective.

Step 2 Time 30 – 120 minutes, Spillage coal burning. Simulate 5 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control At 120 minutes decision made to introduce high flow inertisation – GAG

Step 3 Time 120 – 300 minutes, Spillage coal burning. Simulate 10m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 4 Time 300 – 360 minutes, Spillage coal burning. Simulate 25m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

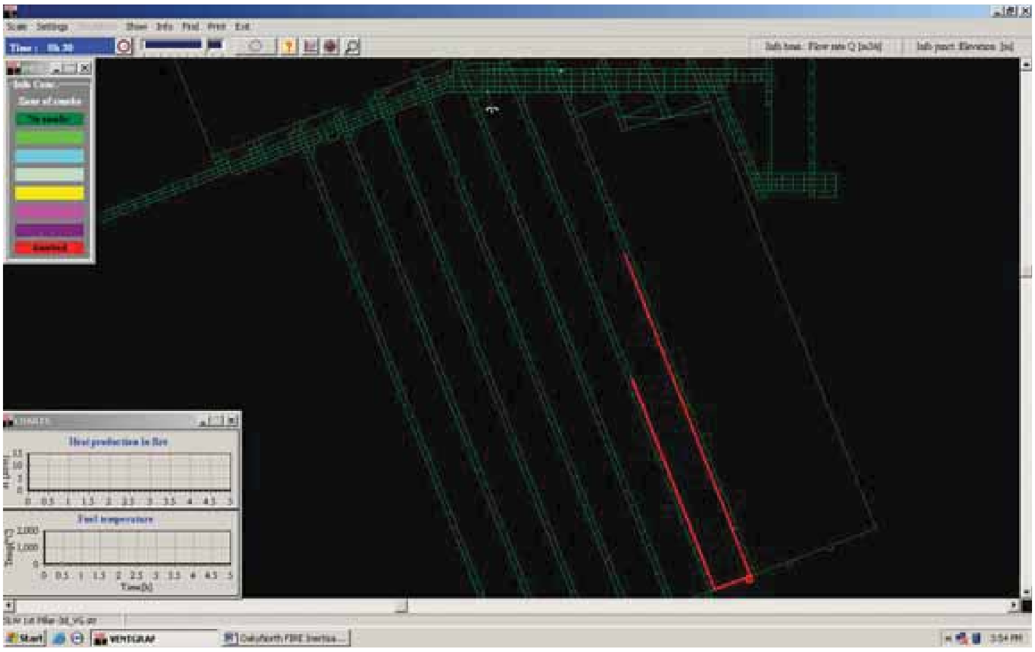


Figure 6.7 Smoke distribution after 30 minutes

Fire is constrained but still burning

At 300 minutes GAG has been set up at the Intake Drift Close emergency door R=10

Examine all three main fan curve operating points; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 5 After 360 minutes shut down No 1 fan; fan louvre doors closed R=20
Examine No 2 and No 3 fan curve operating points

Control Assess effectiveness of GAG

Step 6 After 390 minutes install brattice stoppings Belt drift C heading R = 1

Step 7 After 420 minutes Shut down No 2 fan; fan louvre doors closed R= 10

Examine No 3 fan curve operating point; Localised reversal occurs over fire

Step 8 After 435 minutes Close Main Portal D Heading Emergency Door R = 10

Control Assess effectiveness of GAG

Step 9 After 450 minutes Shut down No 3 fan; fan louvre doors left open
Close Main Portal B Heading Emergency Door R = 10

After 8 hours, reversal brings methane over the fire and causes a large explosion.

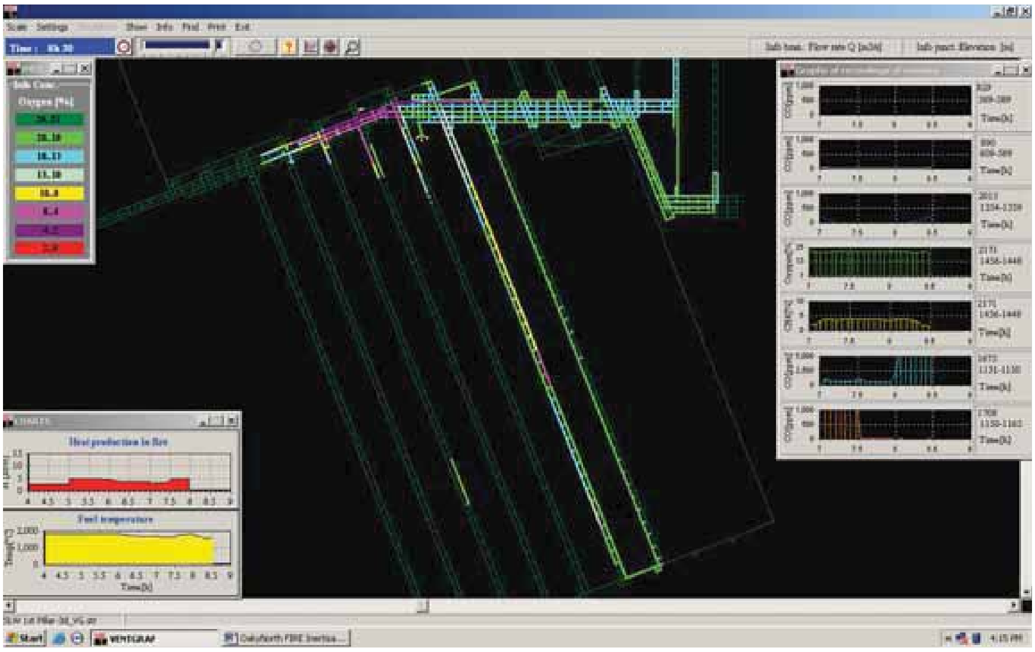


Figure 6.8 Oxygen distribution after 510 minutes

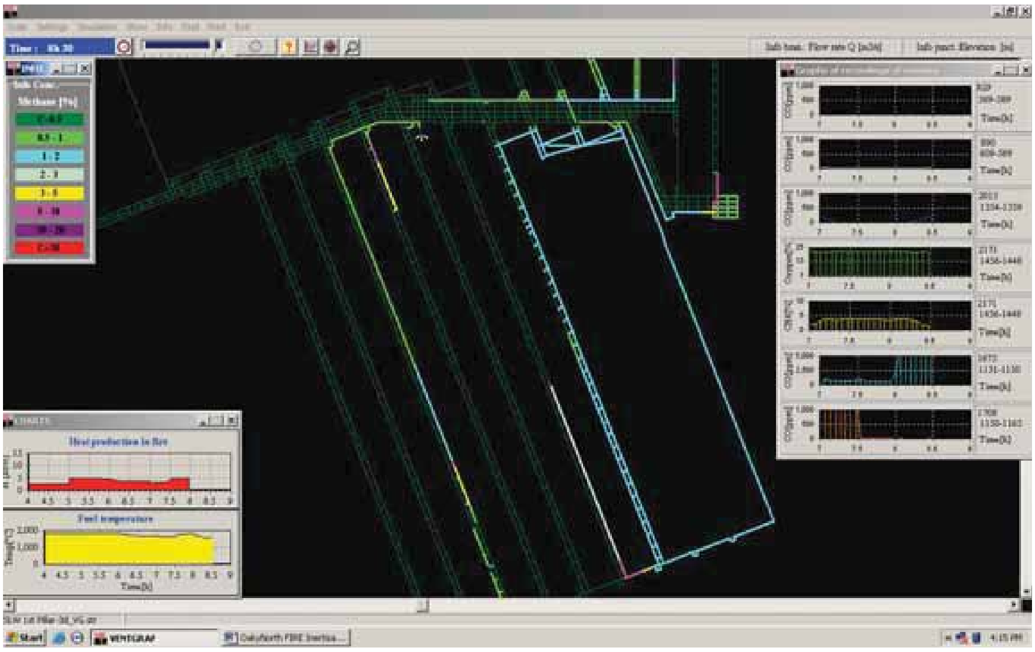


Figure 6.9 Methane distribution after 510 minutes

Control Assess effectiveness of GAG

Summary *Explosion occurred as soon all fans were turned off due to an airflow reversal across fire.*

6.6. Oaky North Fire Inertisation Scenario 4

Scenario: *Belt tripper drive Mains 11ct C Hdg*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.

CH_4 source of 600 litres/s from LW goaf at TG end of face

Prior to running fire simulation pre-enter some of the controls that may be required e.g.

- CO Gas sensors set at points before and after fire, and
- CO Gas sensors set at points either side of bottom of Ventilation Shaft.

Simulation

Step 1 Time 0 – 30 minutes, Spillage coal burning. Simulate 1 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control Fire fighting control commences with hoses; ineffective.

Step 2 Time 30 – 120 minutes, Spillage coal burning. Simulate 5 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control At 120 minutes decision made to introduce high flow inertisation – GAG

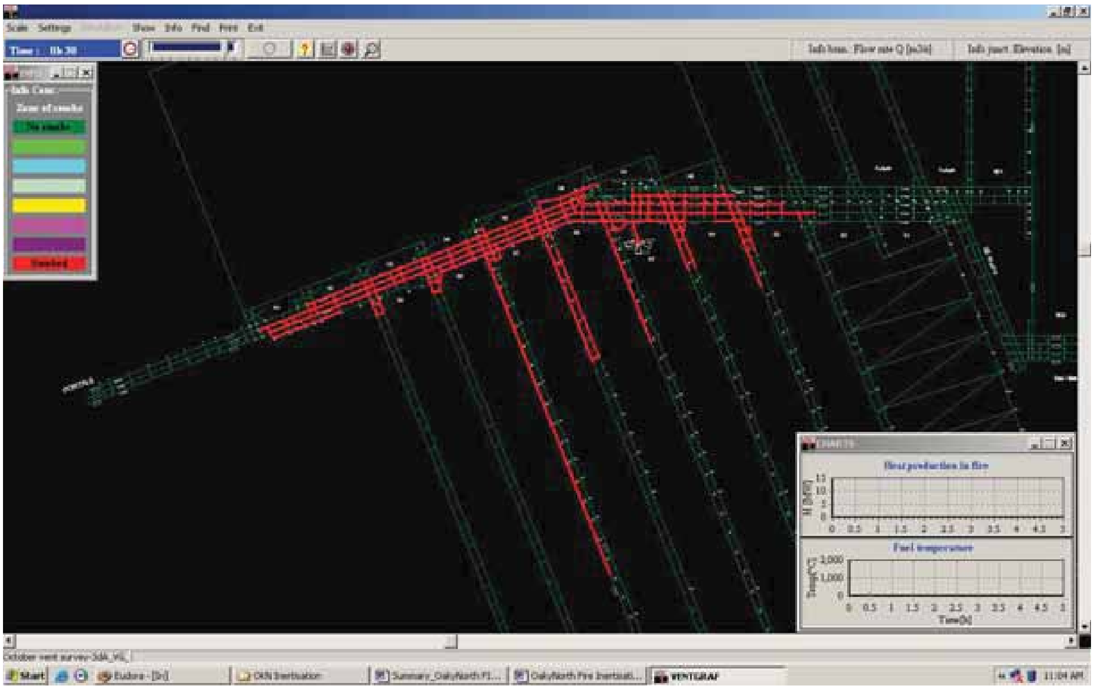


Figure 6.10 Smoke distribution after 30 minutes

Step 3 Time 120 – 300 minutes, Spillage coal burning. Simulate 10m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Step 4 Time 300 – 330 minutes, Spillage coal burning. Simulate 10m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Fire is constrained but still burning

At 300 minutes GAG has been set up at the Intake Drift Close emergency door R=10

Control Assess effectiveness of GAG

Examine all three main fan curve operating points; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 5 330 minutes, Shut down No 1, No 2 and No 3 fans; fan louvre doors closed R=20
Close Main Portal D Heading Emergency Door R = 10
Close Main Portal B Heading Emergency Door R = 10

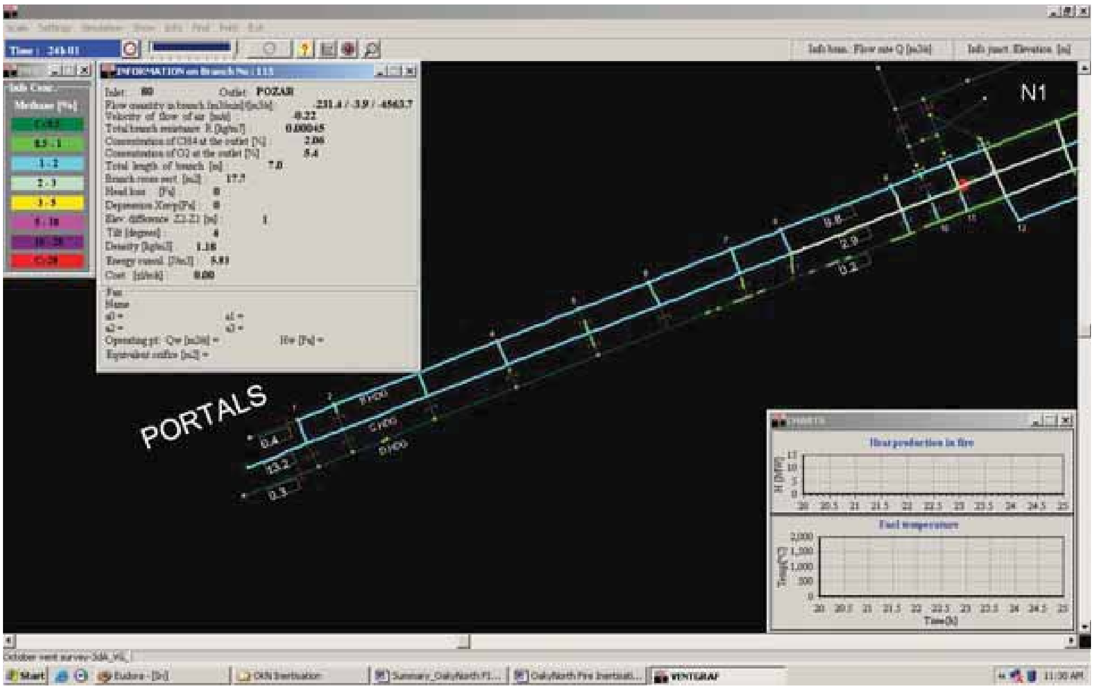


Figure 6.11 Methane distribution after 1440 minutes

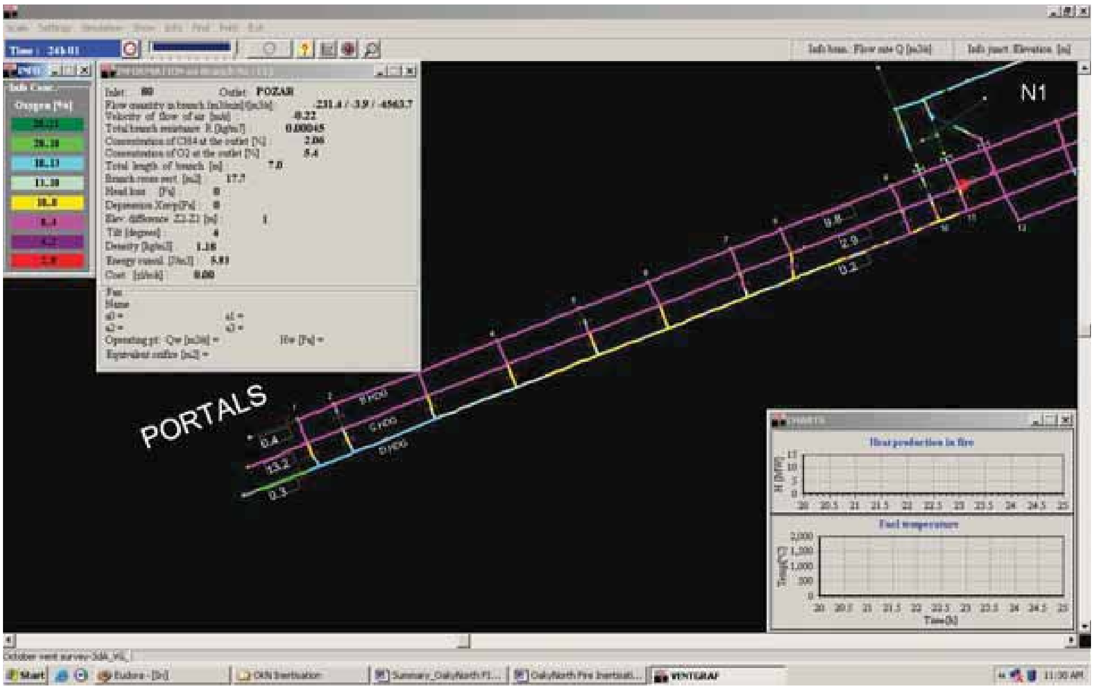


Figure 6.12 Oxygen distribution after 1440 minutes

Control Assess effectiveness of GAG

Summary *With GAG running fire intensity insignificant at 24 hours and oxygen level outbye fire at less than 5.4 percent. Face methane passing over fire potentially causing explosions.*

6.7. Oaky North Fire Inertisation Scenario 5

Scenario: *Development in 7 MG at 26 ct (100m pillar). Eimco vehicle caught fire at 500m outbye of the face. Face 2.2 m^3/tonne CH_4. October 2005*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO and CH_4 Gas sensors in Dev 7 TG 2-3 ct panel returns, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.

Simulation

Figure 6.13 Fire location at initiation

Step 1 Time 0 – 15 minutes: 200 litres diesel fuel is burning; Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 2 Time 15– 30 minutes: Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 3 Time 30 – 60 minutes: 200 litres fuel is burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

Step 4 Time 60 – 120 minutes: an additional 20m length of coal pillar equivalent of a total 47m additional burning; Simulate 47m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

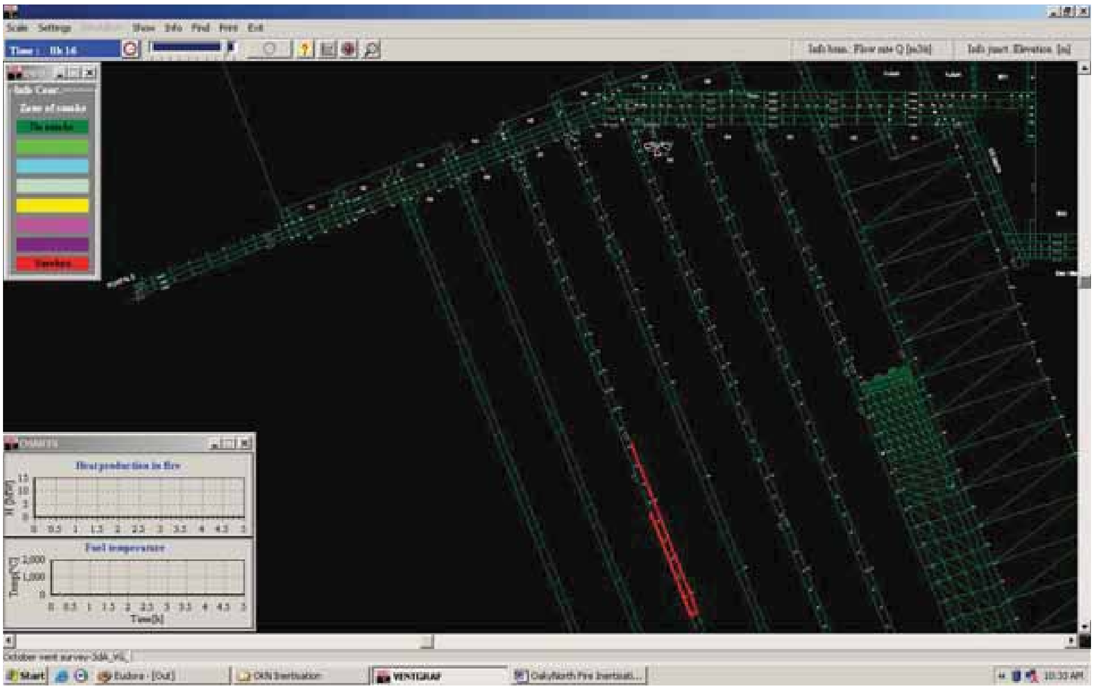


Figure 6.14 Smoke distribution after 15 minutes

Control Assume all mining crewmembers out of mine.

IMT team formed; Decision made to introduce high flow inertisation – GAG as soon as all crews evacuated out of mine.

Step 5 Time 120 – 300 minutes: Additional 20 m entry length coal caught on fire. Simulate 67m length oil fire over entry width; time constant 120s, intensity 7. $CO:CO_2 = 0.1$ (assume H_2 = CO level). Fire very unstable and not under control.

Step 6 Time 300 - ? minutes: Continue simulating 67m length fire over entry width; time constant 120s, intensity 7 $CO:CO_2 = 0.1$. (assume H_2 = CO level); Fire out of control.

Control High flow inertisation GAG unit has arrived and is set up

At 300 minutes: GAG has been set up at the Drift Transport Portal entry and emergency door closed, R=10; Initiate GAG.

At 330 mins shut down one main fan; fan louvre doors closed R=20
At 360 mins shut down second main fan; fan louvre doors closed R=20
At 360 B and C Hdg portal doors closed R = 10 and R = 1 respectively.
Due to less ventilation air methane levels have increased in the return air up to 3.5%.

At 390 mins shut down the third main fan
At 390 D Hdg portal doors closed R = 10.

Explosion occurs due to localised reversal of methane over fire in SMG7.

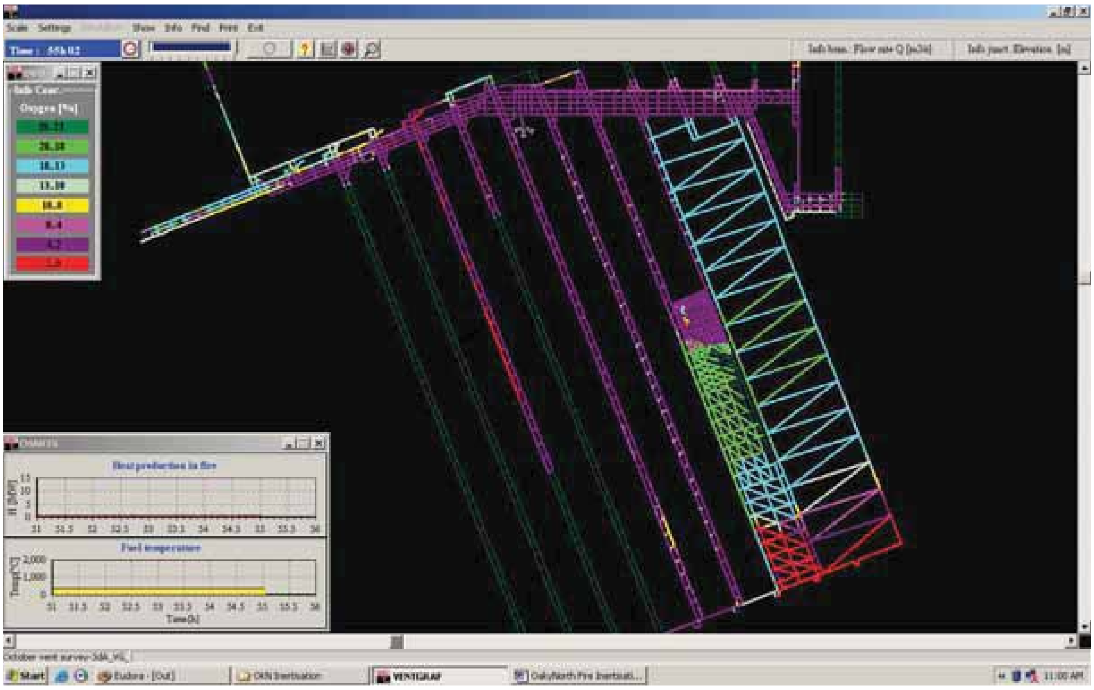


Figure 6.15 O_2 distribution at 3300 minutes

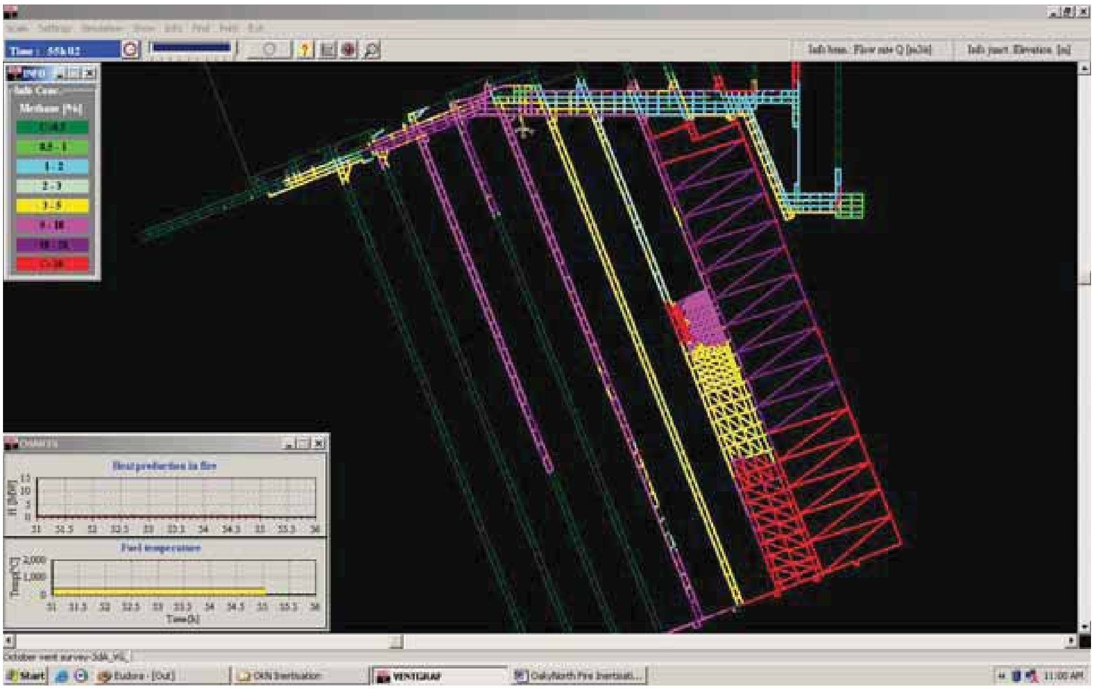


Figure 6.16 Methane (from other sources) distribution at 3300 minutes

Summary Inertisation and extinguishment of fire eventually occurs over a long period of time, but with several methane explosions.

7. REVIEW OF OPTIONS FOR IMPROVING ABILITY TO INERTISE PRIORITY FIRES AT OAKY NORTH MINE

As mentioned in Section 3.3, location position of the inertisation unit's point of coupling to the mine, or the docking point, is a major determinant of potential success for most efficient suppression of a specific underground fire. Traditionally in Queensland docking points have been placed on intake ventilation headings (either travel or conveyor belt roads). Some mines have prepared docking points on boreholes of about 1.0 to 2.0m diameter placed at the back of longwall panels.

Scenarios developed for Oaky North Colliery have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage following a fire. Table 7.1 shows results of the outcome of the five scenarios investigated in Chapter 6.

Analysis of these five scenarios leads to the following comments.

- Category A covers fire in which the inertisation product is directed fully over the fire. None of the five mine priority fires examined achieved the situation in which the simulated fire is directly stabilised to aid recovery in a timely manner.
- Category B covers situations in which the inertisation product goes straight to the fire but there is significant dilution from other ventilation air or leakage through stoppings. Because of dilution stabilisation of a fire through inertisation can only be achieved with some main surface fan changes. One scenario is in this category. Under these situations the fire should, over time, be abated or stabilised to a point where conventional recovery approaches can be initiated.
- Category C covers priority fires in which the GAG output will never reach the fire location without stopping of one or more main surface fans to rebalance ventilation within the pit. In many of these cases requiring fan changes to put GAG output across the fire location effective ventilation air velocity has been reduced to the extent that local reversal across the fire occurs and fire fumes are pulled across the fire. This is an unsatisfactory situation as fire smoke and fumes can carry combustible products. This situation broadly prevails for 40 percent (two scenarios) of the cases examined.
- Category D covers priority fires in which the GAG output will never reach the fire location even if surface main fans are altered. These are fire locations within panel sections in which either the fire behaviour stops normal intake ventilation flow into the section headings or the GAG docking point is in an airway that is isolated from the section. There is no such case in the five scenarios examined.
- Category E covers priority fires in gassy mines in which section production gas make has been included in the simulation modelling. GAG exhaust will never reach the fire location without stopping of one or more main surface fans to rebalance ventilation within the pit.

However this change in ventilation causes working section methane and ventilation air (incl. fire fumes) to reverse across the fire. This is clearly a potentially dangerous situation. This situation was found in 40 percent (two scenarios) of the cases examined.

It was determined that Oaky North is not in the best prepared position to undertake efficient GAG inertisation in the event of a major fire

Recommended general actions that can be undertaken to improve the effectiveness of inertisation in an underground ventilation network can be drawn from the following.

1. Maintain use of existing docking station but with additional underground segregation to control the delivery of inert gas.
2. Try alternative Portal docking station locations.
3. Try alternative Portal docking station locations with additional underground segregation.
4. Drill new borehole to deliver inert gas more directly to the fire site.

The five scenarios have been re-simulated using appropriate actions from the above approaches. These new scenarios are described in the following sections.

Table 7.1 Summary of Scenario Outcomes on the Effects of Inertisation

Scenario No	Fire Location	Fire Type	GAG Location	Segregation actions	Fan Actions	Outcomes
1	Mains C Hdg conveyor at 39ct	Belt (oil)	Transport Drift Portal	None; Only external sealing required for GAG operation.	All shut down	With fans still running, GAG exhaust was diluted significantly. After all fans off, fire insignificant at 10 hours and oxygen level outbye fire less than 2.9 percent. ***(Category B)***
2	Behind South Longwall 3	Goaf spon combustion	Transport Drift Portal	None; Only external sealing required for GAG operation.	All shut down	At 34 hours local reversal occurred to produce a minor methane burn off. With the GAG running after 5 days there is no significant fire. Outbye the fire oxygen is 0.1 percent. ***(Category C)***
3	South LW 4 MG 22CT Tripper drive	Belt Fire	Transport Drift Portal	None; Only external sealing required for GAG operation.	All shut down	Explosion occurred as soon as all fans were turned off due to a reversal. ***(Category E)***
4	Belt tripper drive Mains 11ct C Hdg	Spillage Coal	Transport Drift Portal	None; Only external sealing required for GAG operation.	All shut down	With GAG running Fire intensity insignificant at 24 hours and oxygen level outbye fire at less 5.4 percent. Face methane passing over fire potentially causing explosions. ***(Category C)***
5	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle fire (diesel)	Transport Drift Portal	Close Main Portal B Heading Emergency Door Close Main Portal C Heading Emergency Door Close Main Portal D Heading Emergency Door	All shut down	Explosion occurs due to localised reversal of methane over fire in SMG7. Inertisation eventually occurs over a long period of time, but with several methane explosions. ***(Category E)***

7.1. Oaky North Fire Inertisation Scenario 1A

Scenario *Belt Fire in Mains C Hdg conveyor at 39ct*

Inertisation Strategy *The pit bottom area is very open. Various forms of segregation are introduced to reduce GAG exhaust dilution with intake air in the pit bottom area. GAG remains docked at the Decline Portal.*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.

CH_4 source of 600 litres/s from LW goaf at TG end of face

Prior to running fire simulation pre-enter some of the controls that may be required e.g.

- CO Gas sensors set at points before and after fire, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.
  CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 15 minutes: 30 litres oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 2 Time 15 - 30 minutes: 230 litres oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Step 3 Time 30 – 120 minutes: Oil consumed. 50 m entry length coal fire develops. time constant 14,400s, intensity 5.

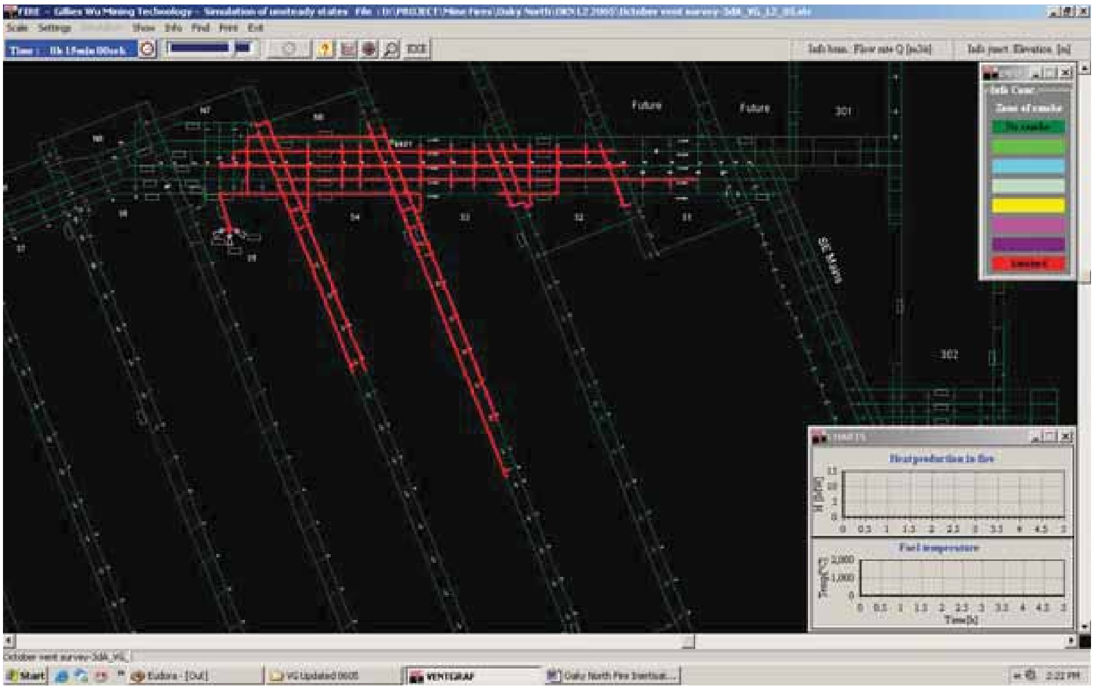


Figure 7.1 Smoke distribution after 15 minutes

Control At 120 minutes decision made to introduce high flow inertisation – GAG

Step 4 Time 120 – 300 minutes: 100 m entry length coal fire develops. time constant 14,400s, intensity 5.

Step 5 300 minutes: segregation devices (e.g. prep seals, brattice, remote controlled doors or manually controlled doors) installed to assist segregation of inert gases. Segregation for pit bottom required at the following points. For delivering of GAG exhaust into

B Hdg

- Close machine door at 37ct B – C
- Prep seal B Hdg 35 – 36

C Hdg

- Prep seal at B Hdg 35-36 R=2
- Open machine door at 37ct B-C
- Brattice around belt structure C Hdg 35 – 36ct R=1
- Brattice around belt structure 37ct C – D R=1
- Prep seal at 36ct C - D R=2
- Prep seal at B Hdg 37-37A ct R=2
- Segregation stopping 38ct, 39ct and 40ct C-D

D Hdg Must turn at least one main fan off

- Close B Hdg, C Hdg & D Hdg, 35 – 36ct
- Close B Hdg & C Hdg 37 - 37A ct
- Open machine door at 37ct B-C

Step 6 Time 300 minutes: GAG has been set up at the Intake Drift Close emergency door R=10; Start GAG. Undertake pit bottom segregation strategy for inertisation of fire in C Heading.

Control Assess effectiveness of GAG

Examine all three main fan curve operating points; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 330 minutes Shut down No 1 fan; fan louvre doors closed R=20

Fire unstable and erratic local air reversals over fire.

Step 8 345 minutes: Shut down No 2 fan; fan louvre doors closed R=20
Examine No 3 fan curve operating point

Step 9 390 minutes: Shut down No 3 fan

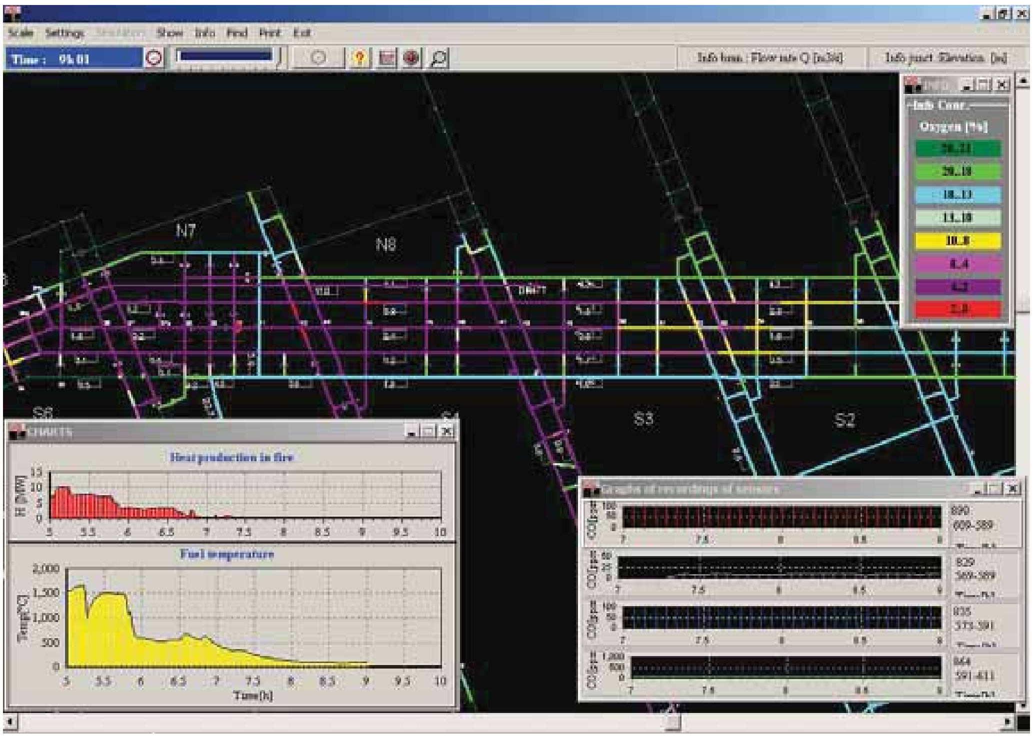


Figure 7.2 Oxygen distribution after 540 minutes

Summary *With GAG running fire intensity insignificant at 9 hours and oxygen level outbye fire at less than 2.5 percent. There is only a slight reduction in the time needed to achieve satisfactory inertisation of the mine with extra segregation.*

7.2. Oaky North Fire Inertisation Scenario 2A

Scenario *LW goaf fire - spider web arrangement. Spontaneous Combustion in goaf behind South Longwall 3 face currently at 15ct. Spontaneous Combustion potentially spread over 600m (Model: October vent survey-3dA_VG_Goaf_BH).*

Inertisation Strategy: *Close Mains C Hdg 35 – 36, Mains B Hdg 35 – 36 and Mains D Hdg 35 – 36; Shut down all fans.*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
- Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO Gas sensors set at points either side of bottom of Ventilation Shaft.
- CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- O_2 sensor in the LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 360 minutes: 1 m entry length coal fuel in 18 c/t MG edge of goaf burning; time constant 14400s, intensity 1 CO:CO_2 = 0.1. (assume H_2 = CO level); fire very unstable and not under control.

Step 2 Time 360– 720 minutes: 5 m entry length coal burning with gas continuing to burn; time constant 14400s, intensity 2.

Step 3 Time 720 – 1080 minutes: Continue coal fire 25 m entry length coal burning; time constant 14400s, intensity 4.

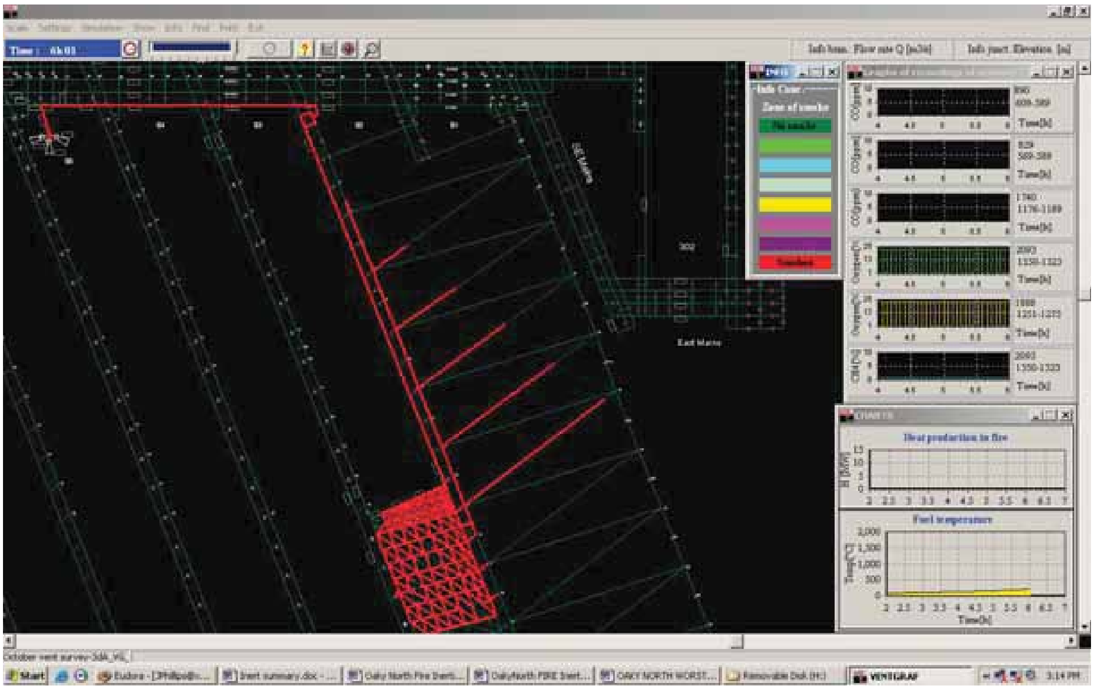


Figure 7.3 Smoke distribution after 360 minutes

Step 4 Time 1080 - 1440 minutes: Continue coal fire 100 m entry length coal burning; time constant 14400s, intensity 8. Fire very unstable and not under control.

CO concentration at 19 hours sets off alarm at bottom of vent shaft.

Step 5 Time 1440 - 1800 minutes: Continue coal fire 200 m entry length coal burning; time constant 14400s, intensity 10.

Step 6 Time 1440 minutes GAG has been set up at the Intake Drift Close emergency door R=10

Close C Hdg 35 – 36 Brattice seal R=1
Close B Hdg 35 – 36 Prep seal R=2
Close D Hdg 35 – 36 Prep seal R=2
Shut down No 1 fan; fan louvre doors closed R=20
Shut down No 2 fan; fan louvre doors closed R=20
Examine No 3 fan curve operating point

Step 7 1680 minutes: Shut down No 3 fan. Close main portal B Emergency Door R=20

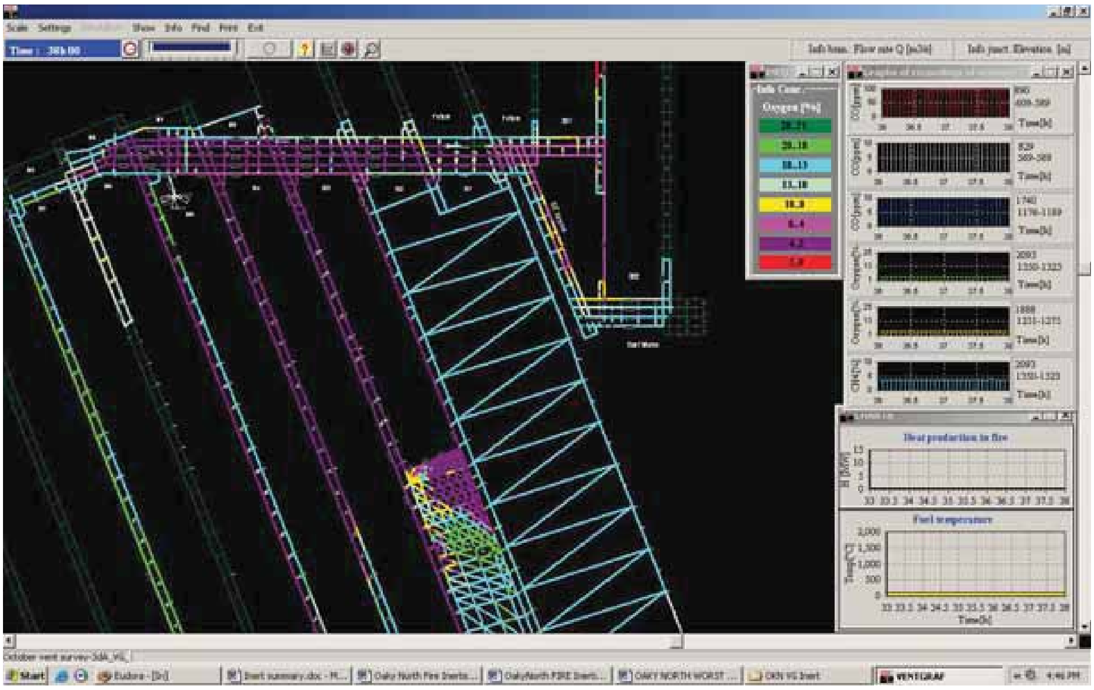


Figure 7.4 Oxygen distribution after 2280 minutes

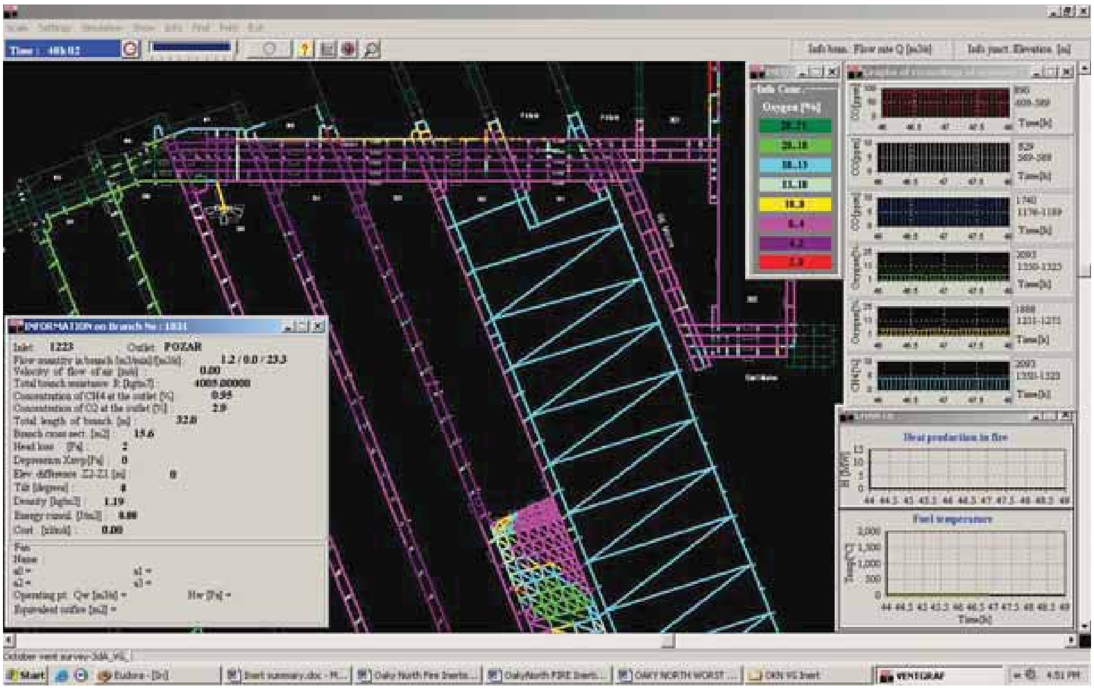


Figure 7.5 Oxygen distribution after 2880 minutes

Summary *After 2 days with the GAG running, there is no significant fire. Outbye the fire the oxygen is 2.9 percent.*

7.3. Oaky North Fire Inertisation Scenario 2B

Scenario *LW goaf fire - spider web arrangement. Spontaneous Combustion in goaf behind South Longwall 3 face currently at 15ct. Spontaneous Combustion potentially spread over 600m (Model: October vent survey-3dA_VG_Goaf_BH).*

Inertisation Strategy: *GAG on Panel Borehole***;** *Close Mains C Hdg 35 – 36, Mains B Hdg 35 – 36 and Mains D Hdg 35 – 36; Shut down all fans*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Borehole at MG 2 ct LW.
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
- Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO Gas sensors set at points either side of bottom of Ventilation Shaft.
- CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- O_2 sensor in the LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 360 minutes: 1 m entry length coal fuel in 18 c/t MG edge of goaf burning; time constant 14400s, intensity 1 $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control.

Step 2 Time 360– 720 minutes: 5 m entry length coal burning with gas continuing to burn; time constant 14400s, intensity 2.

Step 3 Time 720 – 1080 minutes: Continue coal fire 25 m entry length coal burning; time constant 14400s, intensity 4.

Step 4 Time 1080 - 1440 minutes: Continue coal fire 100 m entry length coal burning; time constant 14400s, intensity 8. Fire very unstable and not under control

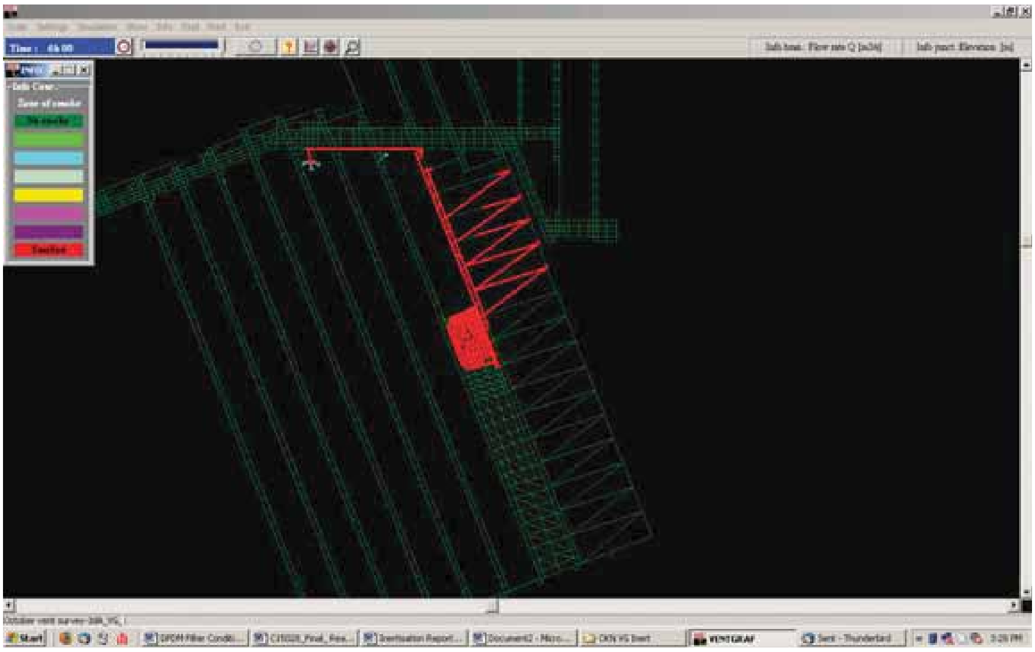


Figure 7.6 Smoke distribution after 360 minutes

CO concentration at 19 hours sets off alarm at bottom of vent shaft.

Step 5 Time 1440 - 1800 minutes: Continue coal fire 200 m entry length coal burning; time constant 14400s, intensity 10.

Step 6 Time 1440 minutes GAG has been set up at the Borehole at LW3 MG 2 ct.

Seal LW3 MG A (R = 1) and B Hdg (R = 5) 1-2ct
Borehole R = 2.3 to represent Borehole open resistance

Close Mains C Hdg 35 – 36 Brattice seal R=1
Close Mains B Hdg 35 – 36 Prep seal R=2
Close Mains D Hdg 35 – 36 Prep seal R=2

Shut down No 1 fan; fan louvre doors closed R=20
Shut down No 2 fan; fan louvre doors closed R=20
Shut down No 3 fan

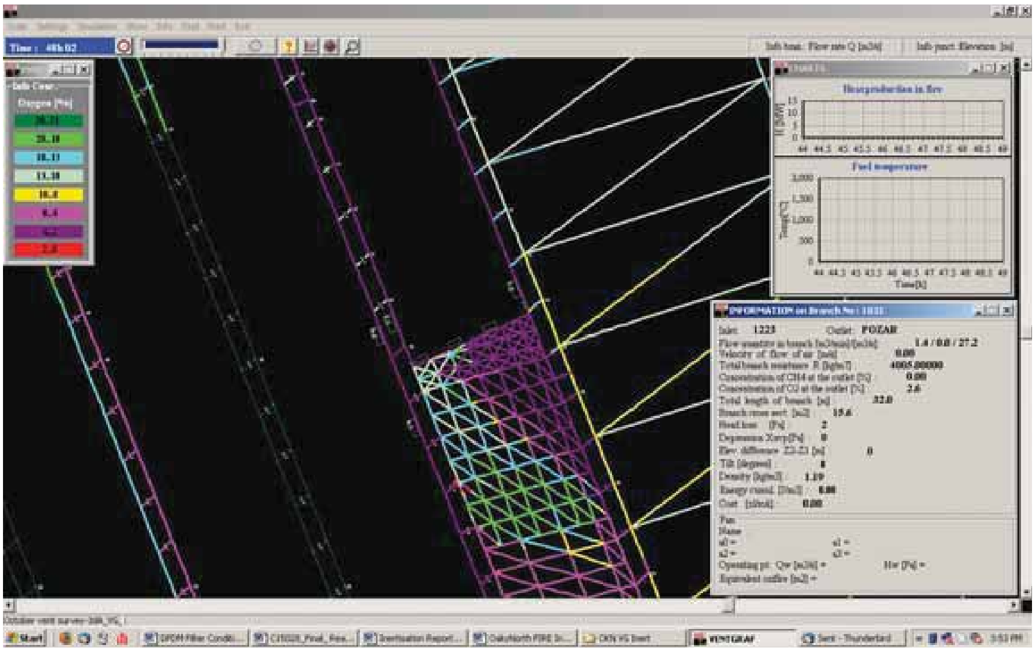


Figure 7.7 Oxygen distribution after 2880 minutes

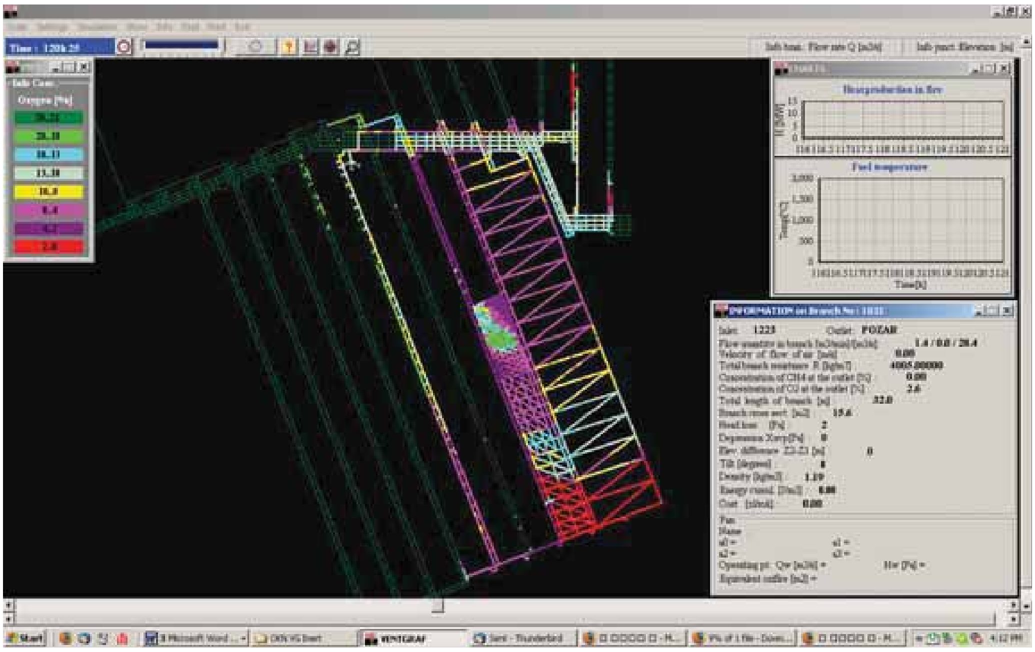


Figure 7.8 Oxygen distribution after 2880 minutes

Summary *After 2 days with the GAG running, there is no significant fire. Outbye the fire the oxygen is 2.6 percent. After 5 days with the GAG running, there is no significant fire. Outbye the fire the oxygen is 2.6 percent.*

7.4. Oaky North Fire Inertisation Scenario 2C

Scenario *LW goaf fire - spider web arrangement. Spontaneous Combustion in goaf behind South Longwall 3 face currently at 15ct. Spontaneous Combustion potentially spread over 600m (Model: October vent survey-3dA_VG_Goaf_BH).*

Inertisation Strategy: *GAG on Panel Borehole; Close Mains C Hdg 35 – 36, Mains B Hdg 35 – 36 and Mains D Hdg 35 – 36; Shut down fans 1 and 2. Close Transport Drift.*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Borehole at MG 2 ct LW.
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
- Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO Gas sensors set at points either side of bottom of Ventilation Shaft.
- CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- O_2 sensor in the LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 360 minutes: 1 m entry length coal fuel in 18 c/t MG edge of goaf burning; time constant 14400s, intensity 1 $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control.

Step 2 Time 360– 720 minutes: 5 m entry length coal burning with gas continuing to burn; time constant 14400s, intensity 2.

Step 3 Time 720 – 1080 minutes: Continue coal fire 25 m entry length coal burning; time constant 14400s, intensity 4.

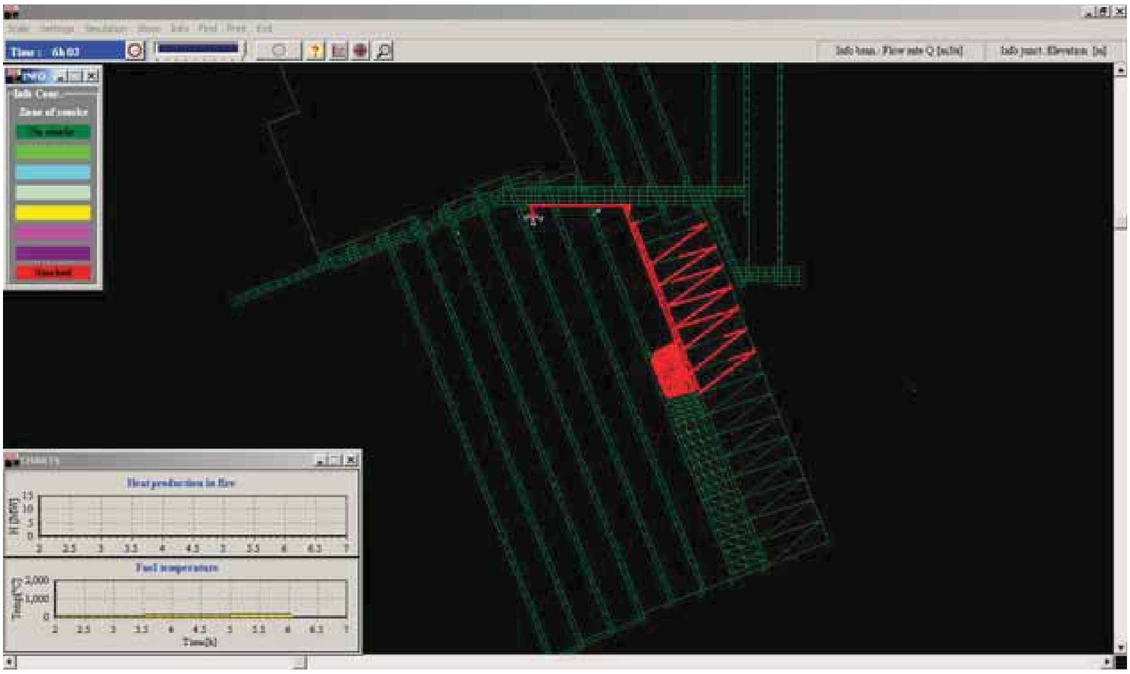


Figure 7.9 Smoke distribution after 360 minutes

Step 4 Time 1080 - 1440 minutes: Continue coal fire 100 m entry length coal burning; time constant 14400s, intensity 8. Fire very unstable and not under control

Step 5 Time 1440 - 1800 minutes: Continue coal fire 200 m entry length coal burning; time constant 14400s, intensity 10.

Step 6 Time 1440 minutes GAG has been set up at the Borehole at LW MG 2 ct
Seal LW3 MG A (R = 1) and B Hdg (R = 5) 1-2ct
Borehole R = 2.3 to represent Borehole open resistance
Close Mains C Hdg 35 – 36 Brattice seal R=1
Close Mains B Hdg 35 – 36 Prep seal R=2
Close Mains D Hdg 35 – 36 Prep seal R=2
Shut down No 1 fan; fan louvre doors closed R=20
Shut down No 2 fan; fan louvre doors closed R=20
Examine No 3 fan curve operating point (Q = 138m^3/s, P = 267 Pa)

Step 7 2880 minutes Close Drift Transport Portal emergency door R=10.

Examine No 3 fan curve operating point (Q = 132m^3/s, P = 499 Pa)

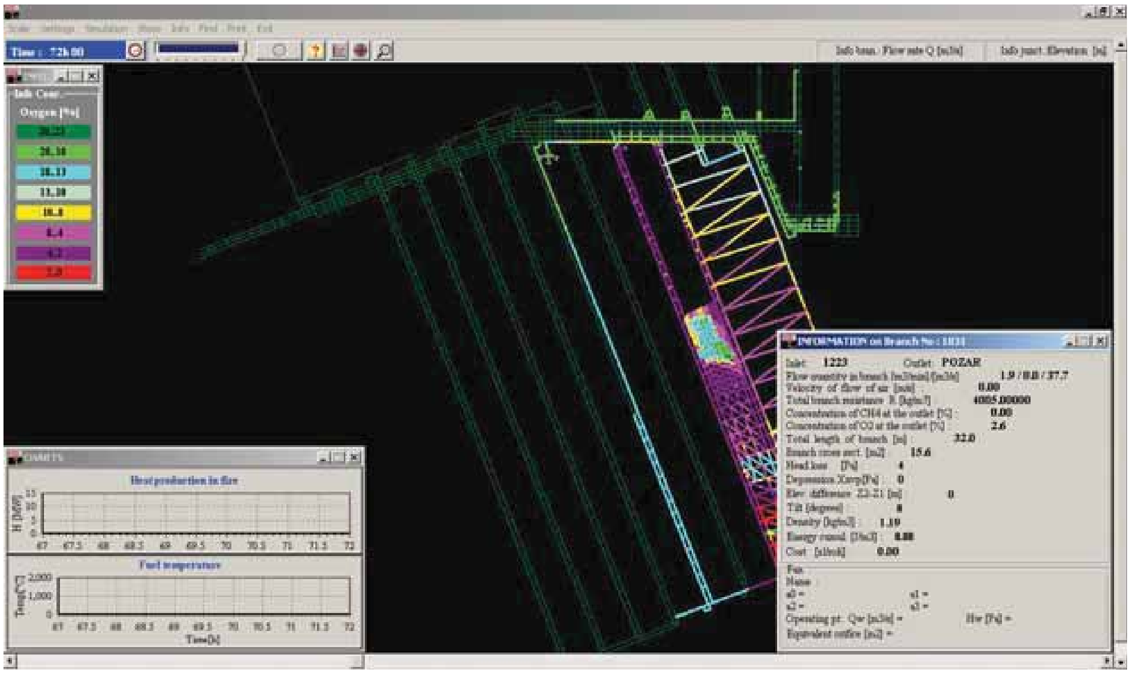


Figure 7.10 Oxygen distribution after 2880 minutes

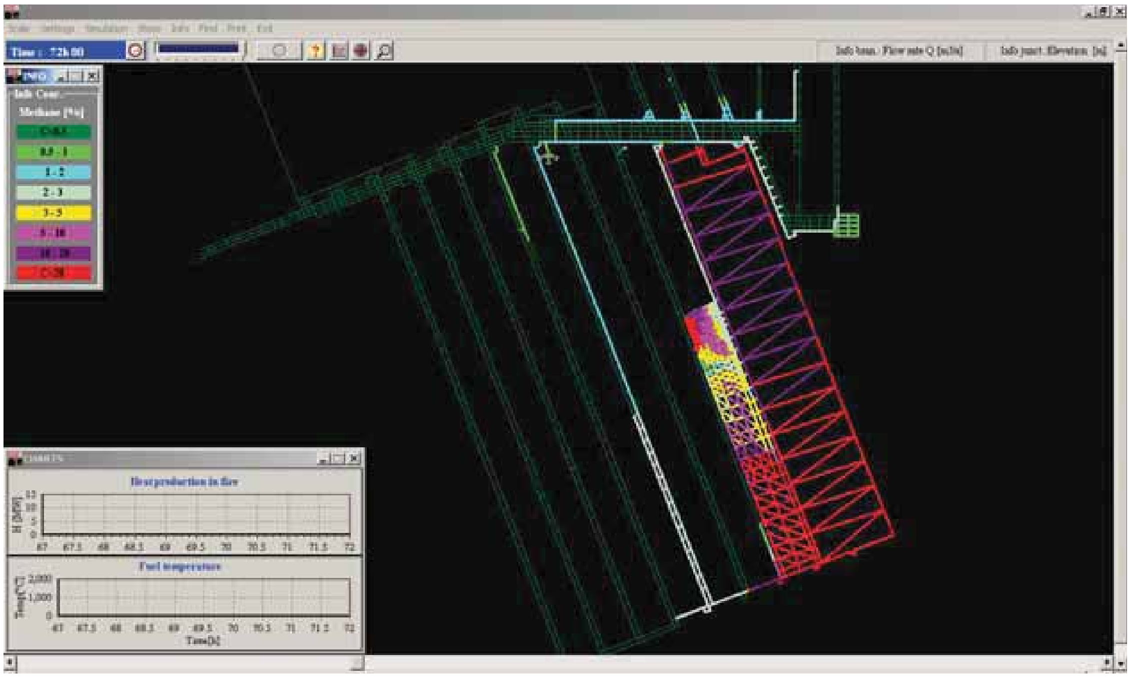


Figure 7.11 Methane distribution after 2880 minutes

Summary *After 2 days with the GAG running, there has been no significant fire. Outbye the fire the oxygen is 4.2 percent. However, by closing the Drift Transport Portal emergency door with the third fan still running, the oxygen level outbye the fire is reduced to 2.6 percent in less than 12 hours. Majority of the mine has methane level of less than 3 percent except the sealed longwall panels and goaf.*

7.5. Oaky North Fire Inertisation Scenario 3A

Scenario *Belt Fire in South LW 4 MG 22CT Tripper drive (Model: March 2006 SLW4).*

Inertisation Strategy: Segregation of Mains B, C and D Headings to assist delivery of GAG inert gases to fire site.

Sections

1. *LW 4 at 37ct 3/06*
2. *Dev 301 MG at 10ct 3/06*
3. *Dev South MG 7 at 28ct 3/06*
4. *Dev Stone Development road header (contractor) MG6 at 8ct at 3/06*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.
- CH4 source of 600 litres/s from LW goaf at TG end of face
- CO Gas sensors set at points before and after fire, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.
  CO sensor on MG leading onto LW face
- O_2 sensor at TG end of LW face
- CH_4 sensor at TG end of LW face

Simulation

Step 1 Time 0 – 30 minutes, Spillage coal burning. Simulate 1 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control Fire fighting control commences with hoses; ineffective.

Step 2 Time 30 – 120 minutes, Spillage coal burning. Simulate 5 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control At 120 minutes decision made to introduce high flow inertisation – GAG

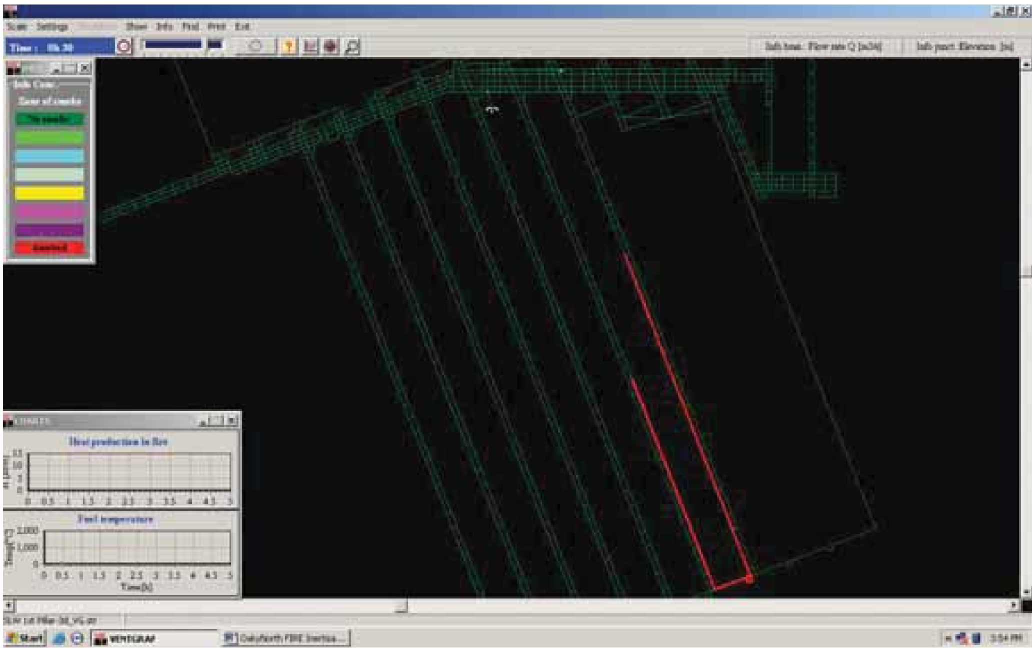


Figure 7.12 Smoke distribution after 30 minutes

Step 3 Time 120 – 300 minutes, Spillage coal burning. Simulate 10m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 4 Time 300 – 360 minutes, Spillage coal burning. Simulate 25m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Fire is constrained but still burning

At 300 minutes GAG has been set up at the Intake Drift Close emergency door R=10

Close C Hdg 35 – 36 Brattice seal R=1
Close B Hdg 35 – 36 Prep seal R=2
Close D Hdg 35 – 36 Prep seal R=2

Shut down No 1 fan; fan louvre doors closed R=20
Shut down No 2 fan; fan louvre doors closed R=20
Examine No 3 fan curve operating point

Reversal occurs, but no methane passes over the fire

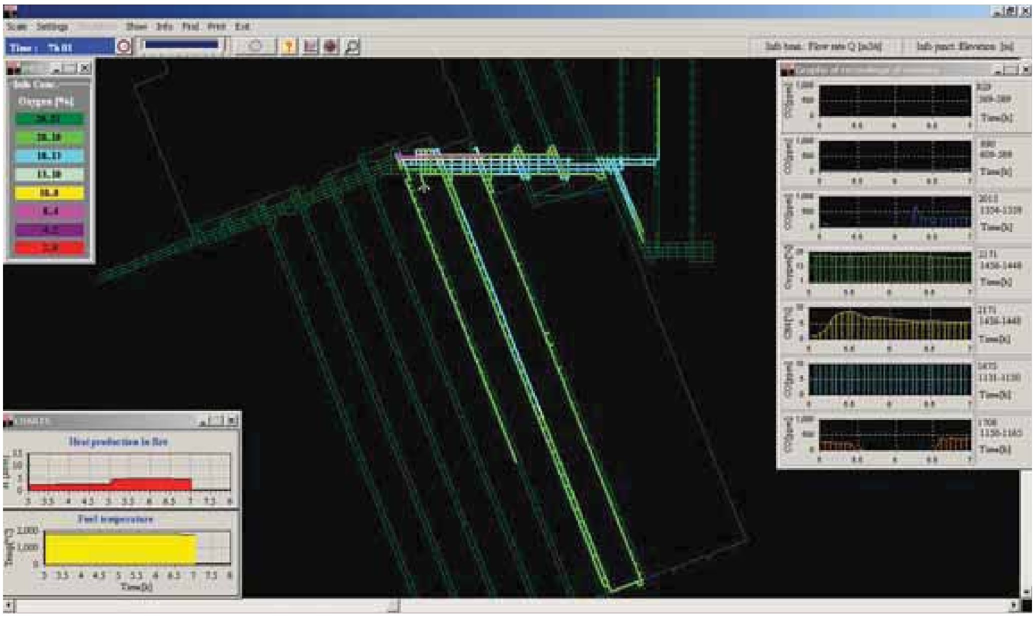


Figure 7.13 Oxygen distribution after 420 minutes

Step 5 Tighten Mains seals. After 420 minutes:
Close C Hdg 35 – 36 Brattice seal R=5
Close B Hdg 35 – 36 Prep seal R=10
Close D Hdg 35 – 36 Prep seal R=10

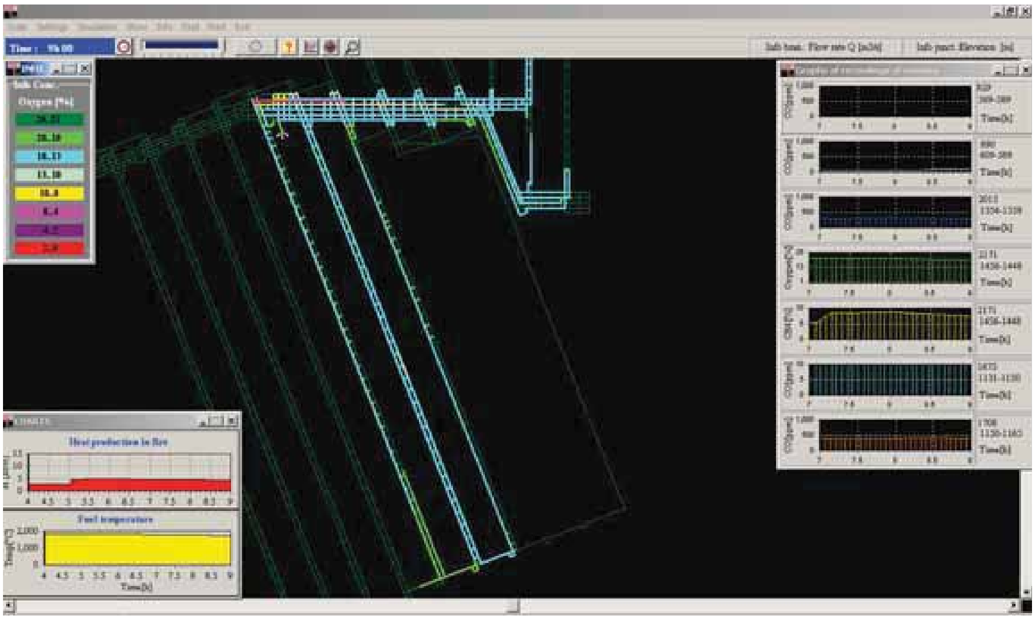


Figure 7.14 Oxygen distribution after 540 minutes

Summary *New segregation strategy does not improve inertisation strategy.*

7.6. Oaky North Fire Inertisation Scenario 4A

Scenario *Belt tripper drive Mains 11ct C Hdg*

Inertisation Strategy Set up GAG at highwall B Heading Portal. Close B, C and D Headings to assist delivery of GAG inert gases to fire site. Seal transport Drift.

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Drift Transport Portal entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.

CH_4 source of 600 litres/s from LW goaf at TG end of face

Prior to running fire simulation pre-enter some of the controls that may be required e.g.

- CO Gas sensors set at points before and after fire, and
- CO Gas sensors set at points either side of bottom of Ventilation Shaft.

Simulation

Step 1 Time 0 – 30 minutes, Spillage coal burning. Simulate 1 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control Fire fighting control commences with hoses; ineffective.

Step 2 Time 30 – 120 minutes, Spillage coal burning. Simulate 5 m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Control At 120 minutes decision made to introduce high flow inertisation – GAG

Step 3 Time 120 – 300 minutes, Spillage coal burning. Simulate 10m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

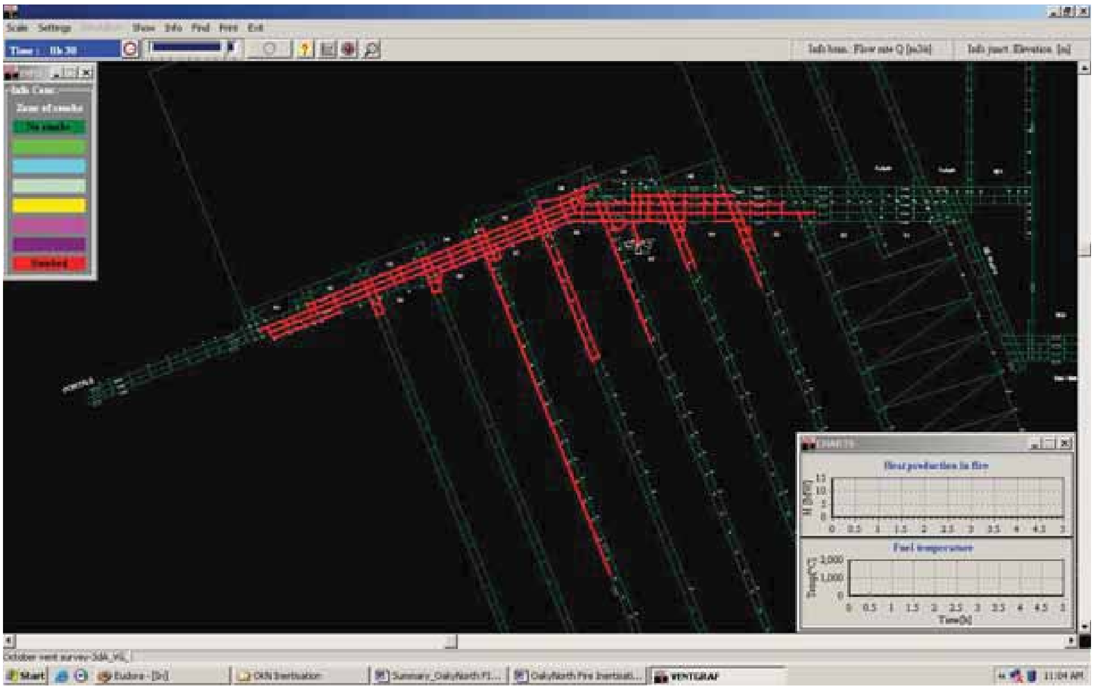


Figure 7.15 Smoke distribution after 30 minutes

Step 4 Time 300 – 330 minutes, Spillage coal burning. Simulate 10m length fire over entry width; time constant 7200s, intensity 7 and $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Fire is constrained but still burning

At 300 minutes GAG has been set up at the Portal B Emergency door R=10
Close Main Portal B Heading Emergency Door R = 10

Control Assess effectiveness of GAG

Examine all three main fan curve operating points
NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 5 330 minutes, Shut down No 1 fan; fan louvre doors closed R=10
Close Main Portal C Heading Emergency Door R = 1

Step 6 390 minutes, Shut down No 2 fan; fan louvre doors closed R=10
Close Main Portal D Heading Emergency Door R = 10

Step 7 390 minutes, Shut down No 3 fan
Close Intake Drift Heading Emergency Door R = 10

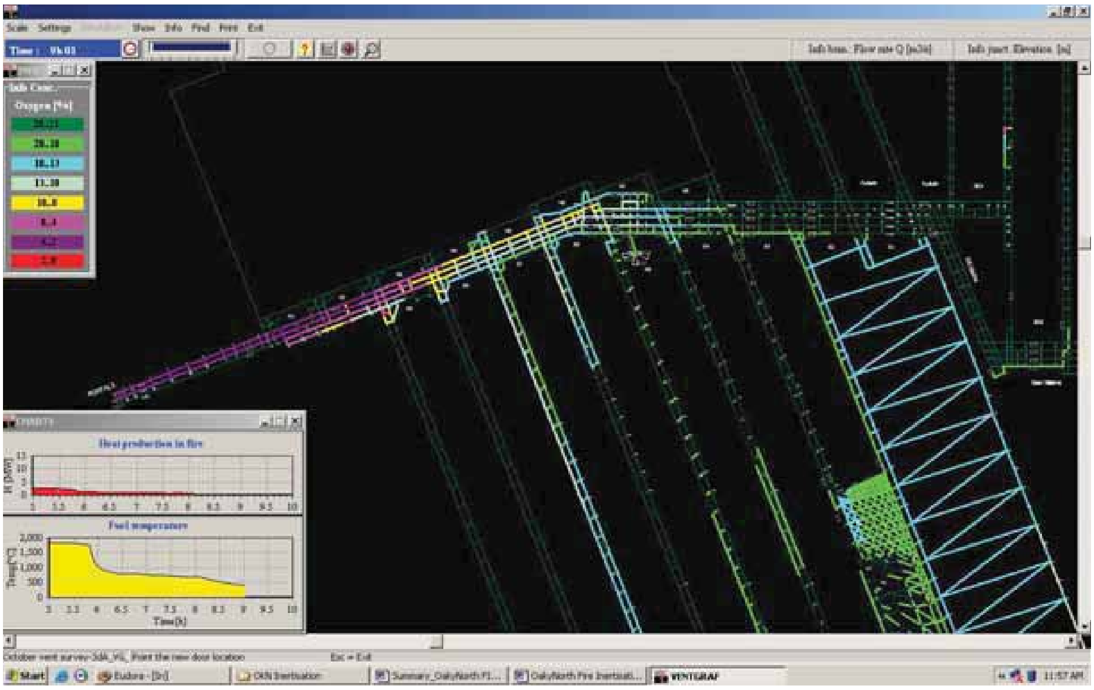


Figure 7.16 Oxygen distribution after 540 minutes

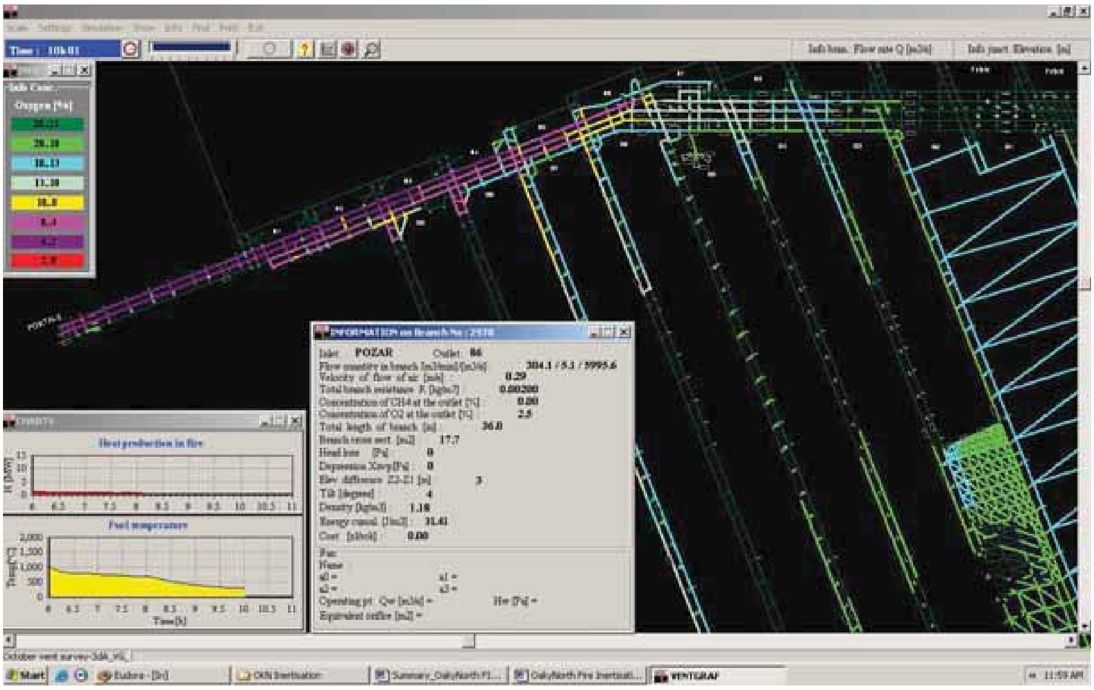


Figure 7.17 Oxygen distribution after 600 minutes

Summary *With GAG running Fire intensity insignificant at 10 hours and oxygen level outbye fire at less 2.5 percent.*

7.7. Oaky North Fire Inertisation Scenario 5A

Scenario *Dev in 7 MG at 26 ct (100m pillar). Eimco vehicle fire at 500m outbye of the face. Face 2.2 m^3/tonne CH_4. (Model: October 2005).*

Inertisation Strategy GAG placed on 1 m dia. Borehole at MG7 2 ct. Seal off SMG7 A and B Hdg 1-2ct. Close Highwall B and C Hdg portal doors. Close C Hdg 35 – 36 Brattice seal, B Hdg 35 – 36 Prep seal and D Hdg 35 – 36 Prep seal.

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ct at 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Borehole at MG7 2 ct
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO and CH_4 Gas sensors in Dev 7 TG 2-3 ct panel returns, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.

Simulation

Step 1 Time 0 – 15 minutes: 200 litres diesel fuel is burning; Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$ (assume H_2 = CO level).

Step 2 Time 15– 30 minutes: Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 3 Time 30 – 60 minutes: 200 litres fuel is burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

Step 4 Time 60 – 120 minutes: an additional 20m length of coal pillar equivalent of a total 47m additional burning; Simulate 47m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

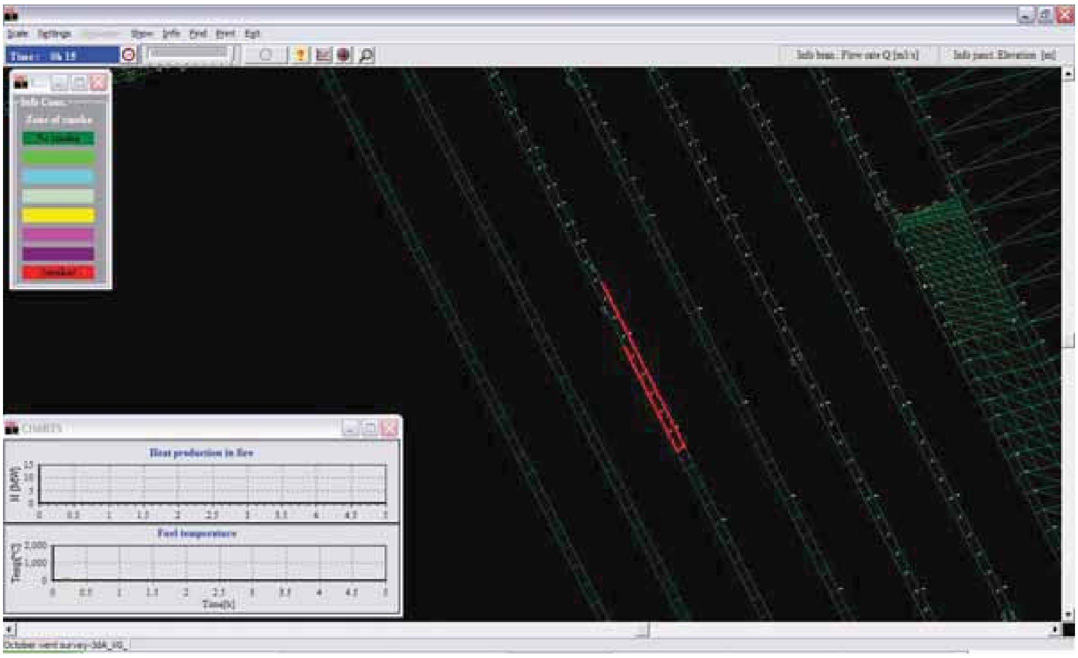


Figure 7.18 Smoke distribution after 15 minutes

Control Assume all mining crewmembers out of mine.

IMT team formed; Decision made to introduce high flow inertisation – GAG as soon as all crews evacuated out of mine.

Step 5 Time 120 – 300 minutes: Additional 20 m entry length coal caught on fire. Simulate 67m length oil fire over entry width; time constant 120s, intensity 7. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control.

Step 6 Time 300 - ? minutes: Continue simulating 67m length fire over entry width; time constant 120s, intensity 7 $CO:CO_2$ = 0.1. (assume H_2 = CO level);

Fire out of control

Control High flow inertisation GAG unit has arrived and is set up

At 300 minutes: GAG has been set up at the Borehole at MG7 2 ct and seal off MG7 A (R = 1) and B Hdg (R = 5) 1-2ct - Change BH R = 2.3.
Highwall B and C Hdg portal doors closed R = 10 and R = 1 respectively.
Close C Hdg 35 – 36 Brattice seal R=1
Close B Hdg 35 – 36 Prep seal R=2
Close D Hdg 35 – 36 Prep seal R=2
Shut down No 1 fan; fan louvre doors closed R=20
Shut down No 2 fan; fan louvre doors closed R=20
Examine No 3 fan curve operating point; Initiate GAG.

After 570 minutes improve C Hdg portal seal to R=10

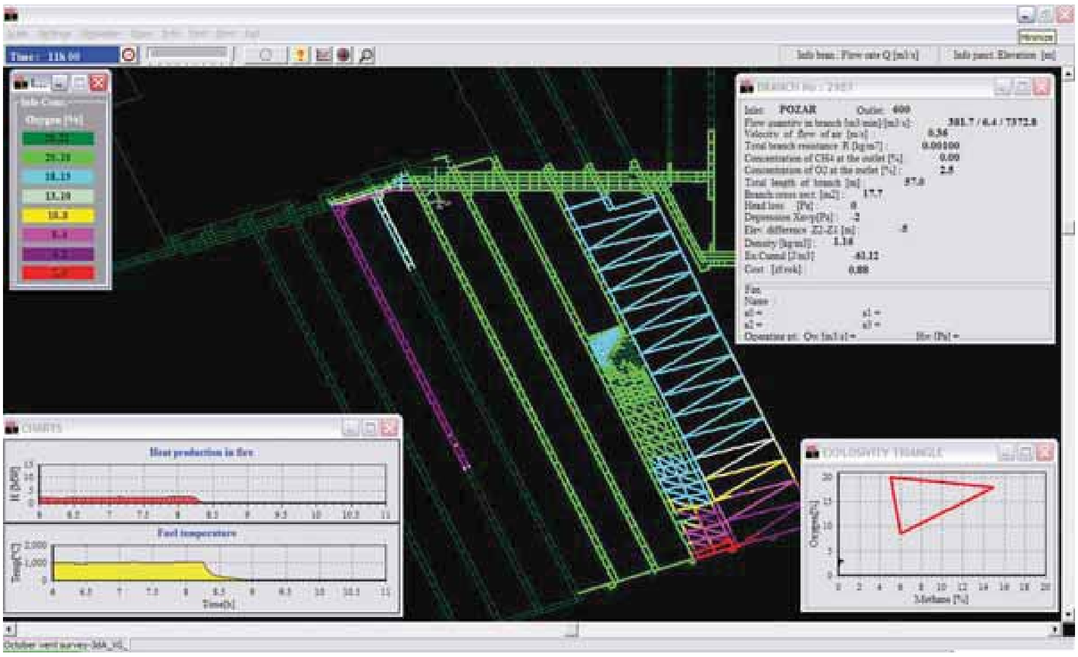


Figure 7.19 O_2 distribution at 660 minutes

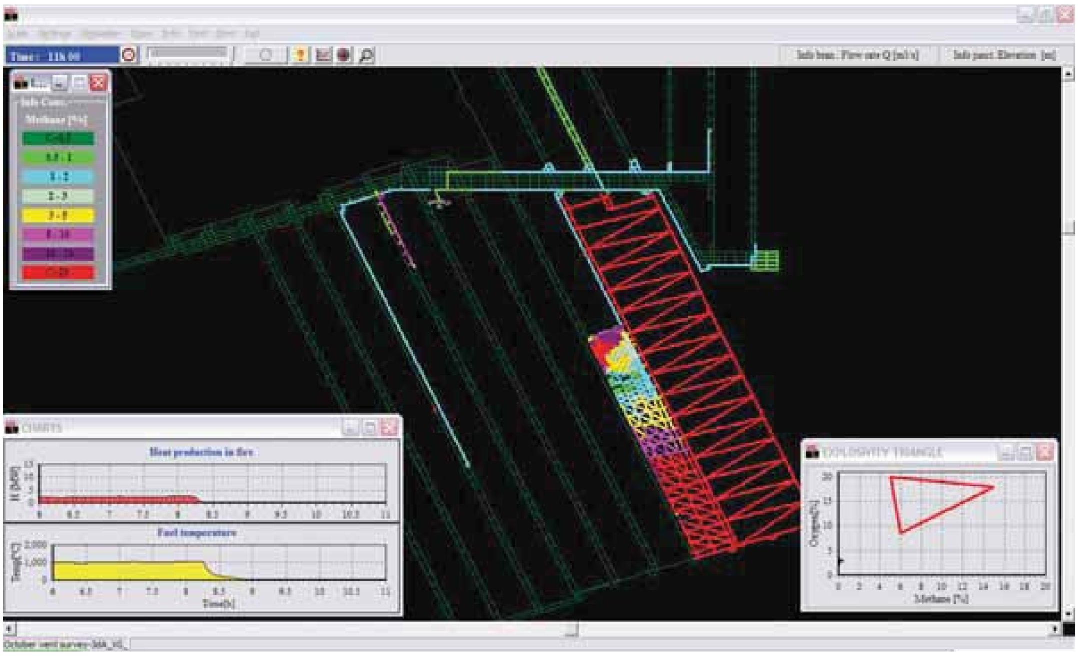


Figure 7.20 Methane (from other sources) distribution after 660 minutes

Summary *With GAG running Fire intensity insignificant and oxygen level outbye fire at less than 2.5 percent at 11hours.*

7.8. Oaky North Fire Inertisation Scenario 5B

Scenario *Dev in 7 MG at 26 ct (100m pillar). Eimco vehicle fire at 500m outbye of the face. Face 2.2 m^3/tonne CH_4. October 2005*

Inertisation Strategy *GAG at Highwall Portal D; Close Highwall B and C Hdg portal doors; Close B Hdg 35 – 36, C Hdg 35 – 36 and D Hdg 35 – 36; Seal transport drift*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Highwall Portal D entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO and CH_4 Gas sensors in Dev 7 TG 2-3 ct panel returns, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.

Simulation

Step 1 Time 0 – 15 minutes: 200 litres diesel fuel is burning; Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 2 Time 15– 30 minutes: Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 3 Time 30 – 60 minutes: 200 litres fuel is burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

Step 4 Time 60 – 120 minutes: an additional 20m length of coal pillar equivalent of a total 47m additional burning; Simulate 47m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

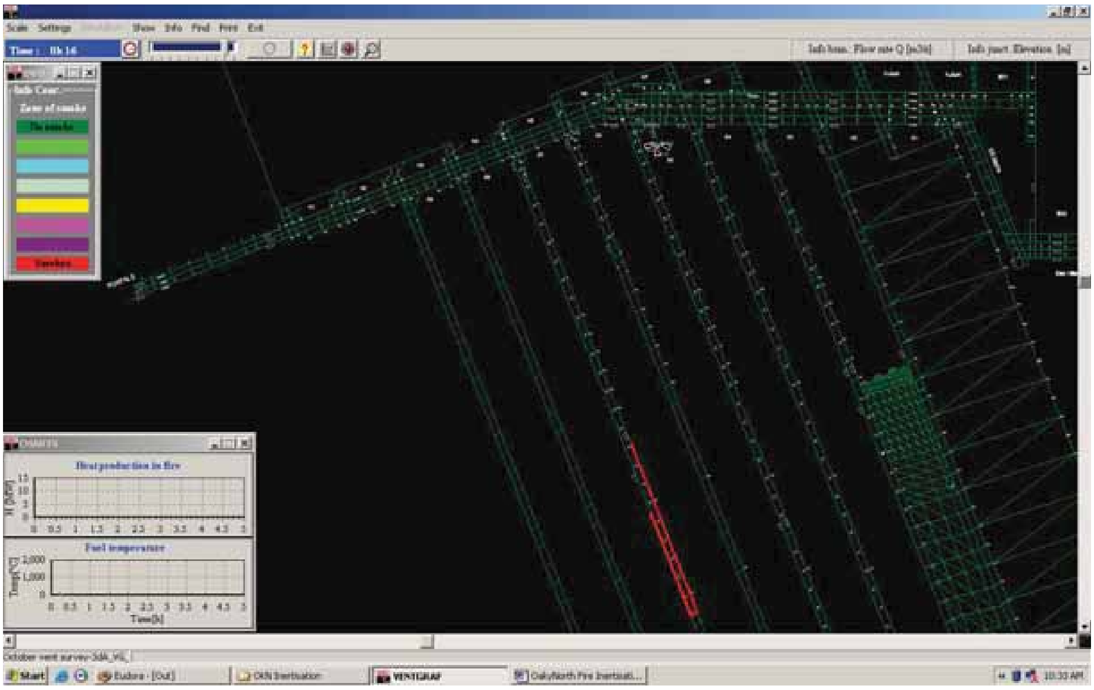


Figure 7.21 Smoke distribution after 15 minutes

Control Assume all mining crewmembers out of mine.

IMT team formed; Decision made to introduce high flow inertisation – GAG as soon as all crews evacuated out of mine.

Step 5 Time 120 – 300 minutes: Additional 20 m entry length coal caught on fire. Simulate 67m length oil fire over entry width; time constant 120s, intensity 7. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control.

Step 6 Time 300 - ? minutes: Continue simulating 67m length fire over entry width; time constant 120s, intensity 7 $CO:CO_2$ = 0.1. (assume H_2 = CO level);

Control High flow inertisation GAG unit has arrived and is set up

At 300 minutes: GAG has been set up at the D Portal entry and emergency door closed, R=10. B and C Hdg portal doors closed R = 10 and R = 1 respectively.

Close C Hdg 35 – 36 Brattice seal R=1; Close B Hdg 35 – 36 Prep seal R=2

Close D Hdg 35 – 36 Prep seal R=2; Shut down No 1 fan; fan louvre doors closed R=20; Shut down No 2 fan; fan louvre doors closed R=20 and examine No 3 fan curve operating point; Initiate GAG.

After 570 minutes improve C Hdg portal seal to R=10;

Fire not able to be extinguished.

After 720 minutes shut down No 3 fan.; Seal transport drift R=10;

Soon after, explosion occurs.

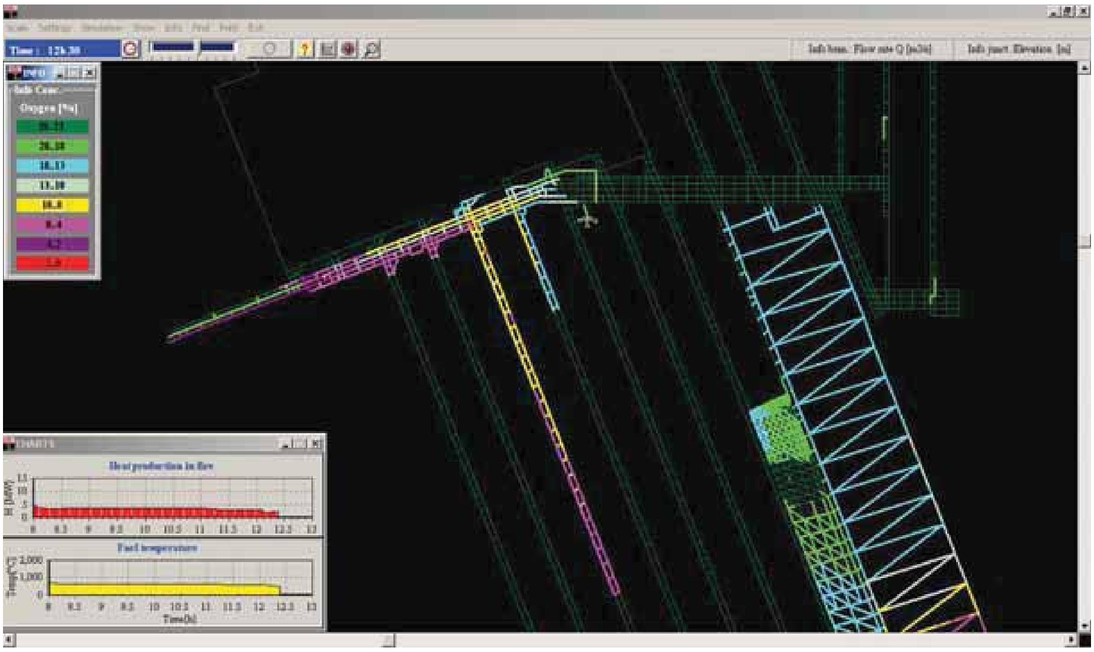


Figure 7.22 O_2 distribution at 750 minutes

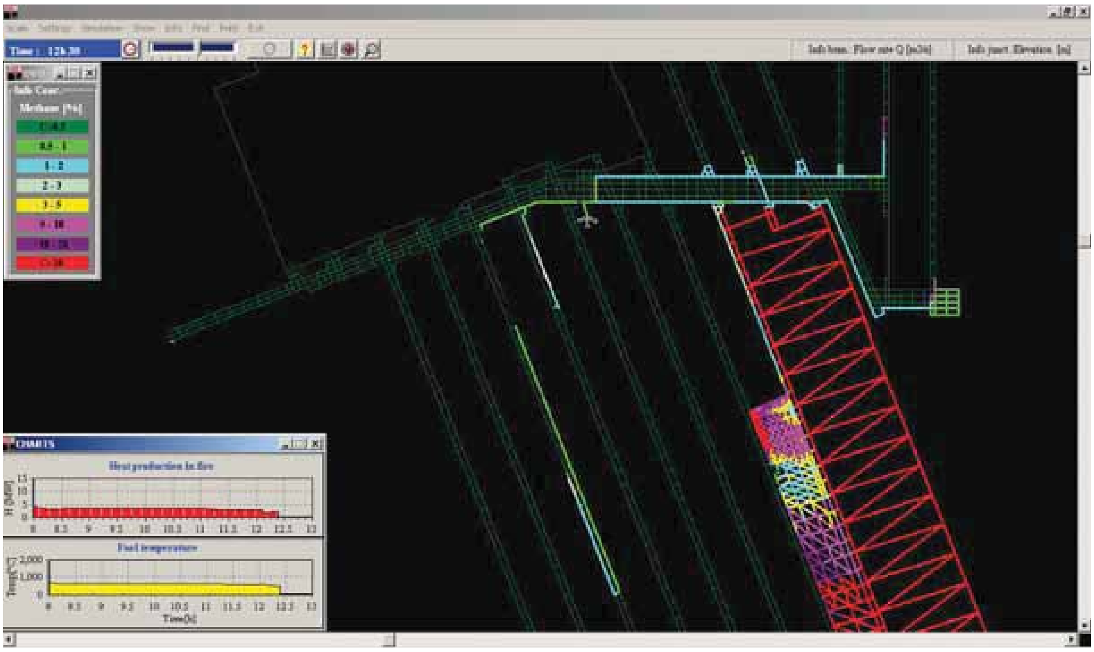


Figure 7.23 Methane (from other sources) distribution after 750 minutes

Summary *Inertisation strategy not effective. Explosion occurred soon after No 3 fan was shut down.*

7.9. Oaky North Fire Inertisation Scenario 5C

Scenario *Dev in 7 MG at 26 ct (100m pillar). Eimco vehicle fire at 500m outbye of the face. Face 2.2 m^3/tonne CH_4. October 2005*

Inertisation Strategy *GAG at Highwall Portal D; Close Highwall B and C Hdg portal doors*

Sections

1. *LW 3 at 13ct11/05*
2. *Dev 301 MG at 5ct 11/05*
3. *Dev South MG 7 at 26ct 11/05*
4. *Dev Stone Development road header (contractor) MG6 at 8ctat 11/05*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation GAG set at Highwall Portal D entry
- CO alarms set at 4.4ppm high alarm and 8.8ppm high high alarm.
- Methane sources on LW face of 160 x 3 litres/s simulated as sources at 20, 74 and 128 chocks.
- Negligible intake methane.
- Methane output at 301 Dev face of 200 litres/s
- Methane output at South MG 7 face of 100 litres/s.
- Methane from 302 MG standing face of 100 litres/s
  Levels from measurements in mine November 2005.
- CH_4 source of 600 litres/s from LW goaf at TG end of face
- CO and CH_4 Gas sensors in Dev 7 TG 2-3 ct panel returns, and
  CO Gas sensors set at points either side of bottom of Ventilation Shaft.

Simulation

Step 1 Time 0 – 15 minutes: 200 litres diesel fuel is burning; Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire is very unstable and not under control.

Step 2 Time 15– 30 minutes: Simulate 7m length fire over entry width; time constant 120s, intensity 10 $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Step 3 Time 30 – 60 minutes: 200 litres fuel is burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

Step 4 Time 60 – 120 minutes: an additional 20m length of coal pillar equivalent of a total 47m additional burning; Simulate 47m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control

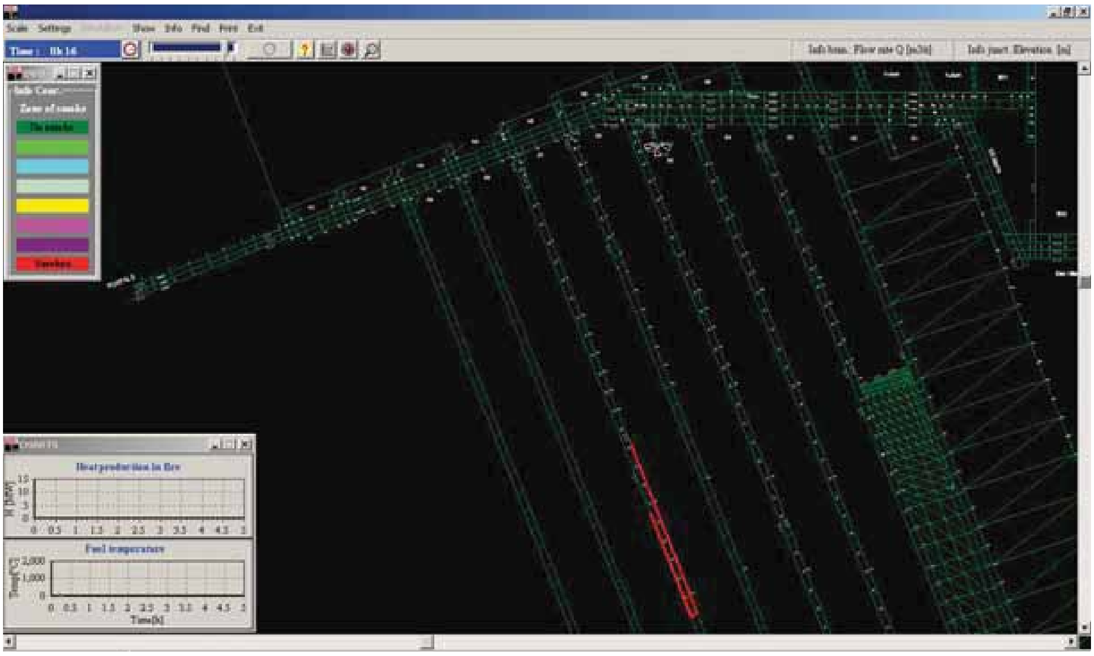


Figure 7.24 Smoke distribution after 15 minutes

Control Assume all mining crewmembers out of mine.

IMT team formed; Decision made to introduce high flow inertisation – GAG as soon as all crews evacuated out of mine.

Step 5 Time 120 – 300 minutes: Additional 20 m entry length coal caught on fire. Simulate 67m length oil fire over entry width; time constant 120s, intensity 7. $CO:CO_2 = 0.1$

Step 6 Time 300 - ? minutes: Continue simulating 67m length fire over entry width; time constant 120s, intensity 7 $CO:CO_2 = 0.1$. (assume H_2 = CO level); Fire out of control

Control High flow inertisation GAG unit has arrived and is set up; At 300 minutes: GAG has been set up at the D Portal entry and emergency door closed, R=10. Initiate GAG.

At 330 mins shut down one main fan; fan louvre doors closed R=20
Due to less ventilation air methane levels have increased in the return air.
At 360 mins shut down second main fan; fan louvre doors closed R=20
At 360 B and C Hdg portal doors closed R = 10 and R = 1 respectively.

Due to less ventilation air methane levels have increased in the return air up to 3.5%.

Oxygen level in Mains high due to air through intake drift, decision made to close the third fan to reduce oxygen level in air across fire.

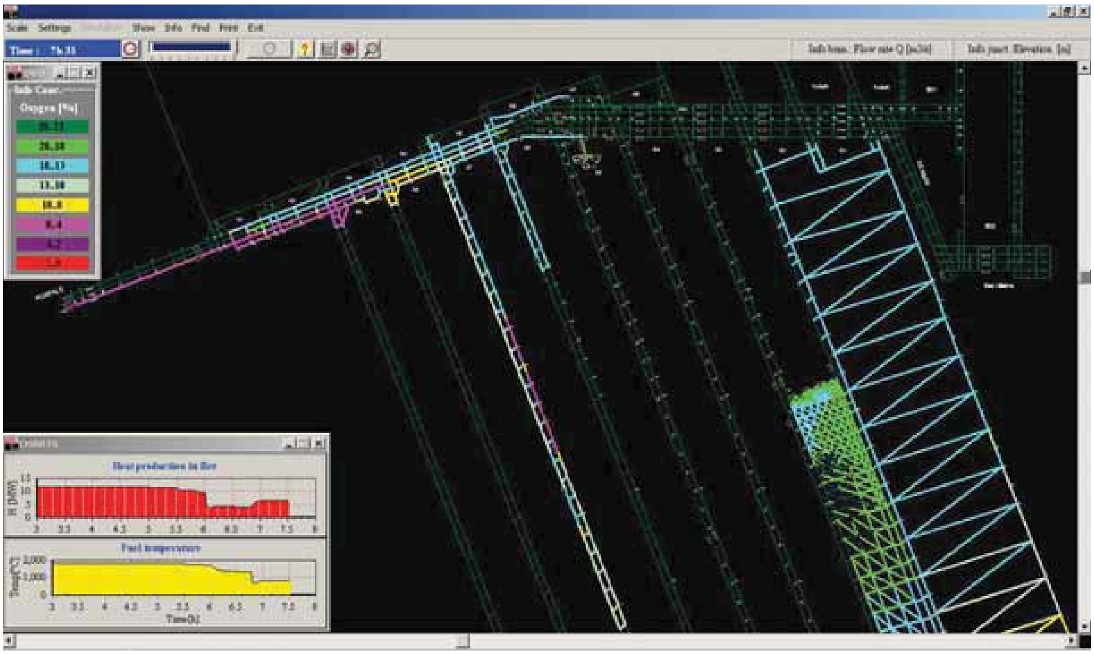


Figure 7.25 O_2 distribution at 450 minutes

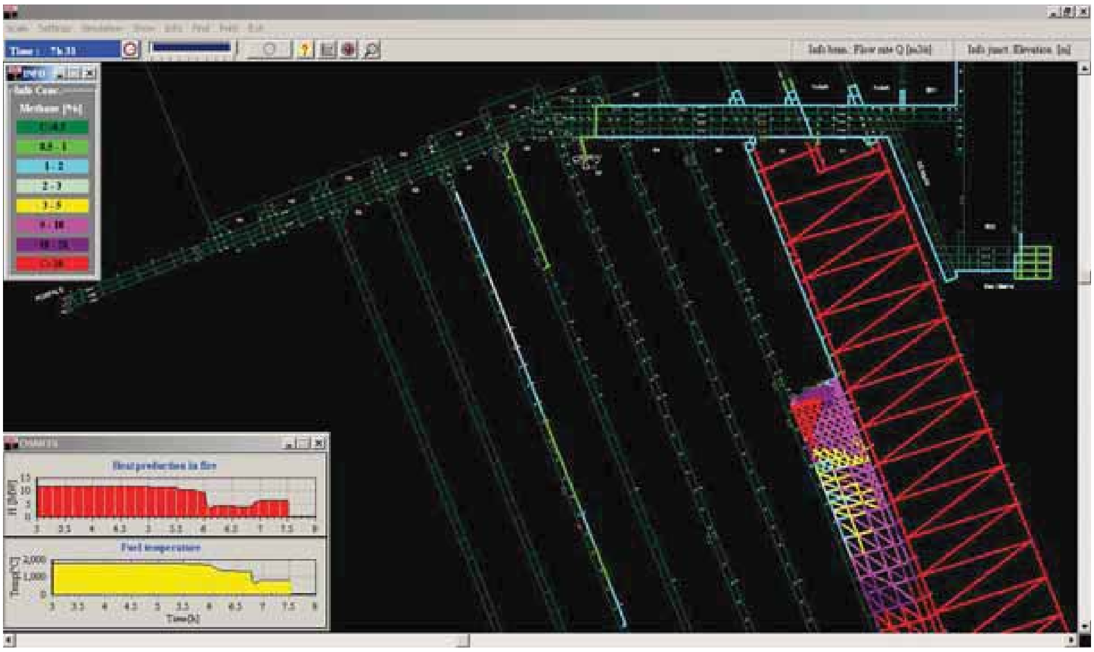


Figure 7.26 Methane (from other sources) distribution after 450 minutes

At 450 mins shut down the third main fan; soon after, explosion occurs as methane reverses flow across fire

Summary *Explosion occurred soon after No 3 fan was shut down.*

Table 7.2 Comparison of inertisation effects between original GAG operation and new segregation and/or GAG Docking Positions.

Scenario No	Fire Location	Fire Type	GAG Location	Segregation actions	Fan Actions	Outcomes
1 (original)	Mains C Hdg conveyor at 39ct	Belt (oil)	Transport Drift Portal	None	All shut down	With GAG running Fire intensity insignificant at 10 hours and oxygen level outbye fire at less than 2.9 percent. ***(Category B)***
1A	Mains C Hdg conveyor at 39ct	Belt (oil)	Transport Drift Portal	Mains C Hdg • Prep seal at B Hdg 35-36 R=2 • Open machine door at 37ct B-C • Brattice around belt structure C Hdg 35 – 36ct R=1 • Brattice around belt structure 37ct C – D R=1 • Prep seal at 36ct C - D R=2 • Prep seal at B Hdg 37-37A ct R=2 • Segregation stopping 38ct, 39ct and 40ct C-D	All shut down	With GAG running Fire intensity insignificant at 9 hours and oxygen level outbye fire at less than 2.5 percent. There is only a slight reduction in the time need to achieve satisfactory inertisation of the mine with extra segregation. ***(Category B)***

Scenario No	Fire Location	Fire Type	GAG Location	Segregation actions	Fan Actions	Outcomes
2 (original)	Goaf behind South Longwall 3	Goaf spon comb	Transport Drift Portal	None	All shut down	At 34 hours local reversal occurred to produce minor methane burn off. With the GAG running after 5 days there is no significant fire. Outbye the fire oxygen is 0.1 percent. (***Category C***)
2A	Goaf behind South Longwall 3	Goaf spon comb	Transport Drift Portal	Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down all fans at the same time; fan louvres for No1 & 2 closed.	All shut down	After 2 days with the GAG running, there is no significant fire. Outbye the fire the oxygen is 2.9 percent. (***Category B***)
2B	Goaf behind South Longwall 3	Goaf spon comb	Borehole (1m dia.) at MG 2 ct LW	Seal off SMG3 A and B Hdg 1-2ct Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down all fans at the same time; fan louvres for No1 & 2 closed.	All shut down	After 2 days with the GAG running, there is no significant fire. Outbye the fire the oxygen is 2.6 percent. (***Category B***)
2C	Goaf behind South Longwall 3	Goaf spon combustion	Borehole (1m dia.) at MG 2 ct LW	Seal off SMG3 A and B Hdg 1-2ct Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down No 1 & 2 fans; fan louvres closed. Close Transport Drift Doors after 2 days.	Shut down 2 fans with 1 fan running	After 2 days with the GAG running, there is no significant fire. Oxygen is 4.2 percent outbye the fire. By closing the Transport Drift Portal door, the oxygen level outbye the fire is reduced to 2.6 percent. A majority of the mine has methane levels of less than 3 percent except the sealed longwall panels and goaf. (***Category B***)

Scenario No	Fire Location	Fire Type	GAG Location	Segregation actions	Fan Actions	Outcomes
3 (original)	South LW 4 MG 22ct Tripper drive	Belt Fire	Transport Drift Portal	None	All shut down	Explosion occurred as soon all fans were turned off due to a reversal. ***(Category E)***
3A	South LW 4 MG 22ct Tripper drive	Belt Fire	Transport Drift Portal	Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down No 1 fan; fan louvre closed Shut down No 2 fan; fan louvre closed Shut down No 3 fan	All shut down	No improvement; Explosion occurred as soon all fans were turned off due to a reversal. ***(Category E)***
4 (original)	Belt tripper drive Mains 11ct C Hdg	Spillage of Coal	Transport Drift Portal	None	All shut down	With GAG running fire intensity insignificant at 24 hours and oxygen level outbye fire at less 5.4 percent. Face methane passing over fire potentially causing explosions. ***(Category C)***
4A	Belt tripper drive Mains 11ct C Hdg	Spillage of Coal	Highwall Portal B	Close Portal B Heading Emergency Door Close Portal C Heading Emergency Door Close Portal D Heading Emergency Door Seal transport drift	All shut down	With GAG running Fire intensity insignificant at 10 hours and oxygen level outbye fire at less 2.5 percent. ***(Category B)***

Scenario No	Fire Location	Fire Type	GAG Location	Segregation actions	Fan Actions	Outcomes
5 (original)	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle fire (diesel)	Transport Drift Portal	Close Portal B Heading Emergency Door Close Portal C Heading Emergency Door Close Portal D Heading Emergency Door	All shut down	Explosion occurs from localised reversal of methane over fire in SMG7. Inertisation eventually occurs, but with several methane explosions. ***(Category E)***
5A	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle fire (diesel)	Borehole (1 m dia.) at MG7 2 ct	Seal off SMG7 A and B Hdg 1-2ct Close Portal B Heading Emergency Door Close Portal C Heading Emergency Door Close Portal D Heading Emergency Door Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down No 1 fan; fan louvre closed Shut down No 2 fan; fan louvre closed Improve C Hdg portal seal	Fans 1 and 2 shut down.	With GAG running fire intensity insignificant at 11 hours and oxygen level outbye fire at less than 2.5 percent. ***(Category B)***
5B	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle fire (diesel)	Highwall Portal D	Close Highwall B and C Hdg portal doors Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down No 1 fan; fan louvre closed Shut down No 2 fan; fan louvre closed Shut down No 3 fan. Seal transport drift	All Shut down	At 720 mins shut down the third main fan and soon after, explosion occurs ***(Category E)***
5C	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle fire (diesel)	Highwall Portal D	Close Highwall B and C Hdg portal doors Close C Hdg 35 – 36 Brattice seal Close B Hdg 35 – 36 Prep seal Close D Hdg 35 – 36 Prep seal Shut down No 1 fan; fan louvre closed Shut down No 2 fan; fan louvre closed Shut down No 3 fan.	All shut down	At 450 mins shut down the third main fan and soon after, explosion occurs ***(Category E)***

7.10. Summary of Scenarios Examined and Alternative Inertisation Strategies

A study has examined the potential for simulation of the effects of inertisation on fires within a mine ventilation network. The project involved applying the VENTGRAPH mine fire simulation software to preplan for situations created by mine fires. As an introduction some general conclusions from relevant work undertaken to date at a range of Australian coal mines is discussed.

Priority fire locations at mines with VENTGRAPH simulation models developed in an ACARP research project entitled "Mine Fire Simulation in Australian Mines using Computer Software" have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation). A review of 35 scenarios showed that there was no fire examined that achieved the situation in which GAG docking inerted the simulated fire to aid recovery in a timely manner. Further, only 20 percent of scenarios showed a situation in which while the inertisation product went straight to the fire site even though it arrived with significant dilution from other ventilation air or leakage through stoppings.

Other introductory sections examined issues with borehole location and sizing for delivery of GAG output and the influence of stopping leakage on GAG exhaust dilution in parallel intake airways

The principal purpose of this study is examination of Oaky North case study priority fires selected from across the pit layout with two in the mains, two in longwall panel gateroads and one in the newly formed longwall goaf. GAG inertisation strategies were examined for the five cases and details of the development of the individual scenarios are set down in chapter 4. Following this in chapter 5 each case scenario study was re-examined to evaluate whether a better inertisation strategy was possible through adoption of either use of additional underground segregation to control the delivery of inert gas, or GAG relocation to an alternative portal docking station locations or the drilling of a new borehole to deliver inert gas more directly to the fire site.

7.10.1. Scenario 1

Scenario 1 examined a Mains belt fire. It was considered as an inertisation partial success (Category B) in that use of the GAG did cause some early stabilisation of the fire. Progressive turning off of the three main surface fans did in time cause the fire to be extinguished through combustion caused reduction of oxygen aided by the addition of inert gases. Seam methane emissions caused gas levels to build up in the panels however these did not recirculate across a Mains located fire.

The Scenario 1A reassessment of approaches to improve the inertisation strategy led to the decision to add segregation to the pit bottom area at the base of the Decline travel road. This area was very open and the objective was to channel inert gases from the travel road docking point into C Heading without loss into and mixing from Mains B and D Headings. The result was that with the GAG running and addition segregation stoppings and doors there was only a slight reduction in the time needed to achieve satisfactory inertisation of the mine with extra the additional segregation. Leakage through the significant number of additional doors and stoppings prevented a better outcome. The question of how these stoppings and doors could be installed or closed in the event of a major fire was not addressed. There was still a need to turn off the fans to achieve stabilisation. Turning off main surface fans in a gassy mine with a major fire is not a question to be decided lightly due to the complexity of the situation.

The alternative strategies of or GAG relocation to an alternative portal docking station locations or the drilling of a new borehole to deliver inert gas more directly to the fire site were not considered to be likely to give an improved situation. GAG docking at highwall portals B or C or D placed the unit further from the fire with more leakage and dilution through stoppings. A borehole into the Mains without significant segregation in the pit bottom area would not achieve advantage.

7.10.2. Scenario 2

Scenario 2 examined a spontaneous combustion fire in the longwall panel goaf on the MG side a few cut throughs back from the face It was considered as a situation where inertisation by itself would not help extinguish the fire in the goaf (Category C). Progressive turning off of the three main surface fans did after much time cause the fire to be extinguished through combustion caused reduction of oxygen aided by the addition of inert gases which reach the fire after alteration of the pit ventilation. With fans off seam methane emissions caused gas levels to build up in the panels and although the VENTGRAPH simulation did not show these recirculating across the fire this could be a dangerous situation.

The alternatives examined for improved inertisation focused on utilisation of underground segregation, use of a borehole for GAG docking and surface segregation. The aim was to review various possible alternative strategies to establish the optimum approach.

The first alternative (Scenario 2A) examined the effect of stopping air from Highwall Portals B, C and D entering the pit and diluting the GAG exhaust coming down the Transport Drift. This was achieved by closing Mains headings and at the same time stopping all surface main fans. At least some fan stoppage was necessary or stalling would have occurred. This strategy improved the stabilisation of the goaf (Category B) significantly although with fans off seam methane emissions caused gas levels to build up in the panels which could potentially lead to a dangerous situation.

The second alternative (Scenario 2B) examined the effect of placing a borehole near the beginning of MG3 leading to the goaf with the fire. The GAG was docked on the borehole and doors closed on the Main Gate headings. As in Scenario 2A Highwall Portal air was closed off at the same time as all fans were stopped. This strategy improved the stabilisation of the goaf (Category B) significantly and marginally better than that achieved in Scenario 2A although the issue of having fans off needs to be recognised.

The third alternative (Scenario 2C) again examined the effect of placing a borehole near the beginning of MG3 leading to the goaf with the fire and with the GAG was docked on the borehole and doors closed on the Main Gate headings. Again as in Scenario 2A and 2B Highwall Portal air was closed off but this time only two fans were stopped with one still operating to keep seam gas accumulations diluted. A Category B outcome was achieved but marginally less efficiently than in Scenarios 2A or 2B. The scenario fire simulation was then continued with a final step of closing the Transport Drift Emergency Door. A Category B outcome was still achieved and with less air in the pit the air reaching the goaf heating had reduced oxygen to the level achieved in Scenario 2B but with the added advantage that one fan was still operating to keep seam gas accumulations diluted. Scenario 2C was the best outcome of this progressive sequence as with one fan still operating seam methane emissions were not observed to build up in the panels and so a potentially dangerous situation was avoided.

7.10.3. Scenario 3

Scenario 3 examined a fire in Longwall 4 MG at a 22 cut through tripper drive. It was considered as an inertisation failure (Category E) in that use of the GAG did not cause stabilisation of the fire, as with dilution of inert exhaust at pit bottom little low oxygen air will effectively reach the fire. Progressive turning off the three main surface fans led, after 8 hours, to reversal of face air carrying explosible concentrations of methane over the fire which caused a large explosion.

Scenario 3A reassessment of approaches to improve the inertisation strategy led to the decision to examined the effect of stopping air from Highwall Portals B, C and D entering the pit and diluting the GAG exhaust coming down the Transport Drift. This was achieved by closing Mains headings and at the same time stopping all surface main fans. This did not improve the situation and it was still considered as an inertisation failure (Category E) as reversal of face air carrying explosible concentrations of methane over the fire still caused a large explosion.

7.10.4. Scenario 4

Scenario 4 examined a fire in the Mains at 11 cut through C Heading tripper drive. The scenario was found to be a situation where inertisation would not help extinguish the fire

(Category C). The fire is ventilated by intake air from the Highwall portals and inert gases added at the Decline Travel Heading docking point will not reach the fire location. Progressive turning off of the three main surface fans did not alter this situation. Seam methane emissions caused gas levels to build up in the panels however these did not recirculate across this Mains located fire.

The Scenario 4A reassessment of approaches to improve the inertisation strategy led to the decision to dock the GAG unit at the Highwall Portal B Heading. This position could direct inert gas directly into B and C Headings and onto the fire and led to a satisfactory outcome (Category B). To avoid dilution portal doors in B, C and D were progressively closed and to avoid stalling Main surface fans were progressively turning off. The emergency door on the Decline Travel Heading was also closed to achieve the minimum time for stabilisation of the fire.

7.10.5. Scenario 5

Scenario 5 examined a vehicle fire in Development MG. It was considered as an inertisation failure (Category E) in that use of the GAG did not cause stabilisation of the fire before several methane explosions with reversal of face air carrying explosible concentrations of methane over the fire. Closure of emergency doors on Highwall portals B, C and D was needed to rebalance the pit ventilation to bring the inert gases to the entrance to the panel. To avoid stalling fans needed to be turned off progressively as with dilution of inert exhaust at pit bottom little low oxygen air will effectively reach the fire.

The alternatives examined for improved inertisation focused on use of alternative docking points. The aim was to review various possible alternative strategies to establish the optimum approach.

The first alternative (Scenario 5A) examined moving GAG docking from the Transport Drift to docking at a Borehole at the beginning of the Development panel MG. Doors at entry to the Panel MG were closed. Highwall Portals B, C and D were closed as well as possible to stop air entering the pit and diluting the Borehole GAG exhaust. At the same time Mains headings were sealed underground and two surface main fans stopped. At least some fan stoppage was necessary or stalling would have occurred. This strategy forced borehole inert gases across the fire. Additional sealing of the Conveyor drift at Portal C was required and as a consequence with minimum air being able to leak into the panel successful stabilisation of the fire occurred (Category B).

The second alternative (Scenario 5B) examined the effects of docking the GAG at Highwall D Heading Portal. The Emergency doors were closed at Highwall B and C Portals as well as further inbye in the Main Gate headings. Progressive turning off the three main surface fans, with the third stopped when the Transport Drift was sealed, led, after 12 hours, to reversal of face air carrying explosible concentrations of methane over the fire which caused a large explosion. This approach was considered as an inertisation failure (Category E) in that use of the GAG did not cause stabilisation of the fire before several methane explosions with reversal of face air carrying explosible concentrations of methane over the fire.

The third alternative (Scenario 5C) was similar to Scenario 5B as it examined the effects of docking the GAG at Highwall D Heading Portal. As before Emergency doors were closed at Highwall B and C Portals as well as further inbye in the Main Gate headings. However the transport drift remained open as all three main surface fans were progressively turned off. This led, after 7.5 hours, to reversal of face air carrying explosible concentrations of methane over the fire which caused a large explosion. This approach was again considered as an inertisation failure (Category E) in that use of the GAG did not cause stabilisation of the fire before several methane explosions with reversal of face air carrying explosible concentrations of methane over the fire.

7.11. Conclusions and Recommendations

The principal part of this study of inertisation strategies has been to examine priority fire locations and best approaches to stabilising of fires. It was determined that Oaky North has a mine layout under which some improvements could be made to inertisation strategies in the event of a major fire

Based on the results from the actions described in Chapter 6 scenarios have been re-simulated with new approaches to inertisation. Outcomes for these re-simulated alternative scenarios were compared with the original simulation results as described in previous sections. A summary of the comparisons is shown in Table 7.3.

General actions that can be undertaken to improve the effectiveness of an existing inertisation situation in an underground ventilation network, apart from general improvement to ventilation control devices, can be drawn from the following.

1. Maintain use of existing inertisation docking station but with use of additional underground segregation to control the delivery of inert gas.
2. Try alternative Portal docking station locations through use of existing portals or installation of new.
3. Try alternative Portal docking station locations with additional underground segregation.
4. Drill new borehole to deliver inert gas more directly to the fire site.

The best inertisation strategy as determined from alternative simulation exercises for the five priority fire locations is summarised in Table 7.3.

Fire Number 1 gave an outcome in which this Mains fire was stabilised by turning off fans and allowing oxygen to be consumed to eventually lead to extinguishment. The mine's pit bottom area is very open. The alternative strategy simulated of segregating various headings in the pit bottom area was complicated by the number of actions required. Placement of stoppings and closing of underground doors is only mildly effective as all these structures will leak and so allow a fire to keep burning. Turning off all fans poses high risk issues in a gassy mine; however in this scenario the fire is in the Mains and so panel flow reversals of methane laden air was not presented as an issue. The conclusion was that GAG inertisation of this fire did not present a satisfactory strategy, not was the approach of placement of ventilation control devices. The traditional approach of sealing the mine after turning off fans provided a solution.

Table 7.3 Summary of optimum outcomes from the five fire simulation exercises

No	Fire Location	Fire Type	GAG Location	Fan Action	Outcome	Category
1	Mains C Hdg conveyor at 39ct	Belt (oil)	Transport Drift Portal	All shut down	Fire stable at 10 hours	B
1A	Mains C Hdg conveyor at 39ct	Belt (oil)	Transport Drift Portal	All shut down	Fire stable at 9 hours	B
2	Goaf behind SLW3	Goaf spon comb	Transport Drift Portal	All shut down	Fire stable at 120 hours, gas over fire possibility	C
2C	Goaf behind SLW3	Goaf spon comb	Borehole (1m dia.) at MG3 2 ct	2 down 1 running	Fire stable at 48 hours	B
3	South LW 4 MG 22ct Tripper drive	Belt (oil)	Transport Drift Portal	All shut down	Reversal, gas explosion	E
3A	South LW 4 MG 22ct Tripper drive	Belt (oil)	Transport Drift Portal	All shut down	Reversal, gas explosion	E
4	Belt tripper drive Mains C Hdg, 11ct	Coal spillage	Transport Drift Portal	All shut down	Fire stable at 24 hours, gas over fire possibility	C
4A	Belt tripper drive Mains C Hdg, 11ct	Coal spillage	Highwall Portal B	All shut down	Fire stable at 10 hours	B
5	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle (diesel)	Transport Drift Portal	All shut down	Reversal, gas explosion	E
5A	Dev in 7 MG at 26 ct B Hdg	Eimco vehicle (diesel)	Borehole (1m dia.) at MG7 2 ct	2 down 1 running	Fire stable at 11 hours	B

Fire Number 2 examined how a longwall goaf fire could be stabilised by use of a borehole which delivered directly to the fire seat. A good quality seal was needed to avoided dilution of the GAG exhaust. In effect the fire affected panel was segregated from the rest of the mine which allowed one fan to continue operating (use of more led to excessive leakage into the panel) maintaining safe ventilation of the mine. Sealing of Mains headings and the Transport Drift assisted the process. Other alternatives could eventually stabilise the fire but required turning off of all main surface fans.

Fire Number 3 was another example where GAG inertisation was not effective. Direct delivery of exhaust to the site was not effective due to dilution; segregation remediation did not materially assist because of leakage. Turning off fans led immediately to ventilation reversal in various panels due to air buoyancy and explosions form face gas flowing under natural ventilation pressure across the fire.

Fire Number 4 examined a fire which could not be inertised using the traditional Travel Drift GAG docking point. It focused on how a Mains heading fire could be stabilised by use of an alternative GAG docking portal. Appropriate segregation allowed direct delivery of exhaust to the fire location. However because the fire was in the Mains effective segregation was not easy and so all surface main fans had to be turned off to give a timely result.

Fire Number 5 again examined a fire which could not be inertised using the traditional Travel Drift GAG docking point. The fire was in a development heading and the best outcome was to use a panel borehole to direct exhaust directly to the fire seat. A good quality seal was needed to avoided dilution of the GAG exhaust. As with Fire Number 2 the fire affected panel was segregated from the rest of the mine which allowed one fan to continue operating (use of more led to excessive leakage into the panel) maintaining safe ventilation of the mine. Sealing of all Portals assisted the process. Other alternatives could eventually stabilise the fire but required turning off of all main surface fans.

In conclusion these fire simulation exercised have shown that some priority Oaky North fires can be stabilised through GAG inertisation strategies. The Number 2 goaf fire strategy developed is a case in point where use of a panel borehole with careful segregation allowed a relatively fast outcome to be achieved. The Number 5 development heading fire was similar in that a borehole GAG delivery gave the best outcome. Both these were achieved with one surface fan operating and maintaining minimum pit ventilation and seam methane dilution. The Number 4 fire, a Mains belt fire, utilised the GAG positively through use of an alternative Portal for docking.

However Number 1 (a Mains belt) and Number 3 (Development heading belt) fires were placed such that an inertisation strategy was not effective because pit layout means excess dilution affects the GAG exhaust quality which can be brought to the fire.

Recommendations arising from the analyses are as follows:

- GAG docking stations should be fabricated for all ventilation intake openings to the mine. The existing apparatus at the Travel Decline should be supplemented by docking points at the Highwall portals, any pit boreholes of appropriate diameter and future main shafts. In effect each docking point can deliver to a restricted geographic zone within the pit; multiple points allow he appropriate point to be utilised.
- Segregation strategies simulated at pit bottom areas have shown that distribution of inert gases to separate Mains headings can be improved. They were useful for fires located inbye from the pit bottom in the Mains but were less effective for the fires located a long way further inbye and in the longwall production and development panels (due to increasing dilution through stoppings).
- It is recommended that a borehole with a diameter of at least 1 m should be considered at the beginning of each panel for potential delivery of inert gases to each longwall production or development face. These boreholes can also be used for other purposes such as delivery of ballast or emergency extrication of people out of the mine. They may be used for other services. Incorporation of remote controlled doors should be considered to give control over which gateroad should be used to carry the inert gases into the panel.

These fire simulation exercises have demonstrated that it is possible to efficiently evaluate possible inertisation strategies appropriate to a complex mine layout extracting a gassy seam and determine which approach strategy (if any) can be used to stabilise a mine in a timely fashion.

8. CASE STUDIES OF FIRE SCENARIOS AT OAKY NO 1 MINE

Scenarios developed for Oaky Creek No 1 Colliery have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage following a fire.

A total of ten scenarios were simulated for Oaky Creek No 1 Colliery based on the mine fire simulation model developed from the ventilation arrangements in December 2005 as shown in Figure 8.1. These scenarios are as follows.

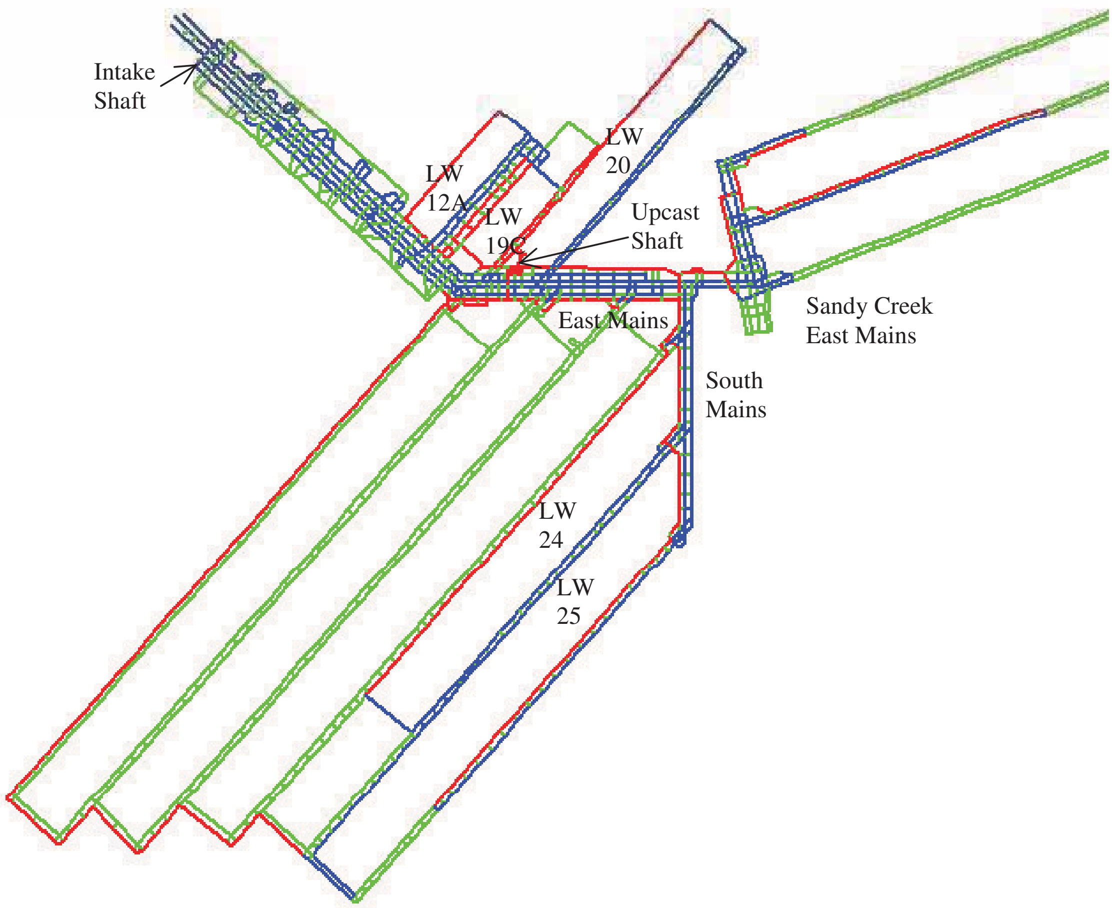


Figure 8.1 Ventilation arrangements at Oaky Creek No 1 Colliery in December 2005

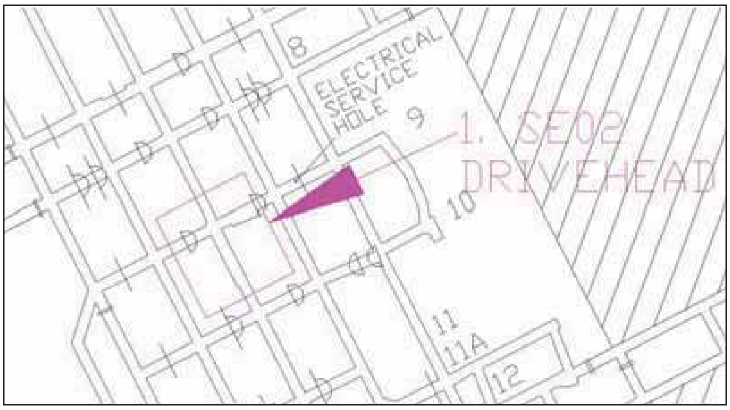
1. SE02 Drivehead - Main Dips C9-C10 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.
2. SE03 Drivehead - Main Dips C21-C22 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.
3. SE04 Drivehead - East Mains C5-C6 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.

4. SE05 Drivehead - South Mains C5-C6 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.
5. LW Drivehead - South Mains D8 to MG24 C2 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.
6. LW Face Friction Ignition - Longwall 24 face. This is modelled as an oil fire that develops into a coal fire.
7. LW Goaf Spontaneous Combustion - Longwall 24 Goaf heating.
8. MG25 Drivehead - South Mains D14 to MG25 C2 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.
9. MG26 Drivehead - Sandy Creek East Mains D6 to MG26 C1 - Belt Drivehead area that is segregated. This is modelled as an oil fire that develops into a coal fire.
10. Jiffy Drive 1 Drivehead - Sandy Creek East Mains C13 to C12 - Belt Drivehead area, which is segregated. This is modelled as an oil fire that develops into a coal fire.

Each of these scenarios is described and discussed in the following section.

8.1. Oaky Creek No 1 Fire Scenario 1

Scenario *In the "C" main dips at bottom of main dips at the C9-C10 (belt drive head area), hydraulic oil has caught on fire. SE02 Drivehead - Main Dips C9-C10 - Belt Drivehead area that is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW12A 80 l/s, LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 22 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Smoke reaches Longwall face at 50 minutes

Control Fire fighting control is suppressing oil fire

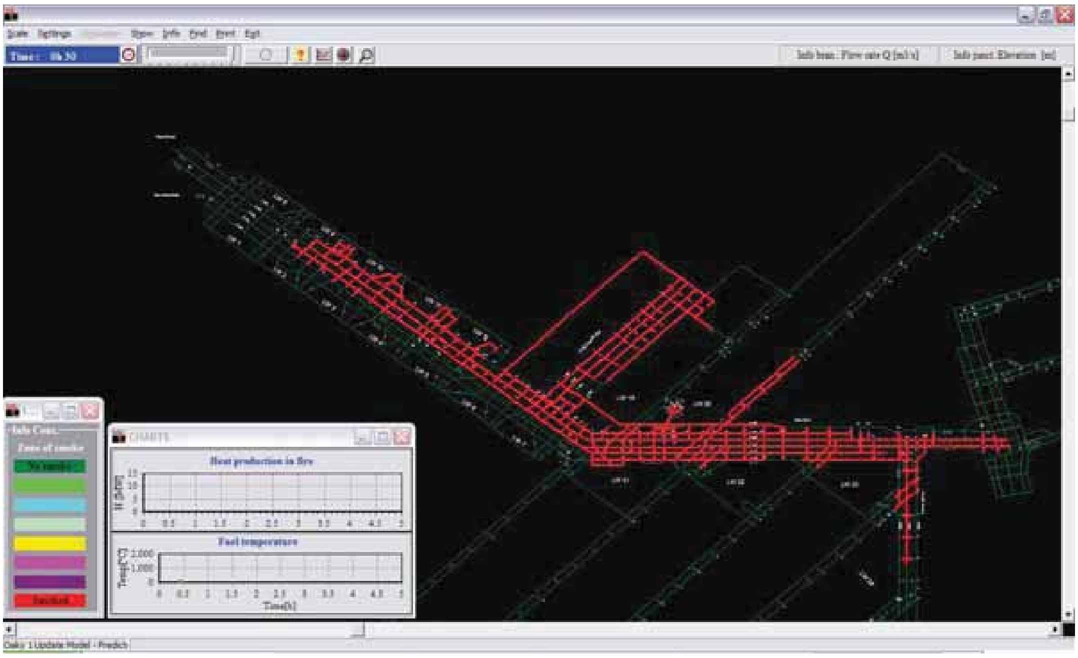


Figure 8.2 Smoke distribution after 30 minutes.

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2$ = 0.1. (assume H_2= CO level); fire very unstable and not under control.

Control Fire fighting ineffective within 120 minutes

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Examine fan curve operating point ; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 At 330 minutes: Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Step 7 At 360 minutes: Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip B Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 At 390 minutes: Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

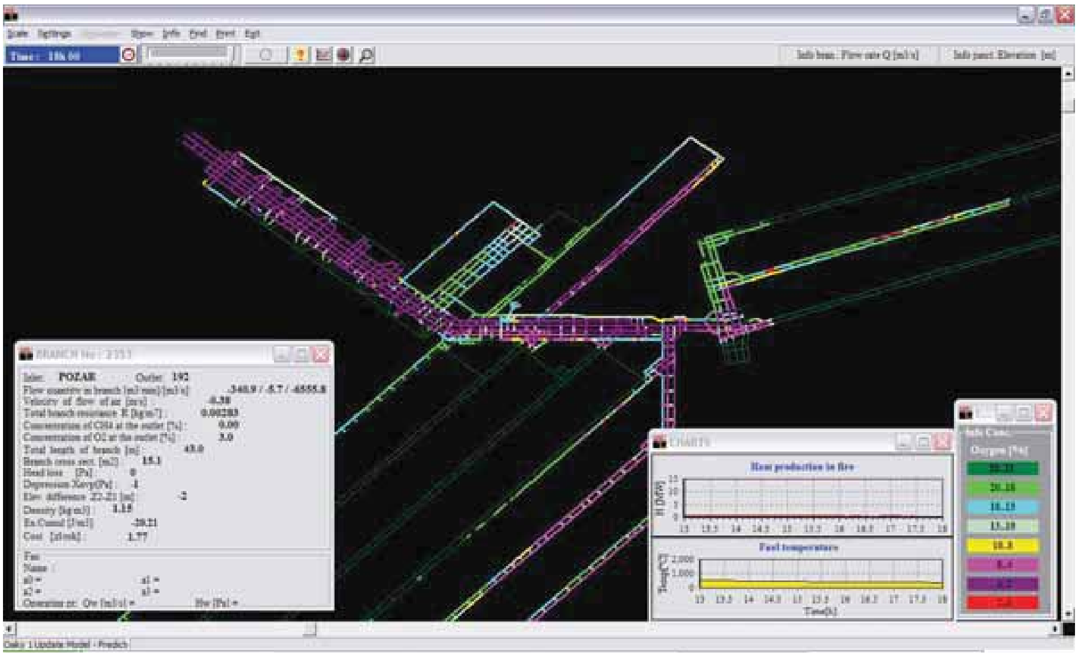
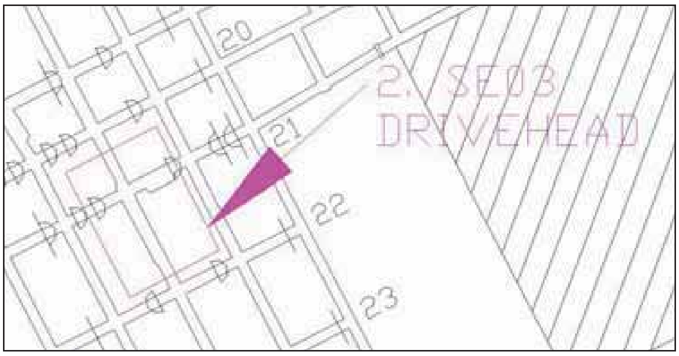


Figure 8.3 Oxygen distribution after 1080 minutes.

Summary *With GAG running Fire intensity insignificant at 18 hours and oxygen level outbye fire at less than 3.0 percent. However ventilation air reversal occurred across the fire after all fans stopped but no methane reversed across the fire.*

8.2. Oaky Creek No 1 Fire Scenario 2

Scenario *In the "C" main dips at bottom of main dips at the C21-C22 (belt drive head area), hydraulic oil has caught on fire. SE03 Drivehead - Main Dips C21-C22 - Belt Drivehead area which is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

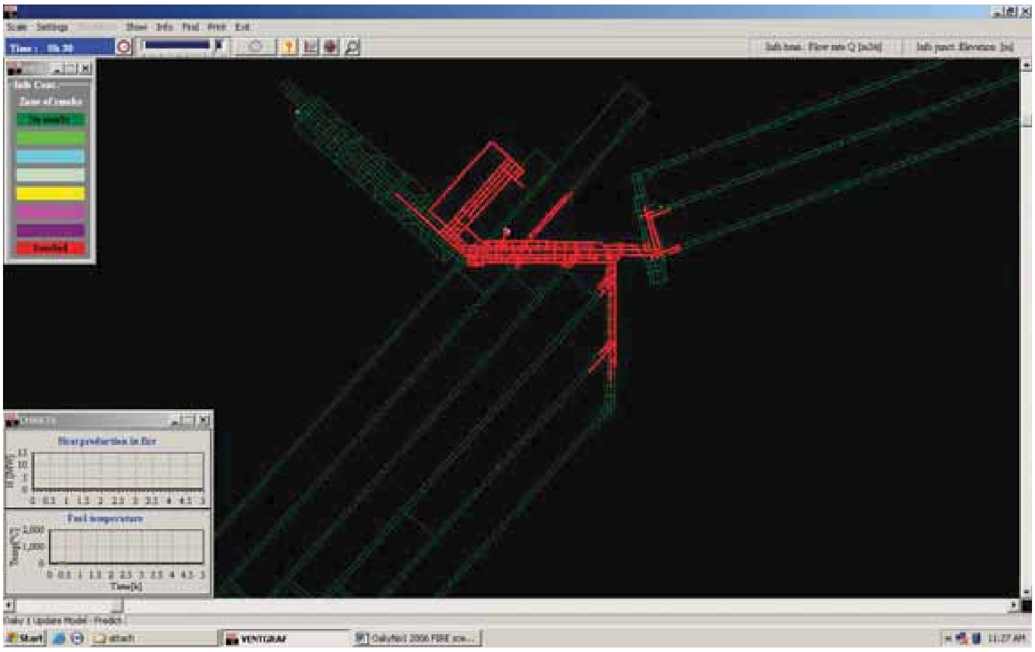


Figure 8.4 Smoke distribution after 30 minutes.

Smoke reaches surface at 15 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator.
Simulate 7m length fire over entry width; time constant 120s, intensity 10.
Smoke reaches MG26 face at 45 minutes
Smoke reaches Longwall face at 47 minutes
Smoke reaches MG25 face at 54 minutes

Control Fire fighting control is suppressing oil fire

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2= CO level); fire very unstable and not under control.

East Main CO concentration is less than 50ppm throughout.

Control Fire fighting ineffective within 120 minutes.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2 = 0.1$ (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 After 330 minutes Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Step 7 After 360 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip B Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

Reversal of air occurs across fire once the final fan is turned off.

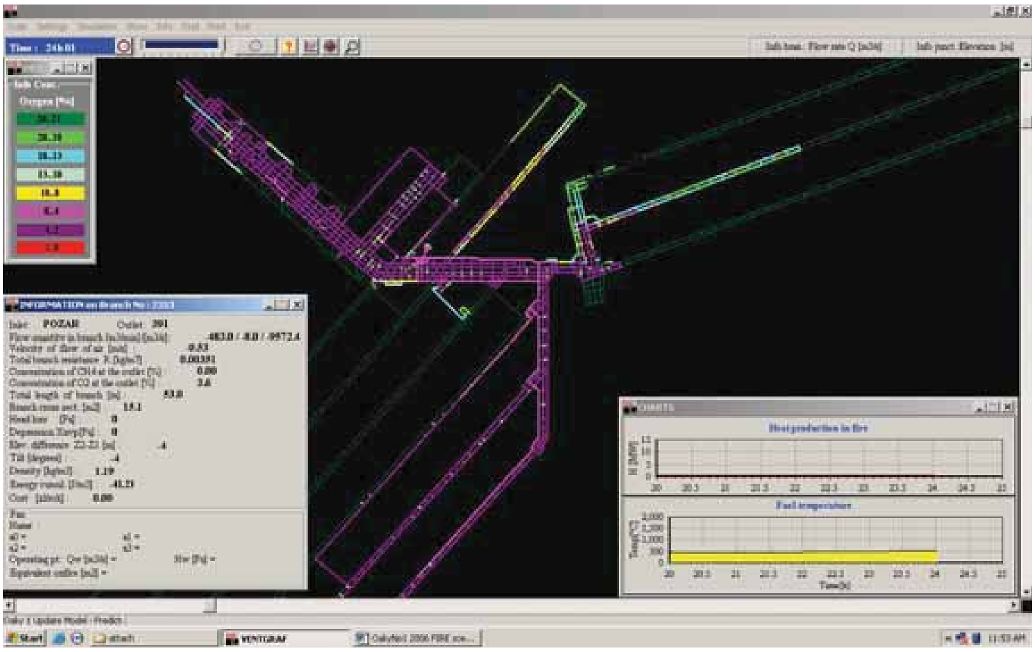
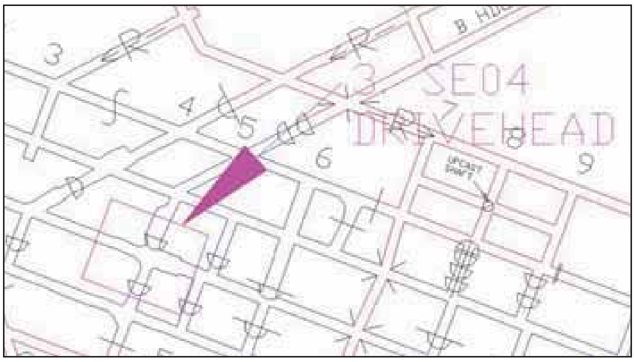


Figure 8.5 Oxygen distribution after 1440 minutes.

Summary *With GAG running Fire intensity insignificant at 24 hours and oxygen level outbye fire at less than 3.5 percent. However ventilation air reversal occurred across the fire after all fans stopped but no methane reversed across the fire.*

8.3. Oaky Creek No 1 Fire Scenario 3

Scenario *In the "C" main dips at bottom of main dips at the C5-C6 (belt drive head area), hydraulic oil has caught on fire. SE04 Drivehead - Main Dips C5-C6 - Belt Drivehead area which is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume $H_2 = CO$ level).

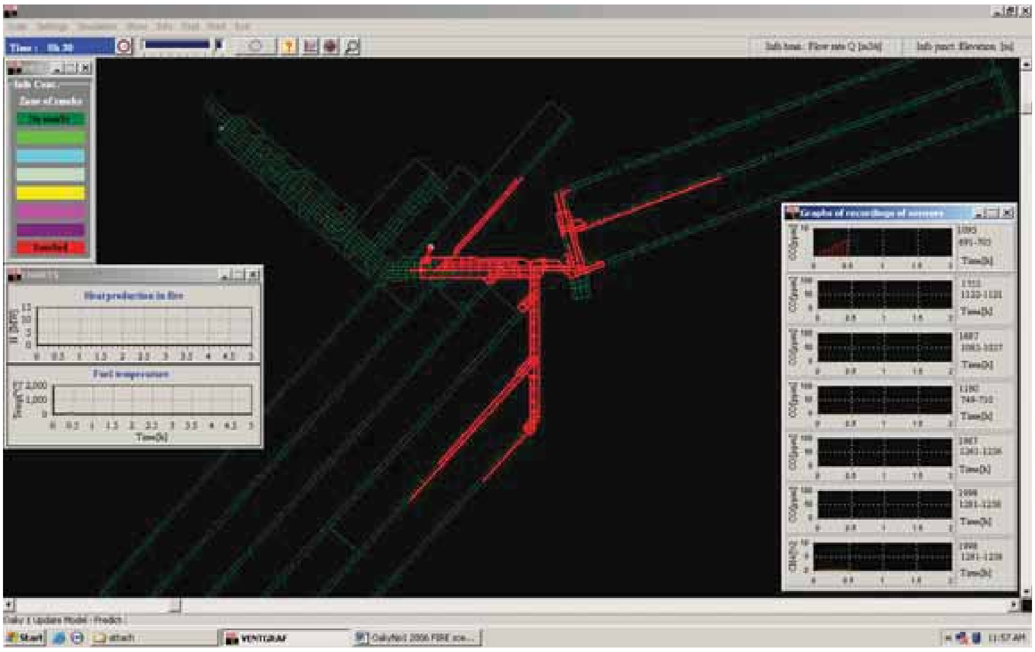


Figure 8.6 Smoke distribution after 30 minutes.

Smoke reaches surface at 5 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.
Smoke reaches Longwall face at 37 minutes
Smoke reaches MG26 face at 37 minutes
Smoke reaches MG25 face at 45 minutes

Control Fire fighting control is suppressing oil fire

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2= CO level); fire very unstable and not under control.

East Main CO concentration is less than 50ppm throughout, except C heading

Control Fire fighting ineffective within 120 minutes.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2 = 0.1$ (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Control Assess effectiveness of GAG
Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 After 330 minutes Shut down No 1 fan; fan louvre doors closed R=10.

Close Portal Dip A Heading Emergency Door R=10.

Step 7 After 360 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip B Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

Localised reversal occurs

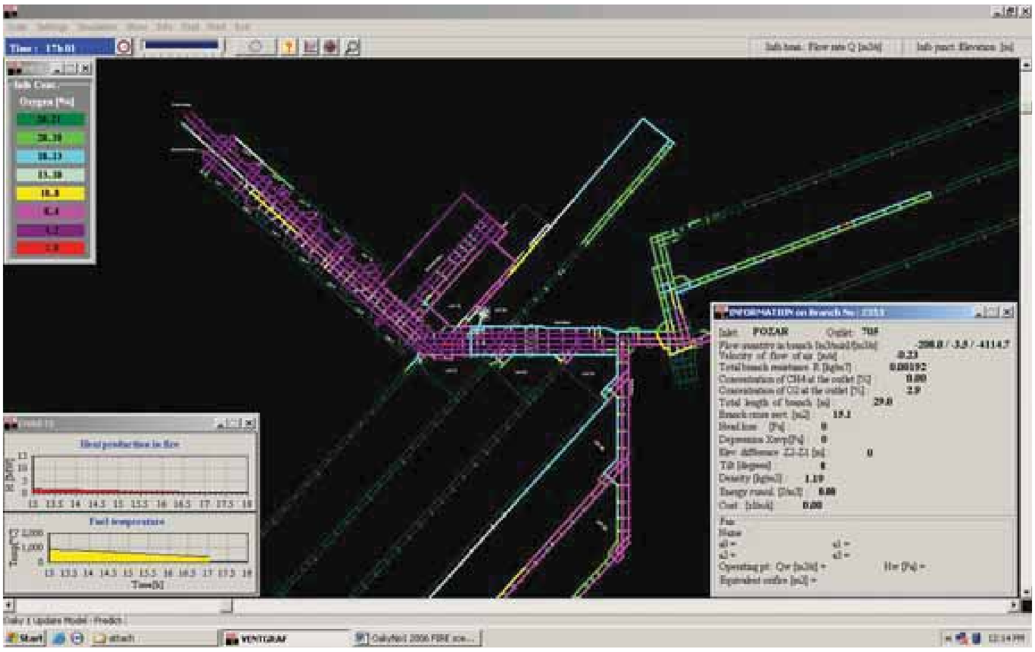
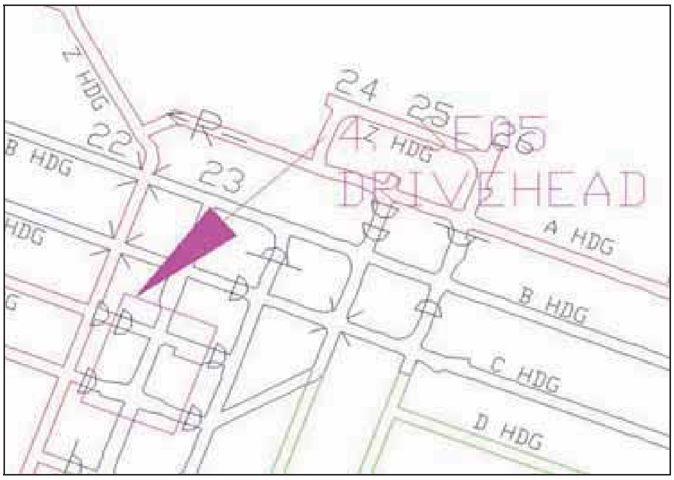


Figure 8.7 Oxygen distribution after 1020 minutes.

Summary *With GAG running Fire intensity insignificant at 17 hours and oxygen level outbye fire at less than 2.9 percent.*

8.4. Oaky Creek No 1 Fire Scenario 4

Scenario *In the East mains dips at bottom of main dips at the 23ct D-E (belt drive head area), hydraulic oil has caught on fire. SE05 Drivehead – East mains Dips 23ct D-E Belt Drivehead area which is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 15 minutes
Smoke reaches Longwall face at 26 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Smoke reaches MG25 face at 33 minutes

Control Fire fighting control is suppressing oil fire

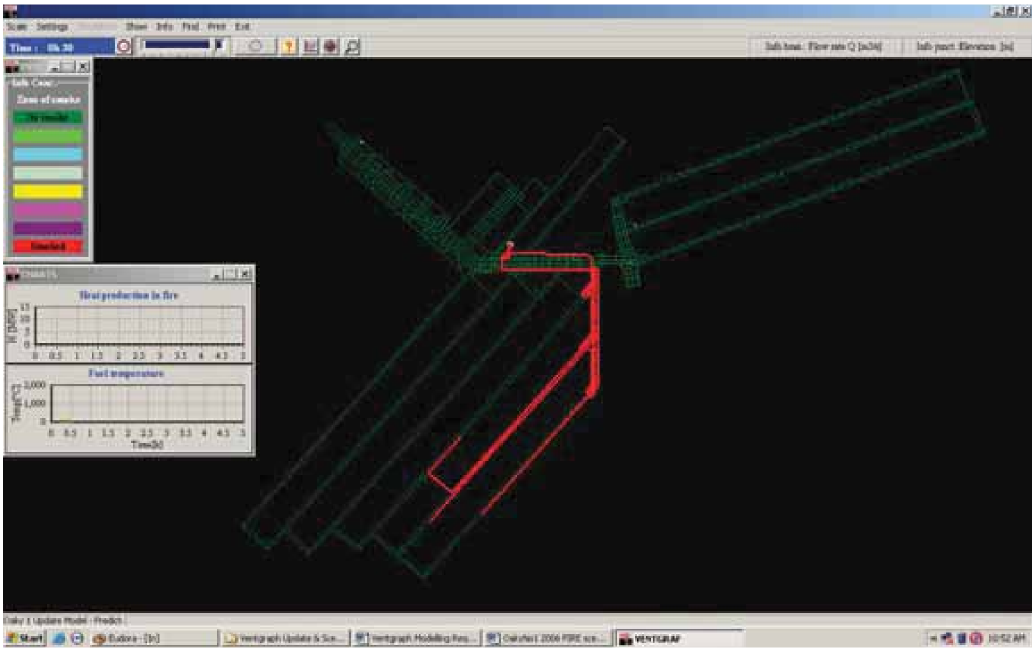


Figure 8.8 Smoke distribution after 30 minutes.

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2$ = 0.1. (assume H_2= CO level); fire very unstable and not under control.

Control Fire fighting ineffective within 120 minutes.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts. Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Control Assess effectiveness of GAG

Examine fan curve operating point

NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 After 360 minutes Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Control Assess effectiveness of GAG

Step 7 After 390 minutes Shut down No 2 fan; fan louvre doors closed R=10.
Close Portal Dip B Heading Emergency Door R=10.

Control Assess effectiveness of GAG

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 After 420 minutes Shut down No 3 fan; fan louvre doors open.
Close Portal Dip C Heading Emergency Door R=1.

Control Assess effectiveness of GAG

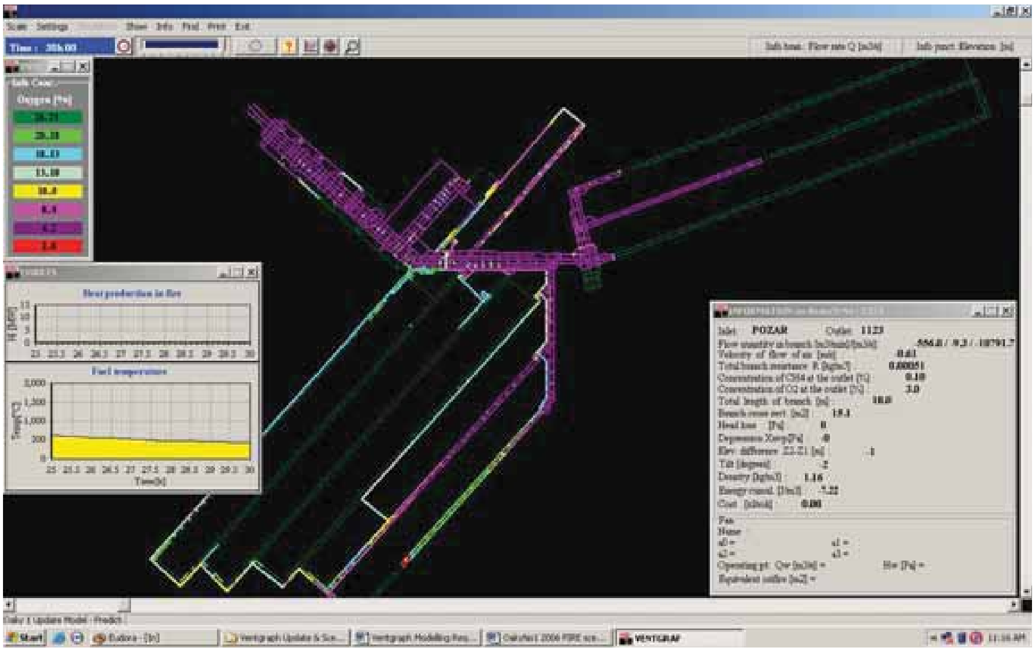


Figure 8.9 Oxygen distribution after 1800 minutes.

Summary *With GAG running Fire intensity insignificant at 30 hours and oxygen level outbye fire at less 3.0 percent.*

8.5. Oaky Creek No 1 Fire Scenario 5

Scenario *In the South Mains D8 to MG24 C2 (belt drive head area), hydraulic oil has caught on fire. LW Drivehead - South Mains D8 to MG24 C2 - Belt Drivehead area which is segregated.*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 27 minutes
Smoke reaches Longwall face at 30 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Control Fire fighting control is suppressing oil fire

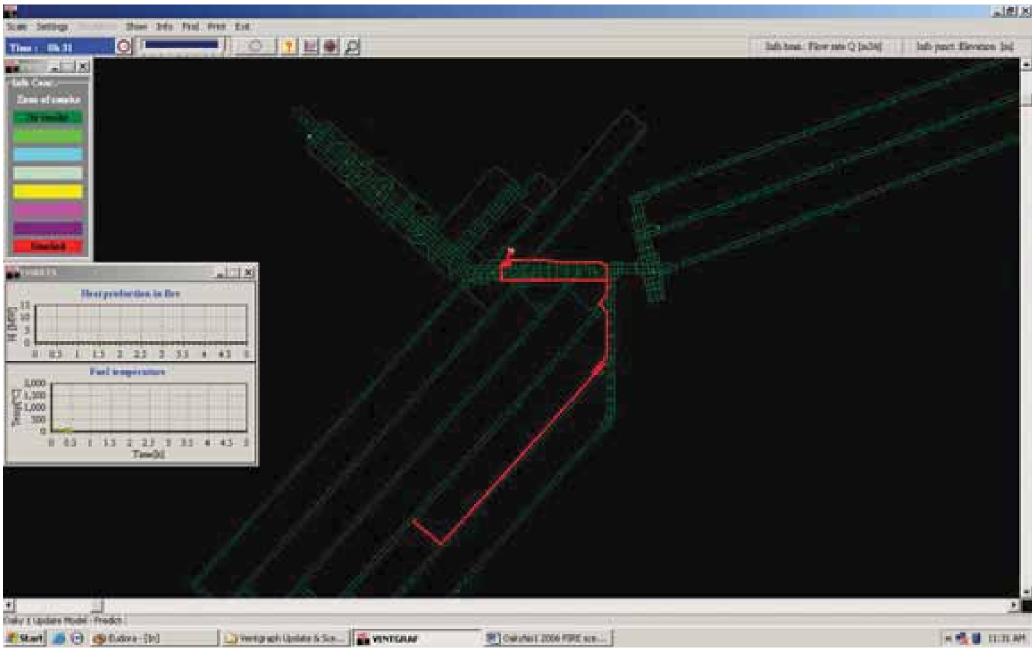


Figure 8.10 Smoke distribution after 30 minutes.

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2$ = 0.1. (assume H_2= CO level); fire very unstable and not under control.

Control Fire fighting ineffective within 120 minutes.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 350 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Control Assess effectiveness of GAG

Examine fan curve operating point

NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 After 390 minutes Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Control Assess effectiveness of GAG

Step 7 After 420 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip B Heading Emergency Door R=10

Control Assess effectiveness of GAG

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 After 450 minutes Shut down No 3 fan; fan louvre doors open.
Close Portal Dip C Heading Emergency Door R=1.

Control Assess effectiveness of GAG

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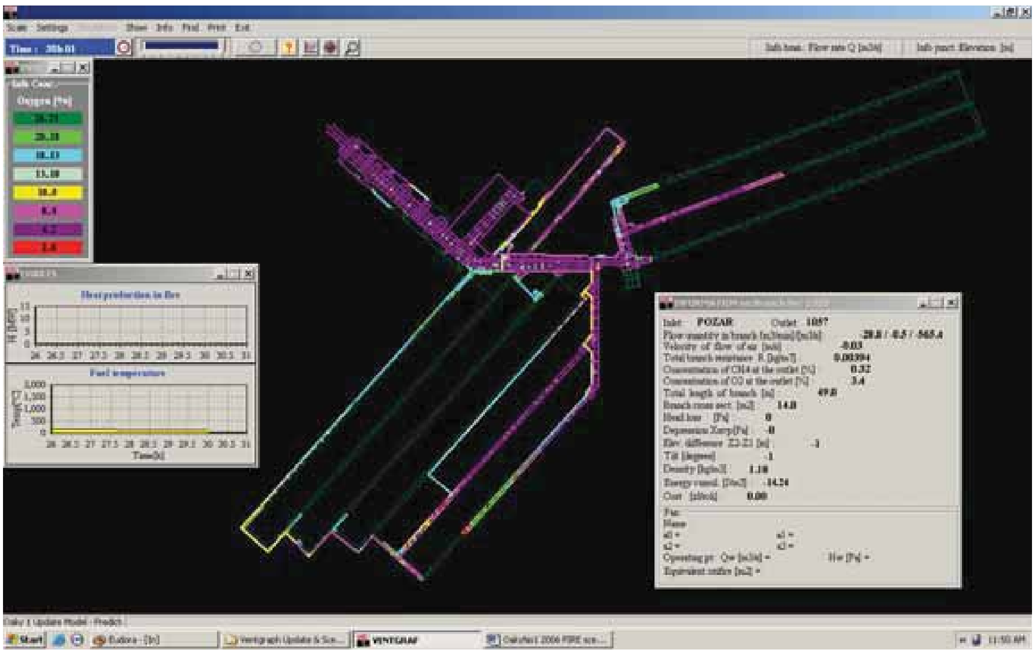
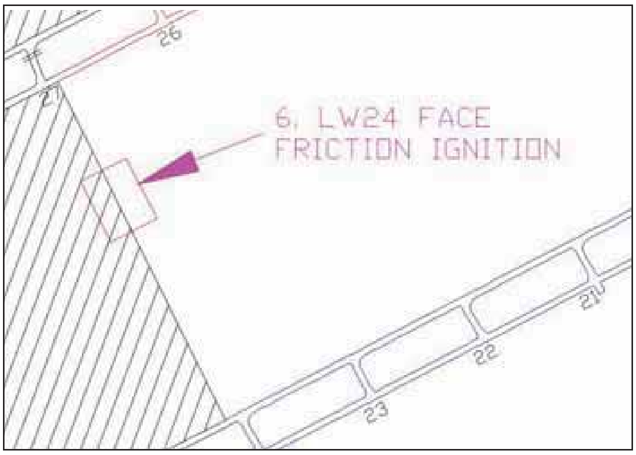


Figure 8.11 Oxygen distribution after 1800 minutes.

Summary *With GAG running Fire intensity insignificant at 36 hours and oxygen level outbye fire at less than 3.2 percent.*

8.6. Oaky Creek No 1 Fire Scenario 6

Scenario *Fire on LW 24 face at mid point caused by friction ignition of methane igniting coal.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.
- O_2 sensor on LW face 100m from MG and O_2 sensor on LW face 200m from MG; O_2 sensors do not occur in the mine.

Simulation

Step 1 Time 0 – 30 minutes: Methane blower burning. Simulate as 30 litres oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Step 2 Time 30 – 60 minutes: 5 m coal length at mid-longwall, time constant 14,400s, intensity 5, $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Smoke reaches surface at 32 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 3 Time 60 – 90 minutes: 20 m entry length coal develops, time constant 14,400s, intensity 5.

Control Fire fighting ineffective within 90 minutes; management decision to change to ventilation control strategies; 30-45 minutes to implement.

Step 4 Time 90 – 240 minutes: 50 m entry length coal develops, time constant 14,400s, intensity 5.

Time 120 minutes: Brattice placed at BSL R=0.2

Control Fire fighting ineffective within 120 minutes; management decision to further change ventilation control strategies; 45 minutes to implement.

Time 165 minutes: Brattice placed Outbye LW equipment, belt dropped R=5

Control Fire fighting ineffective within 165 minutes; management decision to further change ventilation control strategies; 45 minutes to implement.

Time 210 minutes: Brattice placed at first and second CT Outbye LW face R=5

Fire out of control

Control Fire fighting ineffective within 210 minutes; management decision to introduce high flow inertisation – GAG; 120 minutes to implement.

Step 5 Time 240 - 360 minutes: 125 m entry length coal develops, source time constant 14,400s, intensity 5.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 6 Time 360 minutes: Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Control Assess effectiveness of GAG

Examine fan curve operating point. NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 Time 420 minutes: Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Control Assess effectiveness of GAG

Step 8 Time 480 minutes: Shut down No 2 fan; fan louvre doors closed R=10.
Close Portal Dip B Heading Emergency Door R=10.

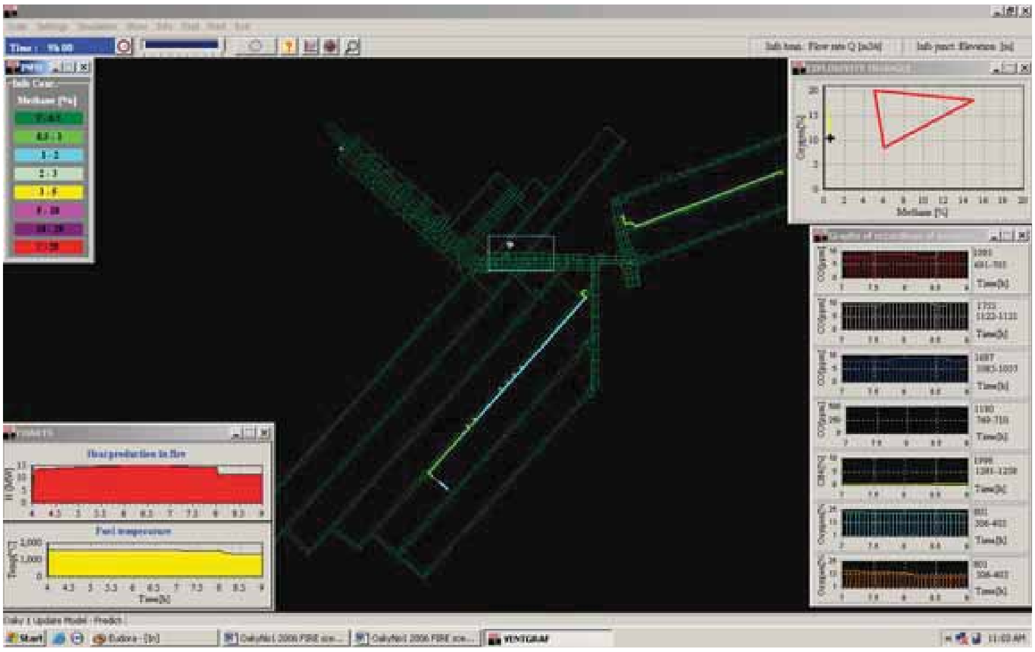


Figure 8.12 Methane distribution after 540 minutes.

Control Assess effectiveness of GAG

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

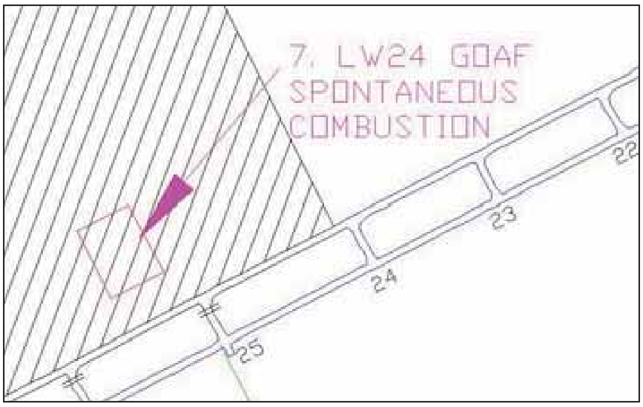
Step 9 Time 540 Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

Reversal occurs bringing methane over the fire source causing an explosion.

***Summary** Reversal occurs bringing methane over the fire source. Explosion occurred*

8.7. Oaky Creek No 1 Fire Scenario 7

Scenario *LW Goaf Spontaneous Combustion - Longwall 24 Goaf heating. Longwall 24 face currently at 26 ct. Fire located at 27 ct on MG side 40m into goaf.*



Prior to running simulation pr-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW12A at 80 l/s, MG25 at 55 l/s and MG26 at 80 l/s.
  LW24 seven sources total 230 l/s, 110 l/s on face, four sources of 30 l/s each spaced 20m in from MG Hdg.
- *O_2 sensor on LW face 100m from MG and O_2 sensor on LW face 200m from MG; O_2 sensors currently do not occur in the mine at these points.*

Simulation

Step 1 Time 0 – 360 minutes: 1 m entry length coal fuel in 18 c/t MG edge of goaf burning; time constant 14400s, intensity 1 $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control.

Smoke reaches surface at 39 mins.
CO at TG exceeds 5ppm at 70 mins.

Step 2 Time 360– 720 minutes: 5 m entry length coal burning with gas continuing to burn; time constant 14400s, intensity 2.

Step 3 Time 720 – 1080 minutes: Continue coal fire 25 m entry length coal burning; time constant 14400s, intensity 4.

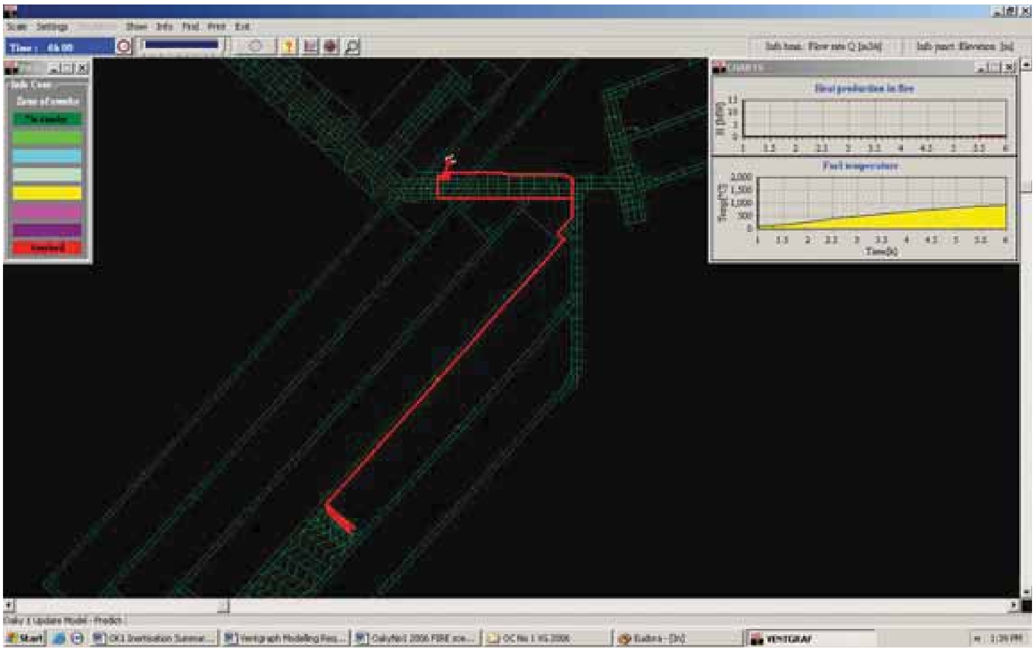


Figure 8.13 Smoke distribution after 360 minutes.

Step 4 Time 1080 - 1440 minutes: Continue coal fire 100 m entry length coal burning; time constant 14400s, intensity 8. Fire very unstable and not under control

CO concentration at 19 hours sets off alarm at bottom of vent shaft.

Step 5 Time 1440 - 1800 minutes: Continue coal fire 200 m entry length coal burning; time constant 14400s, intensity 10.

Step 6 Time 1450 minutes: Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Examine all three main fan curve operating points; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 1500 minutes: Shut down No 1 fan; fan louvre doors closed R=10
Examine No 2 and No3 fan curve operating points

Step 8 1560 minutes: Close Portal Dip A Heading Emergency Door R=10.

Step 9 1590 minutes: Shut down No 2 fan; fan louvre doors closed R=10

Step 10 1620 minutes: Close Portal Dip B Heading Emergency Door R=10

Step11 1680 minutes: Shut down No 3 fan; fan louvre doors open

Close Portal Dip C Heading Emergency Door R=1; at about 28 hours local reversal occurred to produce a minor methane burn off.

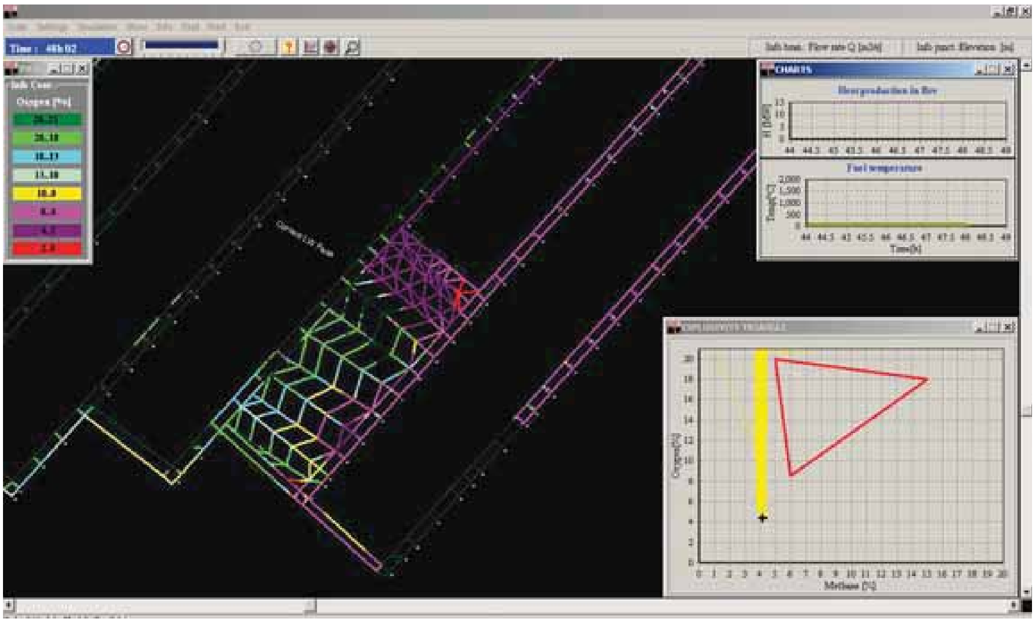


Figure 8.14 Oxygen distribution after 2880 minutes.

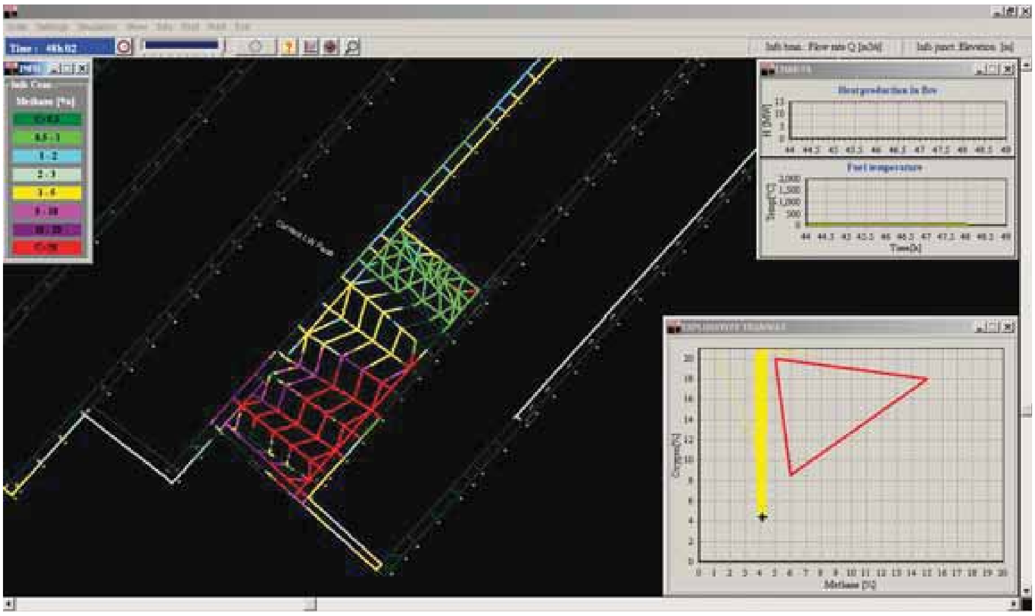


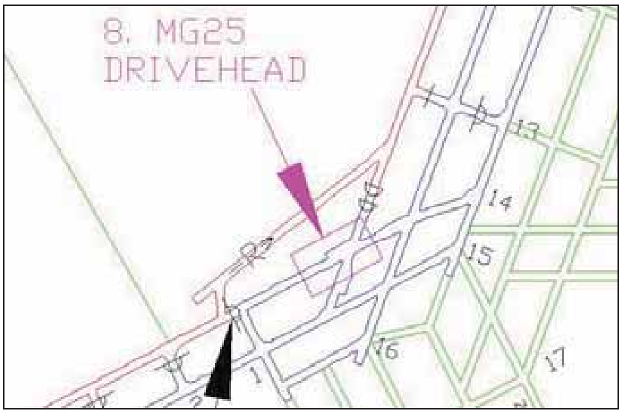
Figure 8.15 Methane distribution after 2880 minutes.

Fire substantially reduced without GAG exhaust reaching fire but GAG ensures full extinguishment.

Summary *With GAG running fire intensity insignificant at 48 hours and oxygen level outbye fire at less than 2.5 percent.*

8.8. Oaky Creek No 1 Fire Scenario 8

Scenario *In the South Mains D14 to MG25 C2 (belt drive head area), hydraulic oil has caught on fire. MG25 Drivehead - South Mains D14 to MG25 C2 - Belt Drivehead area which is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

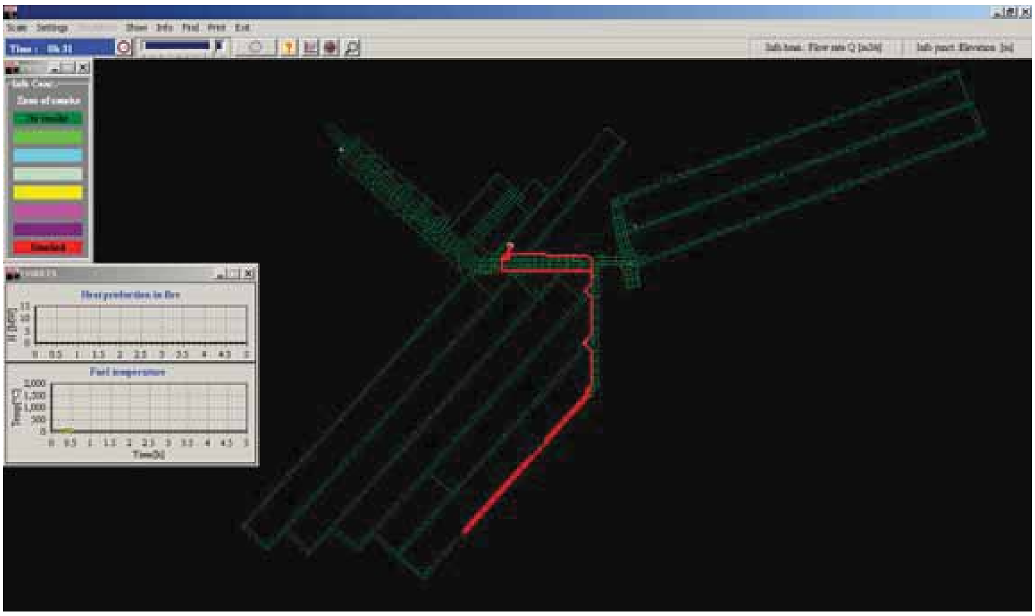


Figure 8.16 Smoke distribution after 30 minutes.

Smoke reaches surface at 27 minutes; Smoke reaches Longwall face at 25 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Control Fire fighting control is suppressing oil fire

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2$ = 0.1. (assume H_2= CO level); fire very unstable and not under control.

Control Fire fighting ineffective within 120 minutes.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts. Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: Continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 After 360 minutes Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Step 7 After 390 minutes Shut down No 2 fan; fan louvre doors closed R=10.
Close Portal Dip B Heading Emergency Door R=10.

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation.

Step 8 After 450 minutes shut down No 3 fan; fan louvre doors open.
Close Portal Dip C Heading Emergency Door R=1.

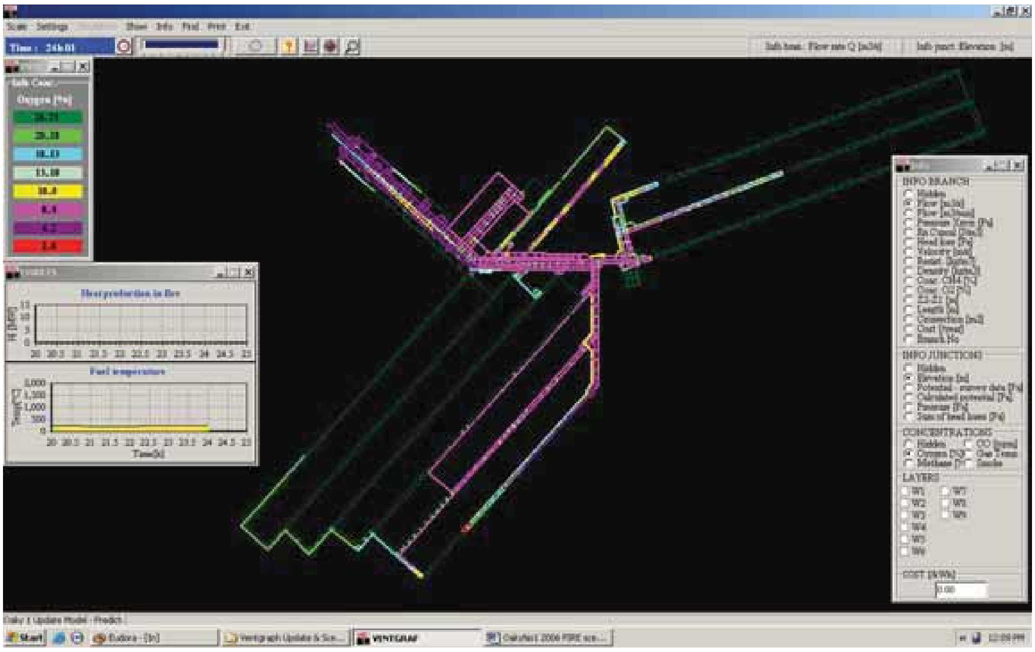


Figure 8.17 Oxygen distribution after 1440 minutes.

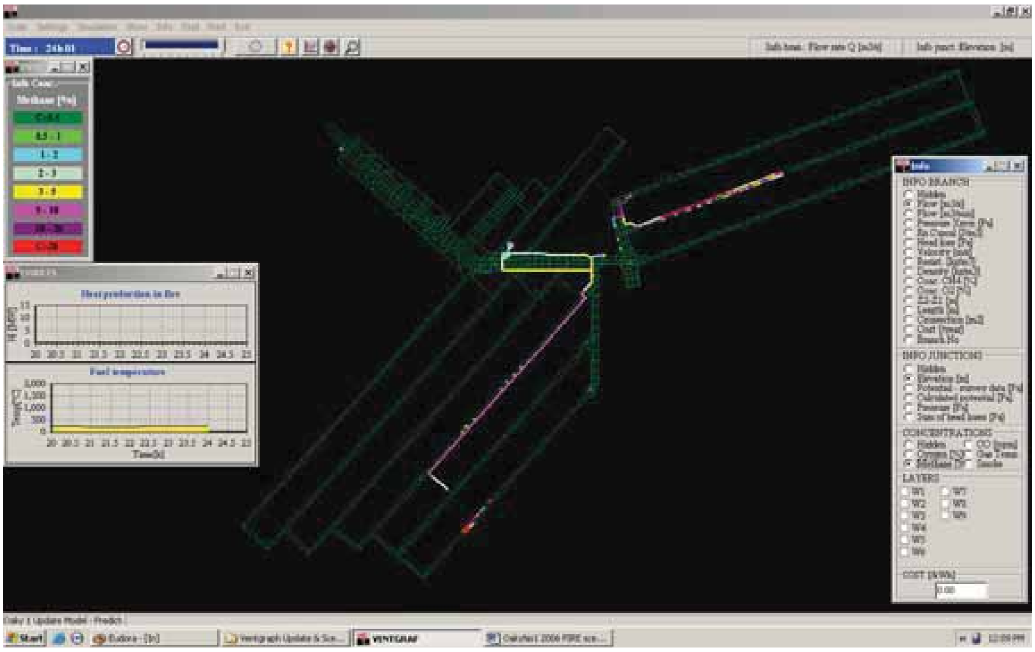
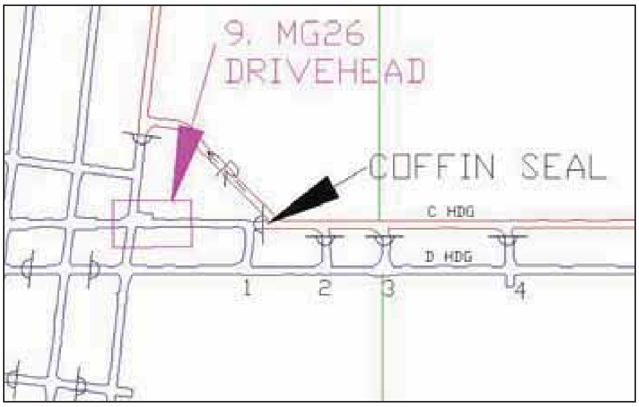


Figure 8.18 Methane distribution after 1440 minutes.

Summary *With GAG running Fire intensity insignificant at 24 hours and oxygen level outbye fire at less 2.9 percent.*

8.9. Oaky Creek No 1 Fire Scenario 9

Scenario *In the Sandy Creek East Mains D6 to MG26 C1 (belt drive head area), hydraulic oil has caught on fire. MG26 Drivehead - Sandy Creek East Mains D6 to MG26 C1- Belt Drivehead area which is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 25 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2= CO level); fire very unstable and not under control.

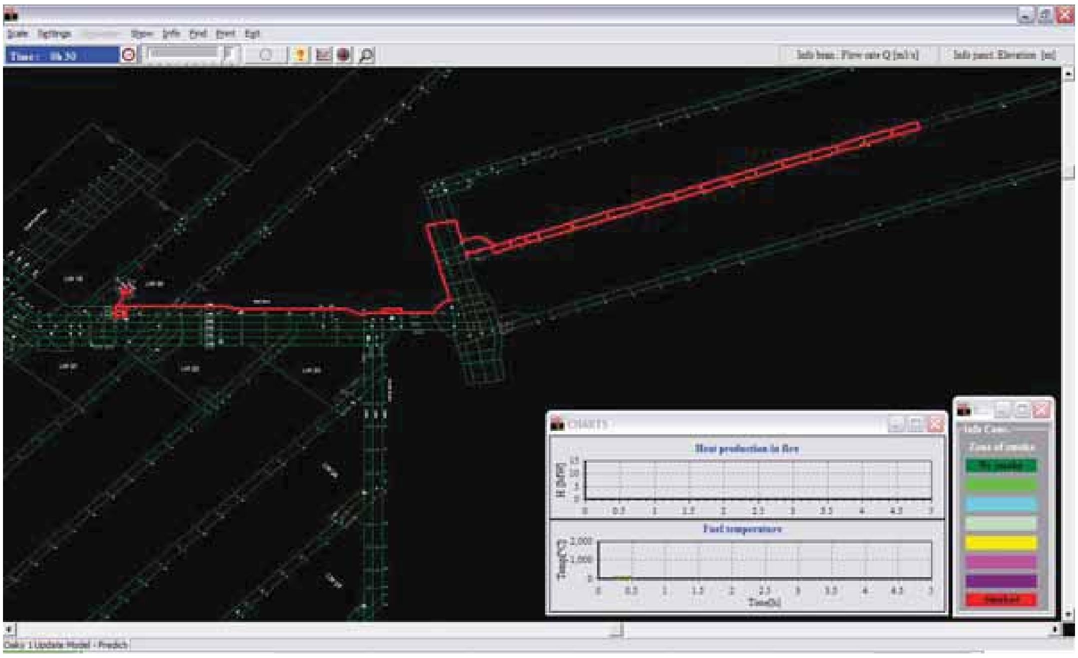


Figure 8.19 Smoke distribution after 30 minutes.

Control Fire fighting ineffective within 120 minutes.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

At 300 minutes GAG has been set up at C Heading Portal Dips and C Heading Emergency Doors closed, R=10

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.
Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 After 330 minutes Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Step 8 After 360 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip B Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 9 After 390 minutes Shut down No 3 fan; fan louvre doors open.
Close Portal Dip C Heading Emergency Door R=1.

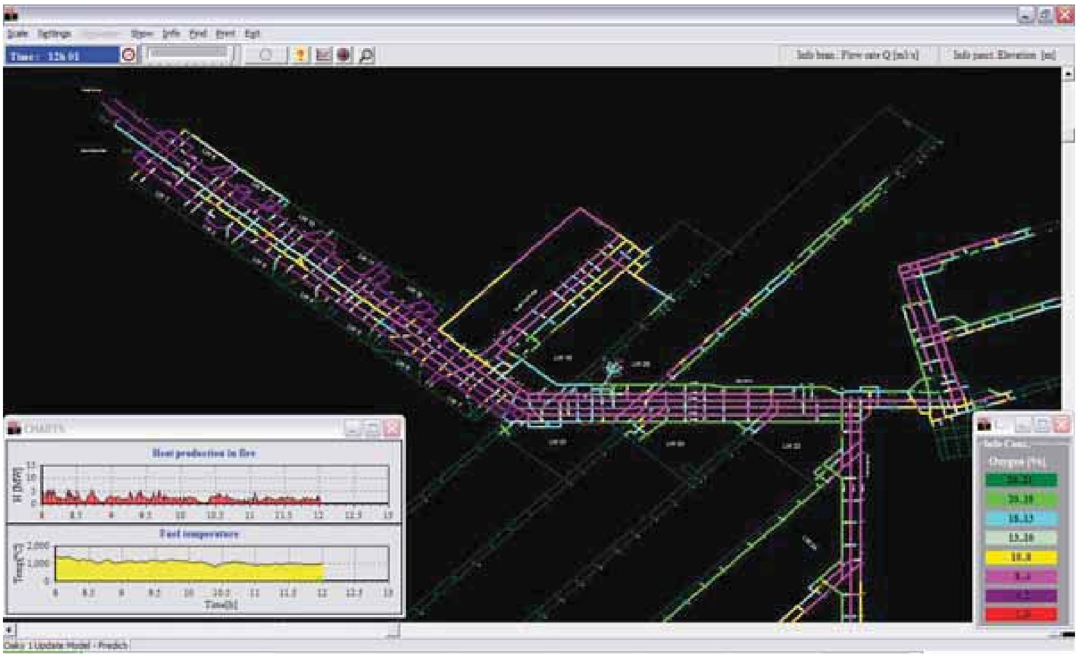
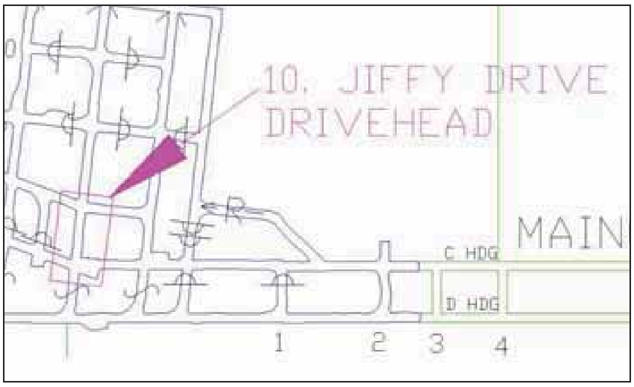


Figure 8.20 Oxygen distribution after 720 minutes.

Summary: ***F****ire fluctuating and not reducing in intensity after 720 minutes with GAG in operation.*

8.10. Oaky Creek No 1 Fire Scenario 10

Scenario *In the Sandy Creek East Mains C13 to C12 (belt drive head area), hydraulic oil has caught on fire. Jiffy Drive 1 Drivehead - Sandy Creek East Mains C13 to C12-Belt Drivehead area which is segregated.*



Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Downcast Shaft entry connected to 3 CT D Heading on Main dips
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 30 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2= CO level); fire very unstable and not under control.

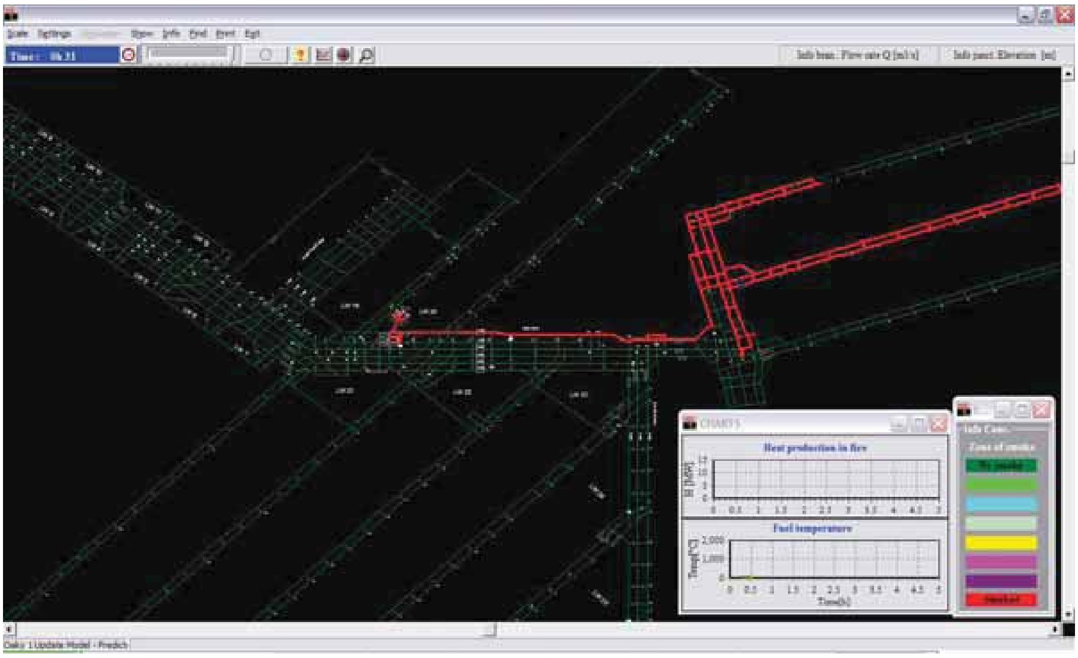


Figure 8.21 Smoke distribution after 30 minutes.

Control Fire fighting ineffective within 120 minutes

Step 4 Time 120 – 300 minutes: Coal is fuel source as all liquid fuel has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2 = 0.1$ (assume $H_2 = CO$ level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

At 300 minutes GAG has been set up at C Heading Portal Dips and C Heading Emergency Doors closed, R=10

Commence GAG control action; GAG has been set up at Downcast Shaft entry connected to 3 CT D Heading on Main dips. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Control Assess effectiveness of GAG

Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 After 330 minutes Shut down No 1 fan; fan louvre doors closed R=10.
Close Portal Dip A Heading Emergency Door R=10.

Step 7 After 360 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip B Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 After 390 minutes Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

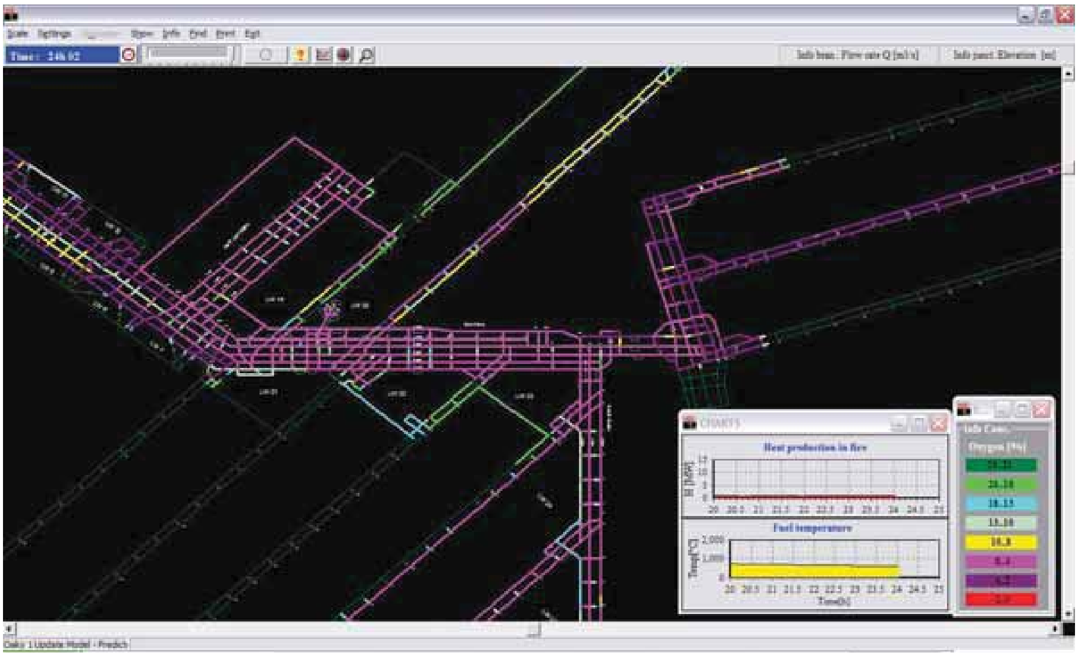


Figure 8.22 Oxygen distribution after 1440 minutes.

Summary *With GAG running, fire intensity insignificant at 24 hours and oxygen level outbye fire at less than 2.9 percent.*

9. REVIEW OF OPTIONS FOR IMPROVING ABILITY TO INERTISE PRIORITY FIRES AT OAKY NO 1 MINE

As mentioned in Section 3.3, location position of the inertisation unit's point of coupling to the mine, or the docking point, is a major determinant of potential success for most efficient suppression of a specific underground fire. Traditionally in Queensland docking points have been placed on intake ventilation access openings (such as travel or conveyor belt roads or adjacent vertical shafts). Some mines have prepared docking points on boreholes of about 1.0 to 2.0m diameter placed at the back of longwall panels.

Scenarios developed for Oaky Creek No 1 Colliery have been examined as to the ability of a GAG inertisation unit placed at the current docking point on the intake shaft adjacent to the Highwall Declines to inert a fire in the mine recovery stage following a fire. Table 9.1 shows results of the outcome of the ten scenarios investigated in Chapter 8.

The ten scenario outcomes have been categorised as follows.

- Category A covers fire in which the inertisation product is directed fully over the fire without significant dilution of the GAG exhaust. None of the ten mine priority fires examined achieved the situation in which the simulated fire is directly stabilised to aid recovery in a timely manner.
- Category B covers situations in which the inertisation product goes straight to the fire but there is significant dilution from other ventilation air or leakage through stoppings. Because of dilution stabilisation of a fire through inertisation can only be achieved with some main surface fan changes. Four Oaky Creek No 1 scenarios are in this category. Under these situations the fire should, over time, be abated or stabilised to a point where conventional recovery approaches can be initiated.
- Category C covers priority fires in which the GAG output will never reach the fire location without stopping of one or more main surface fans to rebalance ventilation within the pit. In many of these cases requiring fan changes to put GAG output across the fire location effective ventilation air velocity has been reduced to the extent that local reversal across the fire occurs and fire fumes are pulled across the fire. This is an unsatisfactory situation as fire smoke and fumes can carry combustible products. This situation broadly prevails for five scenarios of the cases examined.
- Category D covers priority fires in which the GAG output will never reach the fire location even if surface main fans are altered. These are fire locations within panel sections in which either the fire behaviour stops normal intake ventilation flow into the section headings or the GAG docking point is in an airway that is isolated from the section. There is no such case in the ten scenarios examined.
- Category E covers priority fires in gassy mines in which section production gas make has been included in the simulation modelling. GAG exhaust will never reach the fire location

without stopping of one or more main surface fans to rebalance ventilation within the pit. However this change in ventilation causes working section methane and ventilation air (incl. fire fumes) to reverse across the fire. This is clearly a potentially dangerous situation. This situation was found in one scenario of the cases examined.

Alternative approaches to improve the efficiency of GAG inertisation in the event of a major fire can be considered from the following.

1. Maintain use of existing docking station but with additional underground segregation to control the delivery of inert.
2. Try new Portal docking station.
3. Try new Portal docking station possibly with additional underground segregation.
4. Drill new borehole to deliver inert gas directly to the fire site.
5. MG regulator should be opened further to dump belt coffin seal air to return.
6. Coffin seal regulator should be opened further to dump all belt air to return.

The following sections describe how some of the scenarios where improvement was considered possible have been re-simulated based on consideration of these alternative actions as described in this section of the report.

Scenarios which had been assessed at Category B were not re-examined. This is the situation with Scenarios 4, 5, 8 and 10. It was considered that Category B was the best rating they could achieve.

Table 9.1 Summary of Original Scenario Outcomes on the Effects of Inertisation using current GAG Portal.

No	Fire Location	Fire Type	GAG Position	Segregation Actions	Fan Actions	Outcomes	GAG Inertisation	Comments
1	SE02 Drivehead Main Dips C9-C10	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category C***	Exhaust cannot reach fire until all fans off and air reversal occurs. Fire insignificant after 20 hrs.	No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off.
2	SE03 Drivehead Main Dips C21-C22	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category C***	Exhaust cannot reach fire until all fans off and air reversal occurs. Fire insignificant after 24 hrs.	No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off.
3	SE04 Drivehead East Mains C5-C6	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category C***	Exhaust reaches fire but with dilution. Fire insignificant after 17 hrs.	No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off.
4	SE05 Drivehead East Mains C23-D23	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category B***	Exhaust reaches fire but with dilution. With all fans off air reversal occurs. Fire insignificant after 30 hrs.	Methane had potential to reversal across the fire. Ventilation air reversal occurred over the fire after all fans turned off.
5	LW Drivehead South Mains D8 to	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B	Fans turned off one by	***Category B***	Exhaust reaches fire but with dilution. Fire	Air unstable after #2 fan shut down but reversal not

	MG24 C			after #2 Fan off; Close Drift C after #3 Fan off	one		insignificant after 36 hrs.	evident.
6	LW Friction Ignition Longwall 24 face	Gas → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category E***	Exhaust reaches fire but with dilution.	Ventilation air reversal occurred over the fire after all fans turned off. Methane had reversal across the fire and explosion occurred.
7	LW Goaf Spon Comb LW 24 Goaf heating	Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off		***Category C***	Exhaust reaches fire but with dilution. Fire insignificant with all fans off after 48 hrs.	Ventilation air reversal occurred over the fire after all fans turned off. Methane had reversal across the fire.
8	MG25 Drivehead South Mains D14 - MG25 C	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category B***	Exhaust reaches fire but with dilution. Fire insignificant after 24 hrs.	Fire unstable throughout. No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off.
9	MG26 Drivehead Sandy Creek East Mains D6 to MG26	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category C***	Exhaust reaches fire but with dilution. Fire fluctuating and not reducing in intensity after 12 hrs.	Fire unstable throughout. No methane reversal across the fire.
10	Jiffy Drive 1 Drivehead Sandy Creek East Mains C13 to C1	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	***Category B***	Exhaust reaches fire but with dilution. Fire insignificant after 24 hrs.	No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off.

9.1. Oaky Creek No 1 Fire Scenario 1A

Scenario *In the "C" main dips at bottom of main dips at the C9-C10 (belt drive head area), hydraulic oil has caught on fire. SE02 Drivehead - Main Dips C9-C10 - Belt Drivehead area which is segregated.*

***Changed Inertisation Strategy*:** *GAG docked on Portal Dip C heading.*

Prior to running simulation pr-enter some of the controls that may be required e.g.

- Initiation of GAG at Portal Dip C Heading.
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW12A 80 l/s, LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 22 minutes

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Smoke reaches Longwall face at 50 minutes

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$. (assume H_2= CO level); fire very unstable and not under control.

Control Fire fighting ineffective within 120 minutes

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2 = 0.1$ (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at Portal Dip C Heading. Emergency Door at Portal Dip C Heading closed, R=10; Set GAG to 11,000rpm, efficiency 10%.
Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 6 At 330 minutes: Shut down No 1 fan; fan louvre doors closed R=10.

Step 7 At 360 minutes: Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip A Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 8 At 390 minutes: Shut down No 3 fan; fan louvre doors open
Close Portal Dip B Heading Emergency Door R=10

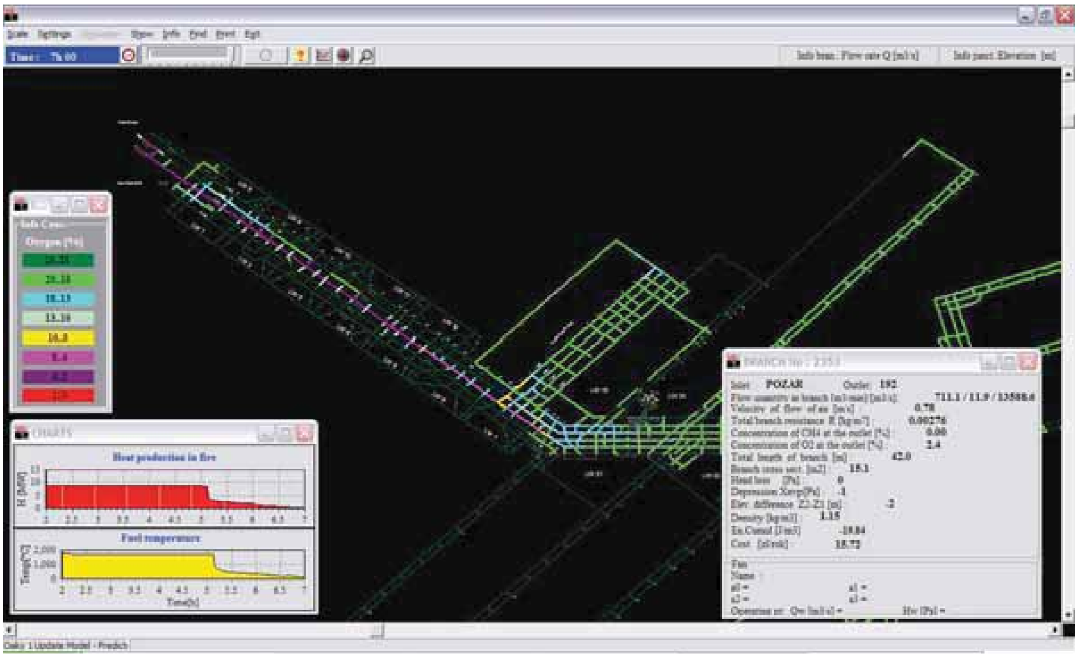


Figure 9.1 Oxygen distribution after 420 minutes.

Control Assess effectiveness of GAG

Summary *With GAG running, fire intensity insignificant at 7 hours and oxygen level outbye fire at less than 2.5 percent. No ventilation air reversal occurred across the fire.*

9.2. Oaky Creek No 1 Fire Scenario 2A

Scenario *In the "C" main dips at bottom of main dips at the C21-C22 (belt drive head area), hydraulic oil has caught on fire. SE03 Drivehead - Main Dips C21-C22 - Belt Drivehead area which is segregated.*

Changed Inertisation Strategy*: *GAG docked on Portal Dip C heading.

Prior to running simulation pr-enter some of the controls that may be required e.g.

- Initiation of GAG at Portal Dip C Heading.
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW12A 80 l/s, LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 16 minutes

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Smoke reaches MG26 face at 45 minutes; Smoke reaches Longwall face at 47 minutes; Smoke reaches MG25 face at 54 minutes

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$.

East Mains CO concentration is less than 50ppm throughout.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2 = 0.1$ (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.
Commence GAG control action; GAG has been set up at Portal Dip C Heading.
Emergency Door at Portal Dip C Heading closed, R=10; Set GAG to 11,000rpm, efficiency 10%.
Examine fan curve operating point; NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 At 330 minutes: Shut down No 1 fan; fan louvre doors closed R=10.

Step 8 At 360 minutes: Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip A Heading Emergency Door R=10
Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 9 Shut down No 3 fan; fan louvre doors open
Close Portal Dip B Heading Emergency Door R=10

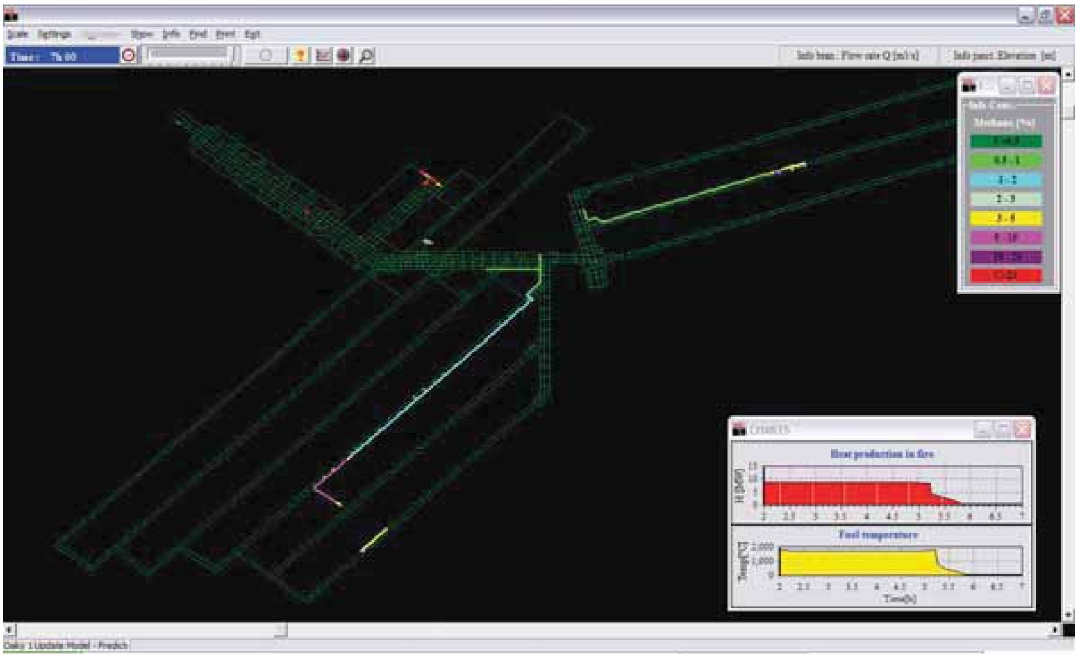


Figure 9.2 Methane distribution after 420 minutes.

Summary *With GAG running Fire intensity insignificant at 7 hours and oxygen level outbye fire at less than 2.6 percent. No ventilation air reversal occurred across the fire.*

9.3. Oaky Creek No 1 Fire Scenario 3A

Scenario *In the "C" main dips at bottom of main dips at the C5-C6 (belt drive head area), hydraulic oil has caught on fire. SE04 Drivehead - Main Dips C5-C6 - Belt Drivehead area which is segregated.*

Changed Inertisation Strategy**:** *GAG docked on Portal Dip C heading.*

Prior to running simulation pr-enter some of the controls that may be required e.g.

- Initiation of GAG at Portal Dip C Heading.
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2 = 0.1$. (assume H_2 = CO level).

Smoke reaches surface at 5 minutes

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.
Smoke reaches Longwall face at 37 minutes; Smoke reaches MG26 face at 37 minutes
Smoke reaches MG25 face at 45 minutes

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2 = 0.1$.

East Main CO concentration is less than 50ppm throughout, except C heading

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2 = 0.1$ (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning.

Commence GAG control action; GAG has been set up at portal C entry. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.
Examine fan curve operating point NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 After 330 minutes Shut down No 1 fan; fan louvre doors closed R=10.

Step 8 After 360 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip A Heading Emergency Door R=10

Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 9 After 390 Shut down No 3 fan; fan louvre doors open
Close Portal Dip B Heading Emergency Door R=1

Localised reversal occurs

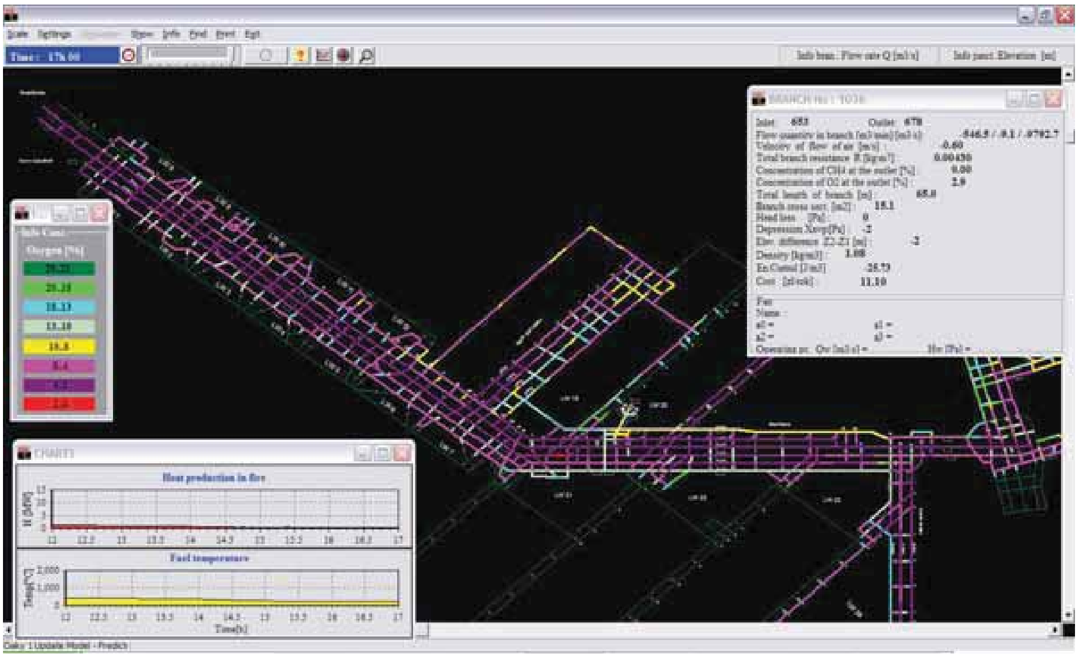


Figure 9.3 Oxygen distribution after 1020 minutes.

Summary *With GAG running Fire intensity insignificant at 17 hours and oxygen level outbye fire at less than 2.9 percent.*

9.4. Oaky Creek No 1 Fire Scenario 6A

Scenario *Fire on LW 24 face at mid point caused by friction ignition of methane igniting coal.*

Changed Inertisation Strategy**:** *GAG docked on Portal Dip B heading.*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Portal Dip B Heading.
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.
- O_2 sensor on LW face 100m from MG and O_2 sensor on LW face 200m from MG; O_2 sensors do not occur in the mine.

Simulation

Step 1 Time 0 – 30 minutes: Methane blower burning. Simulate as 30 litres oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Step 2 Time 30 – 60 minutes: 5 m coal length at mid-longwall, time constant 14,400s, intensity 5, $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Smoke reaches surface at 32 minutes

Control Fire fighting control commences with water jet, fog and low expansion foam suppressing oil fire.

Step 3 Time 60 – 90 minutes: 20 m entry length coal develops, time constant 14,400s, intensity 5.

Control Fire fighting ineffective within 90 minutes; management decision to change to ventilation control strategies; 30-45 minutes to implement.

Step 4 Time 90 – 240 minutes: 50 m entry length coal develops, time constant 14,400s, intensity 5.

Time 120 minutes: Brattice placed at BSL R=0.2

Control Fire fighting ineffective within 120 minutes; management decision to further change ventilation control strategies; 45 minutes to implement.

Time 165 minutes: Brattice placed Outbye LW equipment, belt dropped R=5

Control Fire fighting ineffective within 165 minutes; management decision to further change ventilation control strategies; 45 minutes to implement.

Time 210 minutes: Brattice placed at first and second CT Outbye LW face R=5

Fire out of control

Control Fire fighting ineffective within 210 minutes; management decision to introduce high flow inertisation – GAG; 120 minutes to implement.

Step 5 Time 240 - 360 minutes: 125 m entry length coal develops, source time constant 14,400s, intensity 5.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 6 Time 360 minutes: Commence GAG control action; GAG has been set up at B Heading Portal Dips and B Heading Emergency Doors closed, R=10

Commence GAG control action; Set GAG to 11,000rpm, efficiency 10%.
Examine fan curve operating point. NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 Time 420 minutes: Shut down No 1 fan; fan louvre doors closed R=10
Close Portal Dip A Heading Emergency Door R=10

Step 8 Time 480 minutes: Shut down No 2 fan; fan louvre doors closed R=10

Concern that too much restriction of air to mine will put face into Coward Triangle. Check LW face methane situation

Step 9 Time 540 minutes: Turn down No 3 fan to P_{fan} = 0.25; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

Step 10 Time 630 minutes: Shut down No 3 fan; fan louvre doors open

Reversal occurs bringing methane over the fire source however, oxygen level is very low at 3.3% outbye the fire.

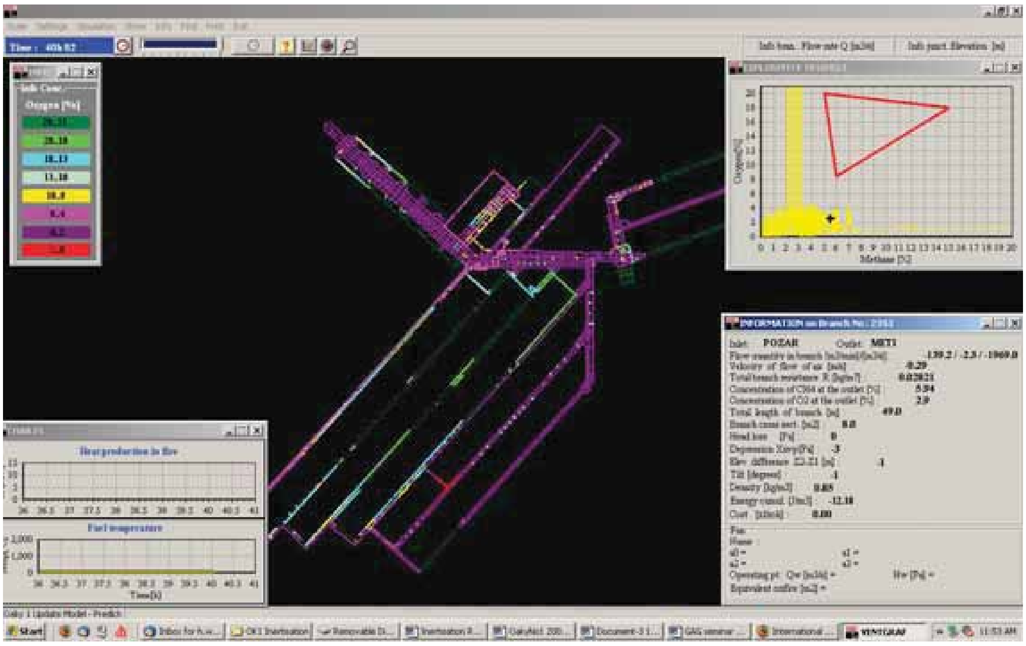


Figure 9.4 Oxygen distribution after 2400 minutes.

Summary *Fire insignificant after 11.5 hrs. New GAG docking position prevents methane explosion. However, reversal of methane at LW face still occurred immediately after the No 3 fan is turned off. Methane laden air had reversed across the fire but with very low levels of oxygen no explosion occurred.*

9.5. Oaky Creek No 1 Fire Scenario 7A

Scenario *LW Goaf Spontaneous Combustion - Longwall 24 Goaf heating. Longwall 24 face currently at 26 ct. Fire located at 27 ct on MG side 40m into goaf.*

Changed Inertisation Strategy**:** *GAG docked on Portal Dip B heading.*

Prior to running simulation pre-enter some of the controls that may be required e.g.

- Initiation of GAG at Portal B Heading.
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW12A at 80 l/s, MG25 at 55 l/s and MG26 at 80 l/s.
  LW24 seven sources total 230 l/s, 110 l/s on face, four sources of 30 l/s each spaced 20m in from MG Hdg.
- *O_2 sensor on LW face 100m from MG and O_2 sensor on LW face 200m from MG; O_2 sensors do not occur in the mine.*

Simulation

Step 1 Time 0 – 360 minutes: 1 m entry length coal fuel in 18 c/t MG edge of goaf burning; time constant 14400s, intensity 1 $CO:CO_2 = 0.1$. (assume H_2 = CO level); fire very unstable and not under control.

Smoke reaches surface at 39 mins.
CO at TG exceeds 5ppm at 70 mins.

Step 2 Time 360– 720 minutes: 5 m entry length coal burning with gas continuing to burn; time constant 14400s, intensity 2.

Step 3 Time 720 – 1080 minutes: Continue coal fire 25 m entry length coal burning; time constant 14400s, intensity 4.

Step 4 Time 1080 - 1440 minutes: Continue coal fire 100 m entry length coal burning; time constant 14400s, intensity 8. Fire very unstable and not under control

CO concentration at 19 hours sets off alarm at bottom of vent shaft.

Step 5 Time 1440 - 1800 minutes: Continue coal fire 200 m entry length coal burning; time constant 14400s, intensity 10.

Step 6 Time 1450 minutes: Commence GAG control action; GAG has been set up at Portal B entry. Emergency Door closed, R=10; Set GAG to 11,000rpm, efficiency 10%.

Examine all three main fan curve operating points. NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 1510 minutes: Shut down No 1 fan; fan louvre doors closed R=10
Examine No 2 and No3 fan curve operating points

Step 8 1560 minutes: Close Portal Dip A Heading Emergency Door R=10.

Step 9 1620 minutes: Shut down No 2 fan; fan louvre doors closed R=10

*Step 10*1680 minutes: Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

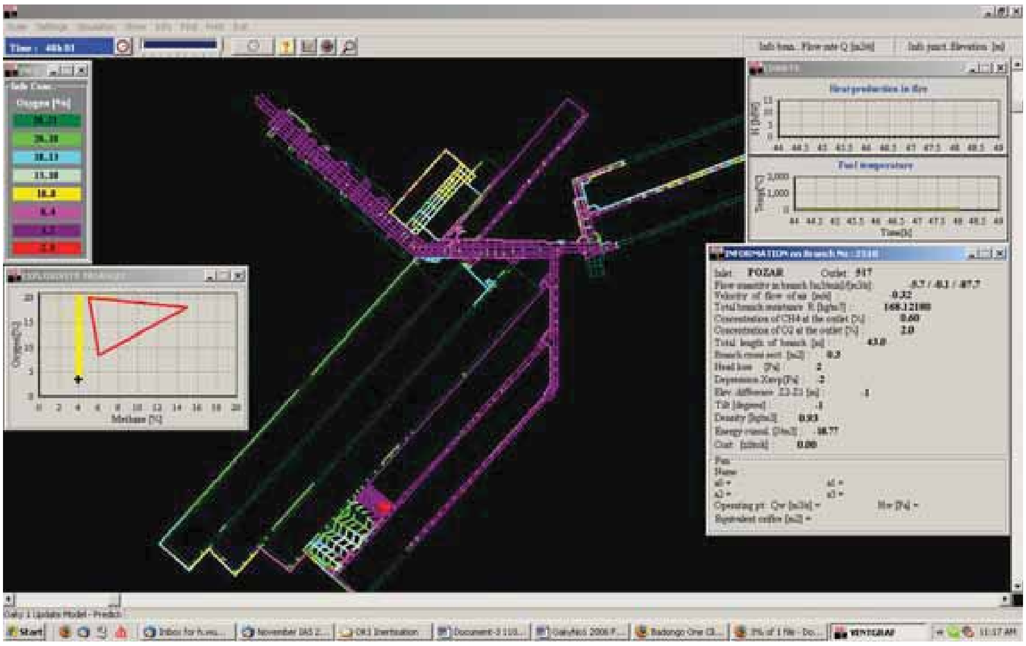


Figure 9.5 Oxygen distribution after 2880 minutes.

Fire substantially reduced without GAG exhaust reaching fire but GAG ensures full extinguishment.

Summary *With GAG running fire intensity insignificant at 48 hours and oxygen level outbye fire at about 0.6 percent. No methane reversal over the fire with new GAG docking position.*

9.6. Oaky Creek No 1 Fire Scenario 9A

Scenario *In the Sandy Creek East Mains D6 to MG26 C1 (belt drive head area), hydraulic oil has caught on fire. MG26 Drivehead - Sandy Creek East Mains D6 to MG26 C1- Belt Drivehead area which is segregated.*

Inertisation Strategy**:** *GAG on Portal Dip B heading.*

Prior to running simulation pr-enter some of the controls that may be required e.g.

- Initiation of GAG at portal B Heading.
- CO gas monitors set at Mains C 9-10ct, C 22-23ct, East Mains C 5a-6ct, South Mains 1-2ct, MG24 Belt 0-1ct, MG24 Belt 16-17ct and MG26 DL. They occur here in the mine.
- CH_4 gas monitor set at MG26 DL.
- Face methane outputs: LW24 at 230 l/s, MG25 at 55 l/s and MG26 at 80 l/s.

Simulation

Step 1 Time 0 – 30 minutes: 30 litres hydraulic oil burning. Simulate 1m length fire over entry width; time constant 120s, intensity 10, $CO:CO_2$ = 0.1. (assume H_2 = CO level).

Smoke reaches surface at 25 minutes

Step 2 Time 30 – 60 minutes: 230 litres cooling oil burning from heat exchanger radiator. Simulate 7m length fire over entry width; time constant 120s, intensity 10.

Smoke reaches Longwall face at 50 minutes

Control Fire fighting control is suppressing oil fire

Step 3 Time 60 – 120 minutes: 230 litres fuel is still burning and 20m length of coal pillar equivalent of 20m additional burning; Simulate 27m length fire over entry width; time constant 120s, intensity 7, $CO:CO_2$ = 0.1. (assume H_2= CO level); fire very unstable and not under control.

Step 4 Time 120 – 300 minutes: all liquid fuel as fire source has been fully consumed. Simulate 50m length coal pillar fire over entry width; time constant 1200s, intensity 6. $CO:CO_2$ = 0.1 (assume H_2 = CO level). Fire very unstable and not under control despite fire fighting attempts.

Fire out of control, withdraw all personnel from mine.

Control Decision made to introduce high flow inertisation – GAG

Step 5 Time 300 – 330 minutes: continue 50 m entry length coal burning. At 300 minutes GAG has been set up at B Heading Portal Dips and B Heading Emergency Doors closed, R=10
Commence GAG control action; Set GAG to 11,000rpm, efficiency 10%.
Examine fan curve operating point. NB Check approach to stall point (Do not allow to stall as program exceeds limitations)

Step 7 After 330 minutes Shut down No 1 fan; fan louvre doors closed R=10.

Step 8 After 360 minutes Shut down No 2 fan; fan louvre doors closed R=10
Close Portal Dip A Heading Emergency Door R=10
Concern that too much restriction of air to mine will put face methane into Coward Triangle. Check LW face methane situation

Step 9 After 390 minutes Shut down No 3 fan; fan louvre doors open
Close Portal Dip C Heading Emergency Door R=1

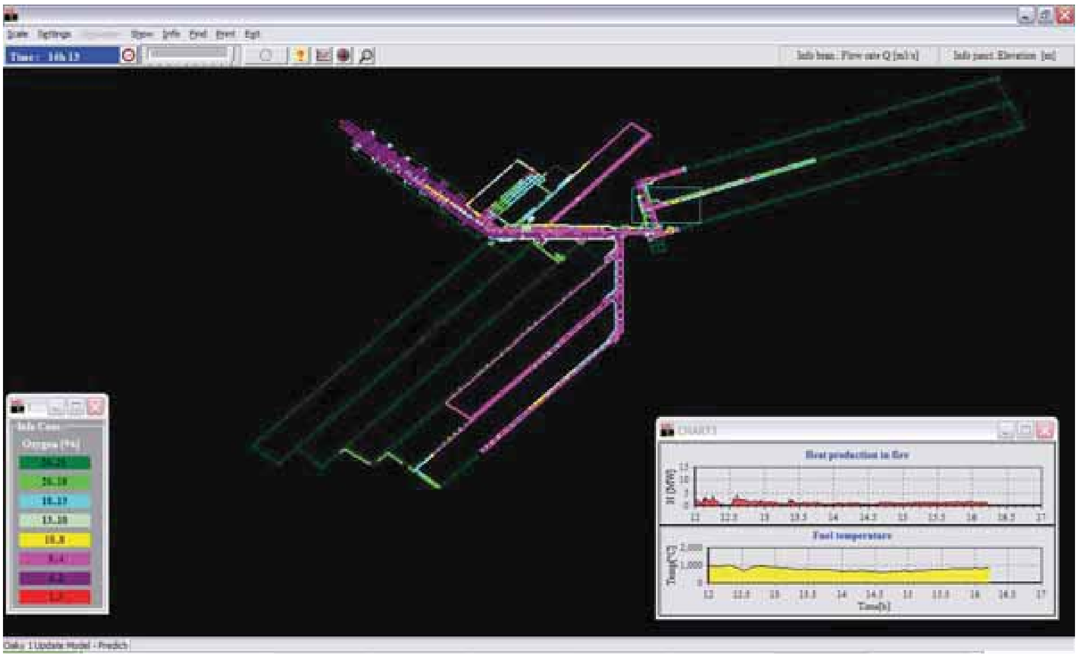


Figure 9.6 Oxygen distribution after 960 minutes.

Summary *Oxygen levels outbye the fire less than 2.9% after 960 minutes, however methane passing across fire with potential to cause explosions. Exhaust reaches fire but with dilution. Fire fluctuating and not reducing in intensity after 16 hrs.*

9.7. Summary of Scenarios Examined and Alternative Inertisation Strategies

A study has examined the potential for simulation of the effects of inertisation on fires within a mine ventilation network. The project involved applying the VENTGRAPH mine fire simulation software to preplan for situations created by mine fires. As an introduction some general conclusions from relevant work undertaken to date at a range of Australian coal mines is discussed.

Priority fire locations at mines with VENTGRAPH simulation models developed in an ACARP research project entitled “Mine Fire Simulation in Australian Mines using Computer Software” have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation). A review of 35 scenarios showed that there was no fire examined that achieved the situation in which GAG docking inerted the simulated fire to aid recovery in a timely manner. Further, only 20 percent of scenarios showed a situation in which the inertisation product went straight to the fire site even though it arrived with significant dilution from other ventilation air or leakage through stoppings.

Other introductory sections examined issues with borehole location and sizing for delivery of GAG output and the influence of stopping leakage on GAG exhaust dilution in parallel intake airways

The principal purpose of this study is examination of Oaky Creek No 1 case study priority fires selected from across the pit layout with five in the mains, two in development panel gateroad, two in a longwall panel and one in the newly formed longwall goaf. GAG inertisation strategies were examined for the ten cases and details of the development of the individual scenarios are set down in chapter 4. Following this in chapter 5 six case scenario studies were re-examined to evaluate whether a better inertisation strategy was possible through GAG relocation to an alternative portal docking station locations to deliver inert gas more directly to the fire site.

Table 9.2 Comparison of inertisation effects between original GAG operation and new segregation and/or GAG Docking Positions

No	Fire Location	Fire Type	GAG Position	Segregation Actions	Fan Actions	Outcomes
1	SE02 Drivehead Main Dips C9-C10	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust cannot reach fire until all fans off and air reversal occurs. Fire insignificant after 20 hrs. No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off ***(Category C)***.
1A	SE02 Drivehead Main Dips C9-C10	Oil → Coal	Portal C entry	Intake shaft closed; Close Drift A after #1 and #2 Fans off; Close Drift B after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Fire insignificant at 7 hours. No ventilation air reversal occurred across the fire ***(Category B)***.
2	SE03 Drivehead Main Dips C21-C22	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust cannot reach fire until all fans off and air reversal occurs. Fire insignificant after 24 hrs. No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off ***(Category C)***.
2A	SE03 Drivehead Main Dips C21-C22	Oil → Coal	Portal C entry	Intake shaft closed; Close Drift A after #1 and #2 Fans off; Close Drift B after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Fire insignificant at 7 hours. No ventilation air reversal occurred across the fire ***(Category B)***.
3	SE04 Drivehead East Mains C5-C6	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Fire insignificant after 17 hrs. No methane reversal across the fire but ventilation air reversal occurred over the fire after all fans turned off ***(Category C)***.
3A	SE04 Drivehead East Mains C5-C6	Oil → Coal	Portal C entry	Intake shaft closed; Close Drift A after #1 and #2 Fans	Fans turned off one by	Exhaust reaches fire but with dilution. Fire insignificant after 17 hrs. No methane reversal across the fire but ventilation air

				off; Close Drift B after #3 Fan off	one	reversal occurred over the fire after all fans turned off ***(Category C)***.
6	LW Friction Ignition Longwall 24 face	Gas → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Ventilation air reversal occurred over the fire after all fans turned off. Methane had reversal across the fire and explosion occurred ***(Category E)***.
6A	LW Friction Ignition Longwall 24 face	Gas → Coal	Portal B entry	Close Drift A after# 1 and #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Fire insignificant after 11.5 hrs. Exhaust reaches fire but with dilution. Ventilation air reversal occurred over the fire after all fans turned off. Methane had reversal across the fire but with very low oxygen level – no explosion occurs ***(Category C)***.
7	LW Goaf Spon Comb LW 24 Goaf heating	Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Fire insignificant with all fans off after 48 hrs. Ventilation air reversal occurred over the fire after all fans turned off. Methane had reversal across the fire ***(Category C)***.
7A	LW Goaf Spon Comb LW 24 Goaf heating	Coal	Portal B entry	Close Drift A after# 1 and #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Fire insignificant with all fans off after 48 hrs. No ventilation air reversal occurred across the fire ***(Category B)***.
9	MG26 Drivehead Sandy Creek East Mains D6 to MG26	Oil → Coal	Intake Shaft	Close Drift A after #1 Fan off; Close Drift B after #2 Fan off; Close Drift C after #3 Fan off	Fans turned off one by one	With GAG running fire intensity insignificant at 48 hours and oxygen level outbye fire at about 0.6 percent. No methane reversal over the fire with new GAG docking position. ***(Category C)***.
9A	MG26 Drivehead Sandy Creek East Mains D6 to MG26	Oil → Coal	Portal B entry	Intake shaft closed; Close Drift A after #1 and #2 Fans off; Close Drift C after #3 Fan off	Fans turned off one by one	Exhaust reaches fire but with dilution. Fire fluctuating and not reducing in intensity after 16 hrs. Ventilation air reversal occurred over the fire after all fans turned off. Methane had reversal across the fire ***(Category E)***.

9.7.1. Scenario 1

Scenario 1 examined a Mains belt fire at SE02 Drivehead Main Dips C9-C10. It was considered as a situation where inertisation by itself would not help extinguish the fire in the belt heading (Category C) as the GAG was docked at the Downcast Shaft entry connected to D Heading 3 ct on Main dips. The inert gases in this case were travelling alone Main Dips A and E headings and were unable to get into C Heading. Progressive turning off of the three main surface fans did after much time cause the fire to be extinguished through combustion caused reduction of oxygen aided by the addition of inert gases which reach the fire after alteration of the pit ventilation. With fans off seam methane emissions caused gas levels to build up in the panels and although the VENTGRAPH simulation did not show these recirculating across the fire this could be a dangerous situation.

The Scenario 1A reassessment of approaches to improve the inertisation strategy led to the decision to dock the GAG unit at the Portal C Heading entry. This position forced inert gas directly into C Heading and onto the fire and led to a satisfactory outcome (Category B). To avoid dilution portal doors in A and B were progressively closed and to avoid stalling Main surface fans were progressively turning off. As the inert gas pressurised the C Heading directly no ventilation air reversal occurred across the fire after the main surface fans were turned off.

9.7.2. Scenario 2

Scenario 2 examined a Mains belt fire further inbye than Scenario 1 at SE03 Drivehead Main Dips C21-C22. It was considered as a situation where inertisation by itself would not help extinguish the fire in the belt heading (Category C). The GAG was docked at the Downcast Shaft entry connected to D Heading 3 ct on Main dips. As a result the inert gases were travelling inbye along Main Dips A and E headings and couldn't get into C heading where the fire was. Progressive turning off of the three main surface fans did after much time cause the fire to be extinguished through combustion caused reduction of oxygen aided by the addition of inert gases which reach the fire after alteration of the pit ventilation. With fans off seam methane emissions caused gas levels to build up in the panels and although the VENTGRAPH simulation did not show these recirculating across the fire this could be a dangerous situation.

The Scenario 2A was a reassessment of approaches to improve the inertisation strategy by docking the GAG unit at the Portal C Heading entry. This position could introduce inert gas directly into C Heading and onto the fire and led to a more satisfactory outcome (Category B). To avoid dilution portal doors in A and B were progressively closed and to avoid stalling Main surface fans were progressively turning off. As the inert gas pressurised the C Heading directly, no ventilation air reversal occurred across the fire after the main surface fans were turned off.

9.7.3. Scenario 3

Scenario 3 examined a Mains belt fire at SE04 Drivehead in East Mains C5-C6. It was considered as a situation where inertisation by itself would not help extinguish the fire in the belt heading (Category C) as the GAG was docked at the Downcast Shaft entry connected to D Heading 3 ct on Main dips. Under this situation the inert gases were unable to get into C Heading but travelled inbye along Main Dips A and E headings. Progressive turning off of the three main surface fans did after much time cause the fire to be extinguished through combustion caused reduction of oxygen aided by the addition of inert gases which reach the fire after alteration of the pit ventilation. With fans off seam methane emissions caused gas levels to build up in the panels and although the VENTGRAPH simulation did not show these recirculating across the fire this could be a dangerous situation.

The Scenario 3A reassessment of approaches to improve the inertisation strategy led to the decision to dock the GAG unit at the Portal C Heading entry. It was hoped that this position could inject inert gas directly into C Headings and onto the fire. However, as segregations along C Heading was not completed at the last cut-through of Mains and the first cut through of East Mains, the inert gas was diluted and not able to deliver to a satisfactory outcome. To avoid dilution portal doors in A and B were progressively closed and to avoid stalling Main surface fans were progressively turning off. No improvement in the outcome (Category C) resulted from the new docking position.

9.7.4. Scenario 6

Scenario 6 examined a fire on LW 24 face at mid point caused by friction ignition of methane igniting coal. It was considered as an inertisation failure (Category E) in that use of the GAG did not cause stabilisation of the fire. Dilution of inert exhaust at pit bottom means little low oxygen air will effectively reach the fire. Progressive turning off the three main surface fans led, after 9 hours, to reversal of face air carrying explosible concentrations of methane over the fire which caused a large explosion.

Scenario 6A reassessment of approaches to improve the inertisation strategy led to the decision to dock the GAG unit at the Portal B Heading entry. This position could introduce inert gas through a more direct route into LW panel and onto the fire. To avoid dilution portal doors in A and C were progressively closed and to avoid stalling Main surface fans were progressively turning off.

Ventilation air reversal occurred over the fire after all fans were turned off. Methane had reversed across the fire but with very low oxygen levels no explosion occurs. The new GAG docking position prevented a potential methane explosion with a slight improvement in the outcome (Category E).

9.7.5. Scenario 7

Scenario 7 examined a spontaneous combustion fire in the longwall panel goaf on the MG side about 40m back from the face. It was considered as a situation where inertisation by itself would not help extinguish the fire in the goaf (Category C). Progressive turning off of the three main surface fans did after much time cause the fire to be extinguished through combustion caused reduction of oxygen aided by the addition of inert gases which reach the fire after alteration of the pit ventilation. With fans off seam methane emissions caused gas levels to build up in the panels to produce a minor methane burnoff and although the VENTGRAPH simulation did not show these recirculating across the fire this could be a dangerous situation.

Scenario 7A reassessment of approaches to improve the inertisation strategy led to the decision to dock the GAG unit at the Portal B Heading entry. This position could introduce inert gas through a more direct route into the LW panel and led to a more satisfactory outcome (Category B). To avoid dilution portal doors in A and C were progressively closed and to avoid stalling Main surface fans were progressively turning off.

Fire substantially reduced without GAG exhaust reaching the fire in the goaf but use of the GAG ensures full extinguishment.

9.7.6. Scenario 9

Scenario 9 examined a panel belt fire at MG26 Drivehead Sandy Creek East Mains D6 to MG26. It was considered as an inertisation failure (Category C) with the GAG was docked at Downcast Shaft entry connected to D Heading 3 ct on Main dips as the fire was not stabilised. The use of the GAG did eventually cause stabilisation of the fire. Progressive turning off of the three main surface fans did in time cause the fire to be reduced in intensity but not extinguished through combustion caused reduction of oxygen aided by the addition of inert gases. Seam methane emissions caused gas levels to build up in the panels however these did not recirculate across this Mains located fire.

The Scenario 9A reassessment of approaches to improve the inertisation strategy led to the decision to dock the GAG unit at the Portal B Heading entry. It was hoped that this position could inject inert gas more directly onto the fire. To avoid dilution portal doors in A and C were progressively closed and to avoid stalling Main surface fans were progressively turning off.

Ventilation air reversal still occurred over the fire after all fans were turned off. Methane had reversed across the fire but with very low oxygen levels no explosion occurs. No improvement in the outcome (Category E) resulted from the new docking position as the inert gas was mixed and diluted in the same way as Scenario 9 with the GAG docking at the Downcast shaft.

9.8. Conclusion and Recommendations

The principal focus of this study of inertisation strategies has been to examine priority fire locations and best approaches to stabilising of fires with availability of GAG inertisation. It was determined that Oaky Creek No 1 Mine has a mine layout under which some improvements could be made to inertisation strategies in the event of a major fire

Based on the results from the simulation actions described in Chapter 8 some scenarios under which an improved strategy was considered possible have been re-simulated with new approaches to inertisation. Outcomes for these re-simulated alternative scenarios were compared with the original simulation results as described in previous sections. A summary of the comparisons is shown in Table 9.3.

The approach taken to improve the effectiveness of the existing mine inertisation situation in the underground ventilation network was to try alternative Portal docking station locations through use of existing ventilation structures. It was assumed that men would be out of the mine and it would not be possible to change underground ventilation structures to alter or improve inertisation. It was also assumed that it would not be possible to drill new boreholes to intersect workings in event of a fire or to use the upcast shaft. The best inertisation strategy as determined from alternative simulation exercises for the six priority fire locations are summarised in Table 9.3.

Fire Number 1 gave an outcome in which this Mains fire was stabilised by docking the GAG to the Drift C Portal. Adjacent intake airways namely the Intake shaft and Drift A and B were progressively sealed as main fans were shut down. This approach allowed inertisation exhaust to move directly to the fire source although there was some dilution. Turning off all fans poses high risk issues in a gassy mine; however in this scenario the fire is in the Mains and so panel flow reversals of methane laden air was not presented as an issue. With GAG running, fire intensity was insignificant at 7 hours (2 hours after starting the GAG) and oxygen level outbye fire at less than 2.5 percent. No ventilation air reversal occurred across the fire.

Fire Number 2 gave a very similar outcome to Fire Number 1. This Mains fire was stabilised by docking the GAG to the Drift C Portal. With GAG running, fire intensity was insignificant at 7 hours (2 hours after starting the GAG).

Table 9.3 Summary of optimum outcomes for the six fire simulation revised exercises

No	Fire Location	Fire Type	GAG Location	Fan Action	Outcome	Category
1	SE02 Drivehead Main Dips C9-C10	Oil → Coal	Intake Shaft	All shut down	Fire stable at 20 hours, fire fumes over fire possibility.	C
1A	SE02 Drivehead Main Dips C9-C10	Oil → Coal	Portal C entry	All shut down	Fire stable at 7 hours.	B
2	SE03 Drivehead Main Dips C21-C22	Oil → Coal	Intake Shaft	All shut down	Fire stable at 24 hours, fire fumes over fire possibility.	C
2A	SE03 Drivehead Main Dips C21-C22	Oil → Coal	Portal C entry	All shut down	Fire stable at 7 hours.	B
3	SE04 Drivehead East Mains C5-C6	Oil → Coal	Intake Shaft	All shut down	Fire stable at 17 hours, fire fumes over fire possibility.	C
3A	SE04 Drivehead East Mains C5-C6	Oil → Coal	Portal C entry	All shut down	Fire stable at 17 hours, fire fumes over fire possibility.	C
6	LW Friction Ignition Longwall 24 face	Gas → Coal	Intake Shaft	All shut down	Reversal, gas explosion	E
6A	LW Friction Ignition Longwall 24 face	Gas → Coal	Portal B entry	All shut down	Fire stable at 11.5 hours, methane over fire possibility.	E
7	LW Goaf Spon Comb LW 24 Goaf heating	Coal	Intake Shaft	All shut down	Fire stable at 48 hours, fire fumes over fire possibility.	C
7A	LW Goaf Spon Comb LW 24 Goaf heating	Coal	Portal B entry	All shut down	Fire stable at 48 hours.	B
9	MG26 Drivehead Sandy Creek East Mains D6 to MG26	Oil → Coal	Intake Shaft	All shut down	Fire unstable at 12 hours and not extinguishing.	C
9A	MG26 Drivehead Sandy Creek East Mains D6 to MG26	Oil → Coal	Portal B entry	All shut down	Fire unstable at 16 hours and not extinguishing. Methane over fire possibility.	E

Fire Number 3 on first appearance gave a similar outcome to Fires Number 1 and 2. Again this Mains fire was stabilised by docking the GAG to the Drift C Portal. With GAG running, fire intensity was insignificant at 17 hours (12 hours after starting the GAG). The difference was substantial air came in the belt heading and caused dilution which slowed the inertisation exercise. Also ventilation air reversed across the fire after all fans were off which is potentially dangerous. Further segregation of the belt heading progressively down dip should overcome these dilution and belt air reversal issues.

Scenario 6 examined a fire on LW 24 face at mid point caused by friction ignition of methane igniting coal which with progressive turning off of the three main surface fans led to reversal of face and a large explosion. The new inertisation strategy of docking the GAG unit at the Portal B Heading entry introduced inert gas through a more direct route into the LW panel and onto the fire but still led to a not fully satisfactory outcome. Again progressive improvements in sealing Mains belt headings will allow less dilution of inert carrying air.

This fire is a long way from the GAG docking point and so inertisation under this scenario will be difficult. The new GAG docking position was an improvement and reduced chance of a potential methane explosion.

Fire Number 7 examined how a longwall spontaneous combustion goaf fire could be stabilised. With the original inerting docking point and fans off gas levels built up in the panel and atmospheric recirculation across the fire could have led to a dangerous situation. Docking the GAG unit at the Portal B Heading entry introduced inert gas through a more direct route into the LW panel and led to a more satisfactory outcome with the GAG ensuring full extinguishment.

Scenario 9 examined a panel belt fire at a considerable distance from the mine Portals. This was deemed an inertisation failure, as the fire was not stabilised. The reassessment led to the decision to dock the GAG unit at the Portal B Heading entry. It was hoped that this position could inject inert gas more directly onto the fire. No improvement in the outcome resulted from the new docking position with an unstable situation persisting.

In conclusion these ten fire simulation exercises have produced scenario results in three categories:

1. Those in which satisfactory inertisation can be achieved from use of the mine's current single docking point at the Main Intake Shaft. This applies to Scenarios 4, 5, 8 and 10.
2. Those in which a better and satisfactory inertisation strategy can be achieved from use of a docking point other than the Main Intake Shaft. This applies to Scenarios 1, 2 and 7. The other alternatives for docking were the Main Drift Headings B or C. It is recommended that GAG docking stations should in future be fabricated for all ventilation intake openings to the mine and currently for Drift Headings B and C.
3. Those in which an unsatisfactory inertisation outcome is achieved from use of the Main Intake Shaft and where the alternative reappraisal led to an unsatisfactory outcome. This applies to Scenarios 3, 6 and 9. Investigations which were outside the scope of this report could be undertaken for these three Scenarios to determine whether use of another access point to the mine, namely a specially excavated borehole would provide a satisfactory inertisation outcome.

Oaky No 1 mine has all current intake air portals close together. This means that some parts of the mine with active workings are at considerable distance from inertisation docking points on access intake airways. Strategically placed boreholes near active workings where priority fires may occur can be placed to advantageously allow inertisation when required.

It is recommended that Oaky No 1 mine examine how use of boreholes or other approaches could effectively allow satisfactory inertisation of priority fires locations used in Scenarios 3, 6 and 9.

General recommendations arising from the analyses are as follows:

1. GAG docking stations should be fabricated for all ventilation intake openings to the mine. The existing apparatus at the Main Intake Shaft should be supplemented by docking points at the Drift Headings and any future pit boreholes of appropriate diameter and future main shafts. In effect each docking point can deliver to a restricted geographic zone within the pit; multiple points allow the appropriate point to be utilised.
2. Segregation strategies simulated at points along the various Mains have shown that distribution of inert gases to separate Mains headings can be improved. Current segregation is less effective for fires located a long way inbye the mine and in the longwall production and development panels (due to increasing dilution through stoppings).
3. It is recommended that a borehole with a diameter of at least 1 m should be considered at the beginning of each panel for potential delivery of inert gases to each longwall production or development face. These boreholes can also be used for other purposes such as delivery of ballast or emergency extrication of people out of the mine. They may be used for other services. Incorporation of remote controlled doors should be considered to give control over which gateroad should be used to carry the inert gases into the panel.
4. Scenarios in which no satisfactory inertisation strategy was apparent should be further examined to determine the merits of locating a borehole or shat in the vicinity to enable satisfactory outcomes.

These fire simulation exercises have demonstrated that it is possible to efficiently evaluate possible inertisation strategies appropriate to a complex mine layout extracting a gassy seam and determine which approach strategy (if any) can be used to stabilise a mine in a timely fashion.

To support the report’s main findings some discussions on borehole delivery of inert gases and aspects of Mains segregation have been included. Some considerations for selecting the best surface portal location placement for the inertisation unit for most efficient suppression of a fire have been examined. There is a brief examination of the possibility of a wider and proactive application of GAG in Australian mines responding to or recovering from mine fires or spontaneous combustion heatings or elimination of the potential explosibility of newly sealed goafs is examined. The primary focus here is on systems involving delivery of GAG exhaust through docking to surface boreholes connecting into underground workings. Attainable designs for panel boreholes and how GAG docking to boreholes can improve delivery of GAG exhaust are discussed. Introduction of inert gases can present difficult emergency management decision making. Should the main mine fans be turned off to reduce dilution of the inert gas, or will this action cause, in conjunction with fire induced buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

Mine fires and heatings are recognised across the world as a major hazard issue. New approaches allowing improvement in understanding their use of inertisation techniques have been examined. The outcome of the project is that the mining industry is in an improved position in their understanding of mine fires, use of inertisation and the use of modern advances to preplan for the handling of possible emergency incidents.

10. PROACTIVE USE OF THE GAG INERTISATION UNIT USING MINE BOREHOLES

10.1. INTRODUCTION

The potential use of appropriately sized boreholes to deliver inertisation output directly to a fire or heating has advantages. An analysis has been undertaken of design considerations for varying diameter and depth boreholes taking into account backpressure considerations inherent in fluid flow through relatively small diameter borehole airways. This exercise examines the relevant theoretical thermodynamic theory required to understand flow behaviour in systems involving borehole delivery of GAG exhaust through docking to pre-drilled surface boreholes into underground workings. The study examines attainable designs for panel boreholes and how GAG docking to boreholes can improve delivery of GAG exhaust through a mine ventilation network.

10.2. Inertisation Through Boreholes

Economic installation of well placed boreholes could allow the proactive use of larger inertisation units such as the GAG in a wider application in Australian mines responding to or recovering from mine fires or spontaneous combustion heatings, the elimination of the potential explosibility of newly sealed goafs or in the making safe of old mine workings prior to final sealing.

Australian coal mines have experienced significant goaf heatings or goaf fires in recent years. Incidents at mines such as Dartbrook in 2002 and 2005/06, Austar in 2003/04, Moranbah North in 2004, North Goonyella in 2004/05 and Newstan in 2005/06 have caused significant loss of production time and in some cases mine reserves. Mine inertisation approaches relying on use of the Mineshield, Nitrogen Pressure Swing Adsorption (Floxal) and Tomlinson Boiler units have been used in these Australian recent mine incidents involving goaf heating. The low output of 2 m^3/s or less of these units has limited their success. The GAG has the ability to supply a much higher output at an operating cost advantage but has not been considered to date for these applications due to inability to deliver the inert exhaust to the affected area.

There is potential for an increased role for the GAG built on experience gained in the use of the GAG and other inertisation units in recent years. This can encompasses

- How GAG docking to boreholes can improve delivery of GAG inert gases to high priority potential fire locations particularly in working panels.
- How GAG docking to boreholes can be used to economically inert goaf spontaneous combustion incidents. More than five Australian collieries has experienced major goaf

heatings in recent years and the small inert gas units have not been of sufficient capacity.

- How GAG docking to boreholes can be used to inert goafs on sealing to avoid explosible atmospheres and movement of atmospheres into the “Coward Triangle”.

Boreholes placed within panels or more remote areas of mine workings have the capability of being used to deliver inert gases to nearby fires and so aid in mine recovery. Since the early 1990s drilling of boreholes through the overburden overlying worked underground seams has come a long way. Some major challenges with unstable strata have been overcome and a number of drilling companies service the market. Many collieries currently utilise one or more boreholes for ventilation or road base delivery purposes. Boreholes can also be used for man escape or delivery of GAG inert flow if necessary.

The challenge faced is how to effectively design these holes cost efficiently. The GAG has capability of delivering an exhaust stream of about 20 m^3/s although some of this is water vapour that quickly drops out of the air stream. There are limits to delivery of GAG output through different diameter holes at varying depths. Deeper holes naturally require larger diameter openings to overcome back pressure. Some require very large diameter boreholes of greater that 1.5 m that are prohibitively expensive.

Inertisation exhaust flow in deeper or smaller diameter holes faces significant back pressure. What is needed is a variable pressure fan that can be placed in line with the GAG flow and overcome substantial back pressure to allow holes of economical dimensions to be utilised.

A primary requirement is to examine attainable designs for panel boreholes under Australian conditions with current drilling technology. Part of this is to calculate design considerations for a variable pressure fan that can assist flow against back pressure. There is a limit to the contribution a variable pressure fan can make to assist flow. An objective will be to define the

- Hole designs (diameters and depths) that can deliver directly without assistance of any fan,
- Hole designs that can deliver with assistance of a fan and the pressure required for this delivery to be attained, and
- Specifications of boreholes design parameters that cannot achieve delivery even with fan assistance.

Inertisation users in Australia and in particular GAG operators such as Mines Rescue organisations need the answers to these questions for future planning. In particular detailed designs are needed by operating mines. Borehole drilling into operating mines has become common place in recent years and designs that allow multiple use for ventilation requirements, delivery of road base, potential man escape and delivery of inert gases provide a step forward for the industry. A systems involving borehole delivery of GAG exhaust is set out in Figure 10.1.

Development of such a system needs enhanced engineering understanding in a number of areas.

- Borehole design parameters need to be established applicable to Australian conditions based on the complex fluid flow theory that describes the dynamic, hot, pressurised exhaust carrying a superheated vapour. To investigate the possibility of using GAG in small diameter boreholes for either production inertisation or fire fighting purposes, it is necessary to understand GAG exhaust fluid behaviour. Steady flow energy equation based on Bernoulli's equation made applicable to compressible flow can be put in a form to describe the behaviour of GAG exhaust fluid being pushed down the borehole. Work needed to overcome resistance to flow exiting the GAG outlet can be evaluated as *Work to handle any issues of energy loss due to compression, work to overcome frictional rubbing drag on outlet walls, work to overcome shock losses, work to overcome elevational buoyancy effects and finally work to overcome water vapour super heating issues.* In the system of passing GAG exhaust down mine boreholes all components will be additive. These can be put in the form of an equation

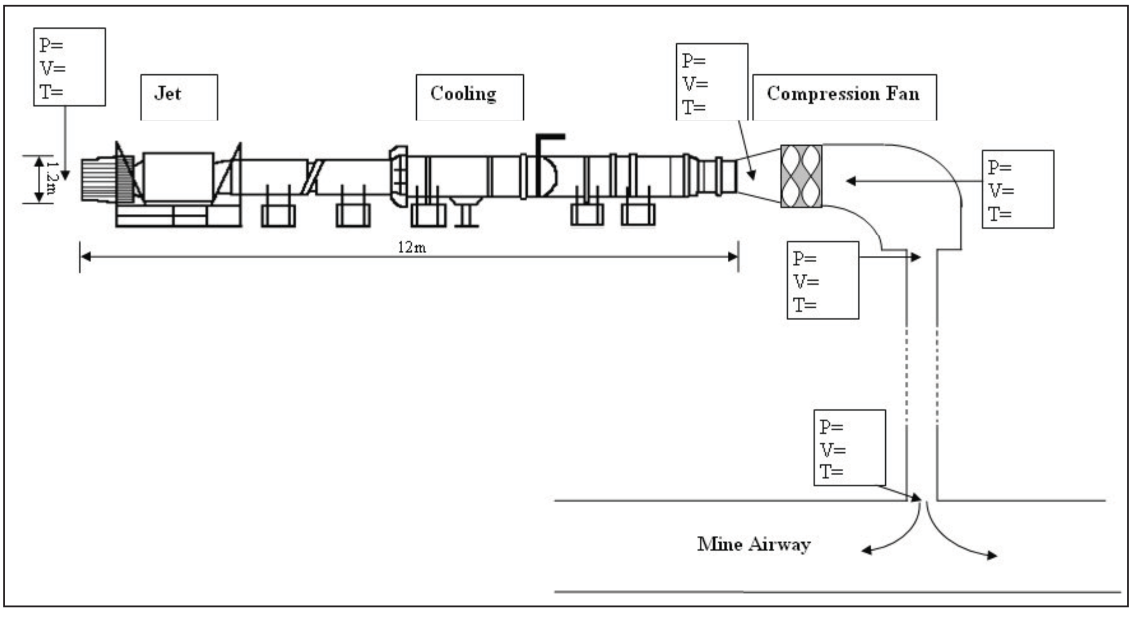


Figure 10.1 Schematic of system comprising GAG unit and compressor fan for borehole delivery

$$W_{12} = \int_1^2 VdP + F_{12} + \left(\frac{u_1^2 - u_2^2}{2}\right) + (z_1 - z_2)g + \text{superheat effects}$$

Where W = Work to achieve flow, VdP = Compression Work, F_{12} = Friction Impedance to fluid passing through pipe, u_1, u_2 = Fluid velocity terms, z_1, z_2 =

Elevation terms. Each component in the equation can be established separately by knowing various fluid flow conditions or parameters.

A discussion on some of the fundamentals of the thermodynamic theory pertinent to the operation of this system of a GAG engine, borehole delivery and assisting compressor fan is briefly set out in the following section.

- Determinations need to be made of the relationships between borehole back pressure and GAG thrust relationships.
- Determinations of the best variable pressure fan design that can be coupled to the system to overcome bask pressure need to be made.
- Determinations of how a variable pressure fan can be powered either through external sources or by direct coupling to the jet engine and utilisation of its potential power need to be made.
- Designs for automation of GAG operations need to be made. The GAG-3 gas turbine is a thrust engine and as such can be used against pressure for inertisation through a reduced diameter borehole. To accomplish this aim the GAG-3 has to be electronically controlled with spare I/O capacity for butterfly valve and proportional control on a tee piece in the exhaust delivery duct. The control system needs to react to duct pressure measurement input and output PID control of the engine rpm and afterburner fuel flow with dynamic measurement of backpressure against the turbine section.
- Mine layout and application of inert gases to fire or heating or to keep sealed area atmospheres out of the explosive range need to be considered. Any use of the GAG must examine its interaction with the complex ventilation behaviour underground during a substantial fire. VENTGRAPH simulation can be used to examine critical issues which include location of the GAG and boreholes for high priority fires or other issues, design dimensions of borehole or other passages required to deliver inert gases and back pressure issues, time required for inertisation output to interact with fire or other issues, effects of seam gas on fire behaviour with inertisation present, changes that can be safely made to the ventilation system during inertisation including switching off of some or all fan, and spontaneous combustion time frame issues.

10.3. Understanding GAG Exhaust Fluid Behaviour Down A Borehole

To investigate the possibility of using GAG in small diameter boreholes for either production inertisation or fire fighting purposes, it is necessary to understand GAG exhaust fluid behaviour.

The GAG-3A jet engine has ability to deliver a thrust of approximately 10 kN. This is effectively a pressure delivery of about 2 MPa. The GAG jet is set up to operate safely with effectively no thrust This is achieved by allowing exhaust to exit the unit across the full crow section of the outlet and there is no contraction to build up pressure. This works well when the GAG is delivering into a large cross section mine airway which creates little backpressure. This can be considered as free flow from the isolated GAG engine.

The discussion that follows has been developed to illustrate in a simplified form the major aspects that need to be considered in delivering jet exhaust down a borehole or through any passageway that creates significant back pressure. The analysis has introduced a compressor fan to assist motivation of the flow through the borehole. However this could as effectively be achieved by harnessing some of the potential thrust that the jet is capable of delivering in its normal mode of doing “real work” in powering an aircraft. The effects of the super heating on the system will vary with a number of conditions and need to be investigated further.

Steady flow energy equation based on Bernoulli’s equation made applicable to compressible flow can be put in a form to describe the behaviour of GAG exhaust fluid being pushed down a borehole. Work needed to overcome resistance to flow exiting the GAG outlet can be evaluated as Work to handle any issues of energy loss due to compression In the example this is simplifies as work associated with passage through a compressor fan), work to overcome frictional rubbing drag on outlet walls, work to overcome shock losses, work to overcome elevational buoyancy effects and finally work to overcome water vapour super heating issues. Depending on the configuration of the outlet conduit these components may not all be additive. However in the system of passing GAG exhaust down mine boreholes all components will be additive. These can be put in the form of an equation. A systems involving borehole delivery of GAG exhaust is set out in Figure 10.2.

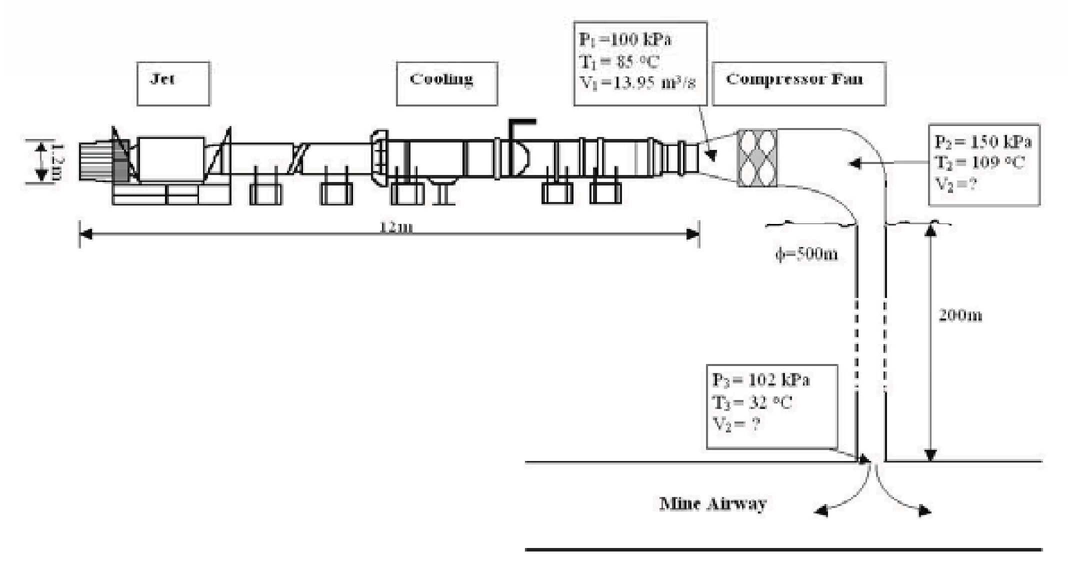


Figure 10.2 Schematic of GAG unit and compressor fan for borehole delivery.

$$W_{23} = \int_2^3 VdP + F_{23} + \left(\frac{u_2^2 - u_3^2}{2} \right) + (z_2 - z_3)g + \text{superheat}$$

(A derivative of McPherson, 1993 Equation 3.25, page 60)

where W_{23} = Work to achieve flow down the borehole, J/kg

VdP = Compression Work by compressor fan

F_{23} = Friction Impedance to fluid passing through pipe
u_2, u_3 = Fluid velocity terms, Shock loss
z_2, z_3 = Elevation terms
plus Superheated moisture energy.

(Superheated moisture energy may be important. This accounts for latent heat energy changes when steam is formed at the water boiling point (boiling point varies with the exhaust flow atmospheric pressure at the specific point). Superheated steam energy will be of greater importance under conditions when the exhaust mixture is forced through small diameter openings due to compression effects. This analysis has not gone into a detailed analysis of the mathematics of this energy transformation process).

10.3.1. The Fluid under Analysis - GAG output behaviour

Assume the GAG is operated at 7,200 rpm. From GAG operating information (Urosek, et al, 2004) as set down in Chapter 2, the GAG jet engine under free flow operating conditions will generate 13.95 m^3/s exhaust gas (0.5-2% O_2, 80-85% N_2, 13-19% CO_2) at 85^oC and atmospheric pressure of 100 kPa. Under this situation there is a requirement for 5.48 kg/s inhaled air, a consumption of 17 litres per minute of Jet A1 fuel (sg. 0.80 kg/m^3) and a mixing with the cooling water at a rate of 7.5 l/s (or 7.5 kg/s). A mass balance of the GAG system is as follows. Inputs to the GAG are

- Air - 5.48 kg/s
- Jet A1 fuel - 0.017 $m^3/min \div 60s \times 0.8\ kg/m^3 = 0.23$ kg/s
- Mixed cooling water - 7.50 kg/s

Thus the total inputs mass is 13.21 kg/s.

Output from the GAG is 13.95 m^3/s at 85^oC saturated conditions and atmospheric pressure of 100 kPa. Total output mass can be calculated by examination of psychometric properties as follows.

At outlet measurement point:

Saturated Vapour Pressure, P_{WS} $= 0.6105 \text{ Exp } (17.23 \times T_{WB}/(237.3 + T_{WB})$
$= 0.6105 \text{ Exp } (17.23 \times 85/(237.3 + 85)$
$= 58.04$ kPa

Apparent Specific volume, ASV $= 287.23 \times (T_{DB} + 273.15)/(P - P_{WS})$
$= 287.23 \times (85 + 273.15)/(100{,}000 – 58{,}040)$
$= 2.45\ m^3/kg$

Mass flow of dry air, m_a $= 13.95/2.45$
$= 5.69$ kg/s

True Density, ρ $= (P – 0.378\ P_W)/(287.23 \times (T_{DB} +273.15))$
$= (100{,}000 – 0.378 \times 58{,}040)/(287.33 \times (85 +273.15))$
$= 0.759\ kg/m^3$

Moisture content, r	= 0.622 × P_{WS}/(P - P_{WS})
	= 0.622 × 58.04/(100,000 – 58,040)
	= 0.860 kg/kg
Mass flow rate, m	= 13.95 × 0.759
	= 10.58 kg/s

This mass flow includes approximately 5.69 kg/s of dry air and 4.89 kg/s of water vapour which is added by the direct contact of water for cooling of the exhaust gas. There is an imbalance of (13.21 - 10.58 = 2.63) kg/s in the system. This imbalance is caused by the excess liquid) water droplets carried over in the exhaust (and into the mine) from the mixing cooling water. Therefore, a breakdown of the GAG exhaust gas can be arrived at as follows.

- Exhaust gas - 5.69 kg/s
- Water vapour - 4.89 kg/s
- Excess water droplets carried over - 2.63 kg/s

The excess water droplets in the exhaust would in part be super heated under compression conditions during the GAG exhaust down a borehole The following sections attempt to establish some understanding of the different components in the system delivering GAG exhaust down a borehole.

1. To Establish Work under Compression

$$W = \int_1^2 VdP = R(T_2 - T_1)\frac{\ln\left(P_2 / P_1\right)}{\ln\left(T_2 / T_1\right)}$$ J/kg, (McPherson, 1993 Equation 3.73)

Now from Figure 1 GAG Diagram, to establish Work change from Points 2 to 3 and assuming the use of a Compressor Fan of output = 50 kPa

If P_2 = 100 kPa (Atm) + 50 kPa (Comp Fan ΔP)
P_3 = 100 kPa (Atm) + (Pressure at depth)
R = Universal gas constant (From McPherson, 1993 table, Page 62) = 368.7

From General Gas Equation: $\frac{V_1}{V_2} = \frac{P_2}{P_1}\frac{T_1}{T_2}$ thus $\frac{T_3}{T_2} = \left(\frac{P_3}{P_2}\right)^{1-\frac{1}{n}}$

$\therefore T_3 = (273+85)\left(\frac{150}{100}\right)^{1-\frac{1}{1.2}}$ and Assuming Polytropic Conditions n = 1.2

$\therefore T_3 = 109°C$

$$\int_2^3 VdP = 368(109-85)\frac{\ln\left(150/100\right)}{\ln\left(109/85\right)}$$

$$= 368 \times (24 \times 6.15)$$

$$= 54.31 \text{ kJ/kg}$$

Work required is 54.31 kJ/kg × 10.62 kg/s = 576.77 kW

2. Friction Impedance in Descending Borehole

Assume Lines borehole ϕ = 500mm with a depth = 200m

Now pressure loss for compressed air in a pipe (or borehole) can be calculated by the following equation (from Borrows et al, 1982, Ch 9 Page 256).

$$\Delta P = \frac{R_f \times m^2 \times L}{\rho} \times 10^{-3}$$

where R_f = resistance factor, m^{-5}
m = mass flow rate, kg/s
L = pipe length, m
ρ = air density, kg/m^3

ρ is calculated from average at top of shaft T_2 = 109°C, P_2 = 150 kPa and at bottom, T_3 = 32°C and P_3 = 100 kPa using the following equation.

$$\rho = \frac{P \times 10^3}{RT}$$

$$\rho = \frac{(150+102)/2 \times 10^3}{368.7 \times (382+313)/2}$$

$$= 0.987 \text{ kg/m}^3$$

R_f for 500 mm pipe diameter is 0.36.\

$$\therefore \Delta P = 0.36 \times (10.62)^2 \times 200 \times 10^{-3} \times \frac{1}{0.987} = 8.23 \text{ kPa}$$

$$\therefore F_{23} = m \times \Delta P = 10.62 \times 8.23 = 87.4 \text{ kW}$$

3. Elevation component - work to overcome elevational buoyancy effects

Elevational buoyancy effects can be calculated by the following equation

$\rho g(Z_2 - Z_3) = 0.987 \times 9.81 \times (200) = 1{,}936.5$ Pa or 1.94 kPa

Work to overcome Elevational buoyancy effects is

$10.62 \times 1.94 = 20.6$ kW

4. Shock losses for exit into mine

Shock Losses $= x\dfrac{v^2}{2g}(Pa)$ Assume hole (entry and exit) $x \approx 1.0$

Q = 5.2 m^3/s at exit (at 32^{o}C, density of 1.143 kg/m^3 and with majority of moisture already having dropped out)

$$Vel = \frac{5.2}{\pi r^2} = \frac{5.2}{\pi \times 0.25^2} = 26.3\,m/s$$

Assume $x = 1.0$

$$\therefore Shock = 1.0 \times \frac{26.3^2}{2 \times 9.81} = 35.3Pa$$

$$\therefore Work = \frac{1}{\rho} \times 10.62 \times 0.035$$

$$\therefore Work = \frac{1}{1.143} \times 10.62 \times 0.035$$

$= 0.33$kW

Therefore, compressor fan would be required to input the follow work

$$W_{23} = \int_2^3 VdP + F_{12} + \left(\frac{u_2^2 - u_3^2}{2}\right) + (z_2 - z_3)g + \text{superheat}$$

The first four terms in the equation as worked out above are

$W_{23} = 576.8 + 87.4 + 20.6 + 0.33 = 685.13$ kW

Thus delivery of 13.95 m^3/s GAG exhaust down a 200 m borehole of 500mm in diameter would require at least 700 kW of energy without consideration of the super heating component.

10.3.2. Flow through various borehole designs

Inertisation exhaust flow through deeper or smaller diameter holes faces significant backpressure. A variable pressure fan placed in line with the GAG flow could overcome substantial backpressure to allow holes of economical dimensions to be utilised.

A primary requirement is to examine attainable designs for panel boreholes under Australian conditions with current drilling technology. Part of this is to calculate design considerations for a variable pressure fan that can assist flow against backpressure. There is a limit (assumed up to 50 kPa) to the contribution a variable pressure fan can make to assist flow. The following tables show attainable borehole sizes for free (up to 2 kPa and compressor fan assisted delivery with various amount of exhaust delivered. Three categories are shown in the table with different shadow colours. Dziurzyński, (2004) stated that the GAG could operate continuously against a backpressure of 2 kPa.

- Hole designs (size and depth) that can deliver directly without assistance of any fan,
- Hole designs that can deliver with assistance of a fan and the pressure required for this delivery to be attained, and
- Specifications of boreholes design parameters that cannot achieve delivery even with fan assistance.

Table 10.1 Attainable borehole sizes for free and compressor fan assisted delivery with various amount of exhaust delivered (Pressure shown in kPa)

Exhaust Q	Borehole	Diameter (mm)															
(m^3/s)	Depth(m)	100	200	300	400	500	600	700	800	900	1000	1200	1400	1600	1800	2000	2400
10.0	100	19453.7	607.93	80.06	19.00	6.23	2.50	1.16	0.59	0.329	0.195	0.08	0.04	0.02	0.01	0.01	0.00
	150	29180.6	911.89	120.08	28.50	9.34	3.75	1.74	0.89	0.494	0.292	0.12	0.05	0.03	0.02	0.01	0.00
	200	38907.4	1215.86	160.11	38.00	12.45	5.00	2.31	1.19	0.659	0.389	0.16	0.07	0.04	0.02	0.01	0.00
	250	48634.3	1519.82	200.14	47.49	15.56	6.25	2.89	1.48	0.824	0.486	0.20	0.09	0.05	0.03	0.02	0.01
	300	58361.1	1823.78	240.17	56.99	18.68	7.51	3.47	1.78	0.988	0.584	0.23	0.11	0.06	0.03	0.02	0.01
	350	68088.0	2127.75	280.20	66.49	21.79	8.76	4.05	2.08	1.153	0.681	0.27	0.13	0.06	0.04	0.02	0.01
	400	77814.8	2431.71	320.23	75.99	24.90	10.01	4.63	2.37	1.318	0.778	0.31	0.14	0.07	0.04	0.02	0.01
	450	87541.7	2735.68	360.25	85.49	28.01	11.26	5.21	2.67	1.483	0.875	0.35	0.16	0.08	0.05	0.03	0.01
	500	97268.5	3039.64	400.28	94.99	31.13	12.51	5.79	2.97	1.647	0.973	0.39	0.18	0.09	0.05	0.03	0.01
	550	106995.4	3343.60	440.31	104.49	34.24	13.76	6.37	3.27	1.812	1.070	0.43	0.20	0.10	0.06	0.03	0.01
	600	116722.2	3647.57	480.34	113.99	37.35	15.01	6.94	3.56	1.977	1.167	0.47	0.22	0.11	0.06	0.04	0.01

Exhaust Q	Borehole	Diameter (mm)															
(m^3/s)	Depth(m)	100	200	300	400	500	600	700	800	900	1000	1200	1400	1600	1800	2000	2400
15.0	100	43770.8	1367.84	180.13	42.74	14.01	5.63	2.60	1.34	0.74	0.44	0.18	0.08	0.04	0.02	0.01	0.01
	150	65656.2	2051.76	270.19	64.12	21.01	8.44	3.91	2.00	1.11	0.66	0.26	0.12	0.06	0.03	0.02	0.01
	200	87541.7	2735.68	360.25	85.49	28.01	11.26	5.21	2.67	1.48	0.88	0.35	0.16	0.08	0.05	0.03	0.01
	250	109427.1	3419.60	450.32	106.86	35.02	14.07	6.51	3.34	1.85	1.09	0.44	0.20	0.10	0.06	0.03	0.01
	300	131312.5	4103.51	540.38	128.23	42.02	16.89	7.81	4.01	2.22	1.31	0.53	0.24	0.13	0.07	0.04	0.02
	350	153197.9	4787.43	630.44	149.61	49.02	19.70	9.12	4.68	2.59	1.53	0.62	0.28	0.15	0.08	0.05	0.02
	400	175083.3	5471.35	720.51	170.98	56.03	22.52	10.42	5.34	2.97	1.75	0.70	0.33	0.17	0.09	0.05	0.02
	450	196968.7	6155.27	810.57	192.35	63.03	25.33	11.72	6.01	3.34	1.97	0.79	0.37	0.19	0.10	0.06	0.02
	500	218854.1	6839.19	900.63	213.72	70.03	28.14	13.02	6.68	3.71	2.19	0.88	0.41	0.21	0.12	0.07	0.03
	550	240739.5	7523.11	990.70	235.10	77.04	30.96	14.32	7.35	4.08	2.41	0.97	0.45	0.23	0.13	0.08	0.03
	600	262625.0	8207.03	1080.76	256.47	84.04	33.77	15.63	8.01	4.45	2.63	1.06	0.49	0.25	0.14	0.08	0.03

Exhaust Q	Borehole	Diameter (mm)															
(m^3/s)	Depth(m)	100	200	300	400	500	600	700	800	900	1000	1200	1400	1600	1800	2000	2400
20.0	100	77814.8	2431.71	320.23	75.99	24.90	10.01	4.63	2.37	1.32	0.78	0.31	0.14	0.07	0.04	0.02	0.01
	150	116722.2	3647.57	480.34	113.99	37.35	15.01	6.94	3.56	1.98	1.17	0.47	0.22	0.11	0.06	0.04	0.01
	200	155629.6	4863.43	640.45	151.98	49.80	20.01	9.26	4.75	2.64	1.56	0.63	0.29	0.15	0.08	0.05	0.02
	250	194537.0	6079.28	800.56	189.98	62.25	25.02	11.57	5.94	3.29	1.95	0.78	0.36	0.19	0.10	0.06	0.02
	300	233444.4	7295.14	960.68	227.97	74.70	30.02	13.89	7.12	3.95	2.33	0.94	0.43	0.22	0.12	0.07	0.03
	350	272351.8	8510.99	1120.79	265.97	87.15	35.02	16.20	8.31	4.61	2.72	1.09	0.51	0.26	0.14	0.09	0.03
	400	311259.2	9726.85	1280.90	303.96	99.60	40.03	18.52	9.50	5.27	3.11	1.25	0.58	0.30	0.16	0.10	0.04
	450	350166.6	10942.71	1441.01	341.96	112.05	45.03	20.83	10.69	5.93	3.50	1.41	0.65	0.33	0.19	0.11	0.04
	500	389074.0	12158.56	1601.13	379.96	124.50	50.04	23.15	11.87	6.59	3.89	1.56	0.72	0.37	0.21	0.12	0.05
	550	427981.4	13374.42	1761.24	417.95	136.95	55.04	25.46	13.06	7.25	4.28	1.72	0.80	0.41	0.23	0.13	0.05
	600	466888.8	14590.28	1921.35	455.95	149.40	60.04	27.78	14.25	7.91	4.67	1.88	0.87	0.45	0.25	0.15	0.06

In Table 10.1 it shows that if the borehole diameter is 800mm, the GAG can deliver 15 m^3/s of exhaust without assistance of a compressor fan to overcome the backpressure from the borehole for up to 100 m in borehole depth. However some fan assistance is required for the borehole depths in excess of 100 m.

For 500 mm borehole, it could deliver 15 m^3/s of exhaust for borehole depth up to 350 m with compressor fan assistance. When the borehole depth is more than 350m, it is not able to deliver 15 m^3/s of exhaust even with fan assistance but it is possible to deliver a lesser amount of exhaust of 10 m^3/s.

The following figures show various borehole designs (diameter/depth) for free delivery of various GAG exhaust quantities with a borehole frictional (back) pressure of less than 2 kPa and for fan assisted delivery with a borehole frictional (back) pressure of less than 50 kPa.

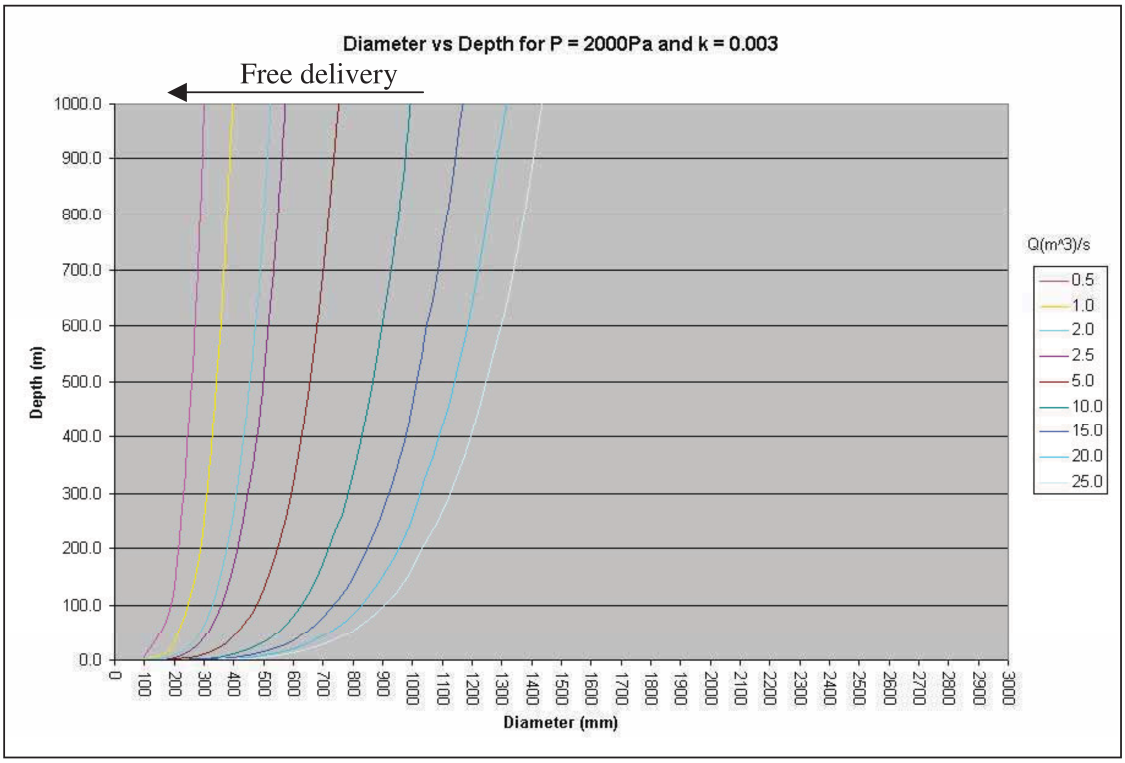
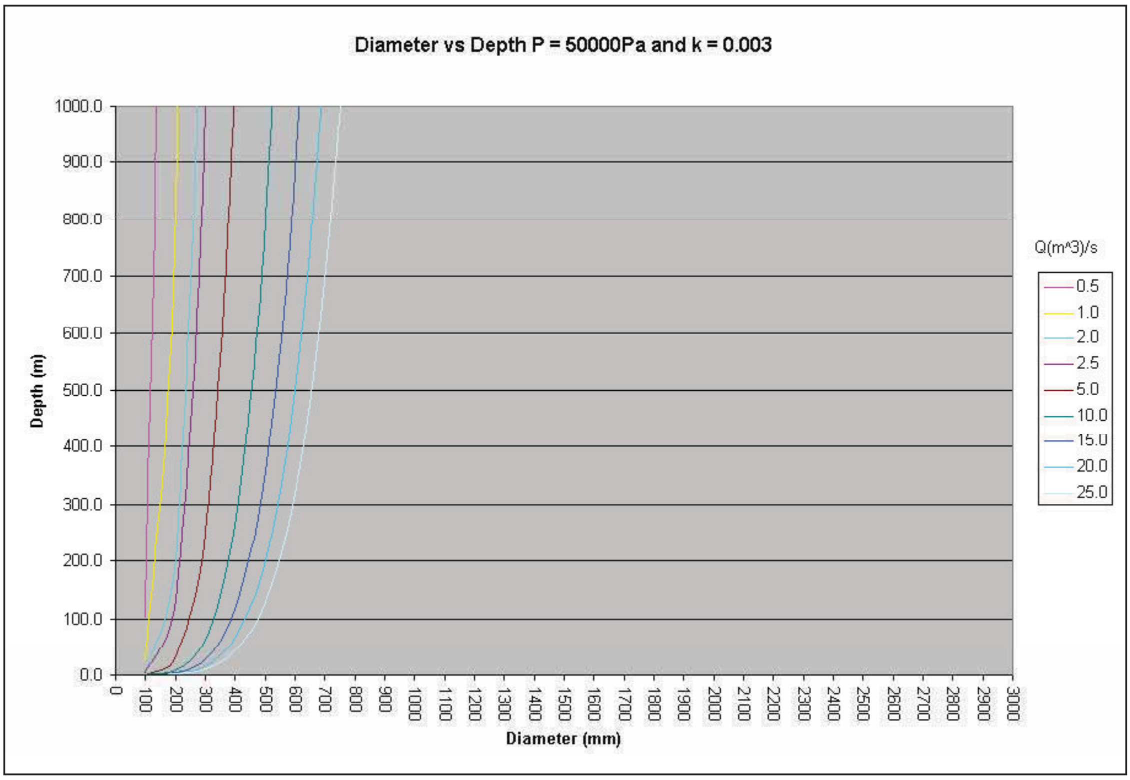


Figure 10.3 Borehole design for free delivery of GAG exhaust.



10.4. CONCLUSIONS

Borehole design parameters have been established applicable to Australian conditions based on the complex fluid flow theory that describes the dynamic, hot, pressurised exhaust carrying a superheated vapour. Determinations have been made of the relationships between borehole back pressure and GAG thrust relationships and the best approach to vary the jet engine thrust to overcome this bask pressure. These mathematical relationships can now be applied to investigate the possibility of using GAG in small diameter boreholes for either production inertisation or fire fighting purposes. This would be a verification exercise taking the equations describing GAG exhaust fluid behaviour based on the steady flow energy equation and comparing the theoretical predictions of GAG exhaust fluid behaviour with actual measurements of pressure, quantity and temperature at various locations downstream from GAG exhaust trials proposed.

11. CONCLUSION AND RECOMMENDATIONS

The primary objective of the project was to use mine fire simulation software to gain better understanding of how inertisation (GAG, Mineshield, Pressure Swing Adsorption and Tomlinson Boiler) units can interact with the complex ventilation behaviour underground during a substantial fire. Inertisation systems for handling underground fires, sealing of mine or mine sections, spontaneous combustion heatings and elimination of the potential explosibility of newly sealed goafs have been accepted as important safety approaches within the Australian industry.

Computer simulation of mine fires and effects on ventilation networks has been introduced to the industry with considerable interest and success. This has already put a significant number of mines in an improved position in their understanding of mine fires and the use of modern advances to preplan for mine fires and the handling of possible emergency incidents. The project has relied on substantial mine site and mines rescue support.

The project endeavoured to increase understanding of behaviour of mine fires in modern mine ventilation networks with the addition of inert gas streams. It also aimed to develop inertisation related modifications to the fire simulation software.

Inertisation has been accepted to have an important place in Australian mining emergency preparedness. The two jet engine exhaust GAG units purchased from Poland by the Queensland government in the late 1990s for the Queensland Mines Rescue Service have been tested and developed and mines made ready for their use in emergency and training exercises. Their use in real and trial mine fire incidents has underlined the need for more information on their application.

The NSW Mineshield (liquefied nitrogen) apparatus dates to the 1980s and has been actively used a number of times particular in goaf heating incidents. The Tomlinson (diesel exhaust) boiler has been purchased by a number of mines and is regularly used as a routine production tool to reduce the time in which a newly sealed goaf has an atmosphere “within the explosive range” and for goaf spontaneous combustion heatings. Nitrogen Pressure Swing Adsorption (Floxal) units are available and in use both for reducing time in which goafs are “within the explosive range” and for goaf spontaneous combustion heatings. Each of these facilities puts out very different flow rates of inert gases. Each is broadly designed for a different application although there is some overlap in potential usages.

Various types of inertisation systems currently available and in use in Australian coal mines for elimination of the potential explosibility of newly sealed goafs, for combating goaf spontaneous combustion heatings, for sealing of old mine workings or for stabilising fires in high priority locations have been examined. Systems have been compared to aid decision making in selection.

The potential for simulation of the effects of inertisation on fires within a mine ventilation network was examined. The project involved applying the VENTGRAPH mine fire simulation software to preplan for mine fires. Work undertaken to date at some Australian coal mines is discussed as examples. The effort has been built around the modelling of fire scenarios in selected different mine layouts.

Case studies have been developed to examine usage of the GAG inertisation unit. One section examined seam gas emissions in the face area; addition of the inert gas stream adds another level of complexity to the already complicated interrelationships between the mine ventilation system, the presence of seam gases and a mine fire. Should the main mine fans be turned off to reduce dilution of the inert gas, or will this action cause, in conjunction with fire induced buoyancy effects, airflow reversal and the drawing of combustion products or seam gases across a fire leading to an explosion?

Another section has focused on selection of the surface portal location for placement of the GAG for effective fire suppression. The difficulties that some current approaches present are highlighted. The advantages that can be gained from use of various inertisation docking positions depends on a number of considerations including the location of the fire, the relative distance from the inertisation docking portal location and the attributes and complexity of the mine ventilation network. Operation of a GAG unit requires preplanning in terms of infrastructure requirements for a GAG surface portal docking station and access for operating personnel, fuel, water and other operating requirements.

Priority fire locations at a wide selection of mines with a developed and current Ventgraph simulation model have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation). Many mine layouts were reviewed and from these 35 scenarios were considered appropriate for use of the GAG. These fires were categorised A to E in terms of ability of the GAG exhaust to effectively stabilise and extinguish the fire. As examples of results no fires met the category A description, 14 percent met category D and 20 percent met category E. The conclusion is that the current situation is not well placed to effectively inert most colliery priority fires.

These simulation exercises undertaken with a wide range of Australian mines focused attention to the situation that many potential underground mine fire sources cannot be successfully inertised with the GAG docked at the current specified point. This inability to deliver GAG output is particularly so for fires in extended areas of workings or in panels. Two important conclusions are

- Successful delivery of GAG output from units on the surface must consider other (that is alternative to Mains Travel or Conveyor Heading portals) delivery conduits directly into workings near the fire through existing or purpose drilled boreholes.

- During a fire the stopping of the main surface fan or fans will lead to rebalancing of pit ventilation and in some cases potential explosions through air reversals bringing poorly diluted explosible seam gases or fire products across the fire site.

Another section has looked at inertisation and dilution issues in Mains headings. These present a complex ventilation network and with additional interference from a fire, maintaining control of the movement of inert gas is more difficult than elsewhere in the mine. Even good quality segregation stoppings allow significant dilution of inertisation flows over relatively short distances. There is a section that has examined considerations presented by "punch" mines layouts. A number of recent punch longwall mines are accessed off highwalls. These mines have some provision for GAG docking from within the highwall pit but all have put down boreholes to workings which enable the GAG team to operate the engine from the surface.

The calibration exercise was in two parts. The back analysis of the gas monitoring data during a fire at the US Pattiki Mine showed that a VENTGRAPH model could be established to simulate satisfactorily this incident. The inertisation exercise during part sealing of the Newlands South workings (without a fire present) highlighted a number of findings.

- The GAG quantity measured exhausting from the mine area being sealed was at first considered to be unrealistically low. However further analysis, as detailed in Chapter 10 of this report, indicates that accounting for temperature and moisture mass changes explains any differences.
- The hypothesis that some of the GAG exhaust, with diurnal pressure changes, will flow into and out of goafs is of interest and needs to be accounted for.

Further monitoring of mine site GAG exercises are warranted to give greater understanding to this complex system.

A chapter has given a brief overview of the VENTGRAPH simulation software. It has highlighted the new features that have been added to the software as a consequence of this inertisation project and in particular the ability to use up to four different types of inertisation gases (at varying flow rates) across a mine layout simultaneously and the ability to include carbon dioxide and nitrogen seam gases as well as methane.

Exercises based on Oaky North and Oaky No 1 mines have involved "evaluation or auditing" of ability to deliver inert gases generated from GAG units to high priority underground fire locations. These exercises have been built around the use of the fire simulation computer program VENTGRAPH and modelling of fire scenarios across the mine layouts. A coding system, A to E, has been developed to assist interpretation within the audit exercises.

The primary objective of the exercises was to use mine fire simulation software to gain a better understanding of how inertisation approaches using the GAG jet engine exhaust can interact with the complex ventilation behaviour underground during a substantial fire.

The principal sections examining Oaky North Colliery focus on the development of scenarios for examining priority fire locations and firstly their effect on the mine ventilation system and secondly the influence of introducing inertisation gases to stabilise the fire. Inertisation outcomes in all case scenarios have been examined through introduction through the mine's present docking point at the Transport Drift. Each scenario has then been re-examined one or more times to establish if a different docking point, altered underground ventilation segregation or other approach would be more effective in stabilising the simulated fire.

Five major case study scenarios based on the modelling of fires with introduced inertisation in a number of high priority different points geographically spread within the Oaky North longwall mine layout have been discussed. Possible alternative strategies for successfully inerting the fires have been examined and conclusions drawn to the success or otherwise of these approaches. Approaches focus on use of alternative portal docking points, increased underground segregation and possible use of boreholes to delivery GAG exhaust directly to the fire seat.

These fire simulation exercises have shown that some priority Oaky North fires can be stabilised through GAG inertisation strategies. One scenario goaf fire strategy developed is a case in point where use of a panel borehole with careful segregation allowed a relatively fast outcome to be achieved. Another scenario development heading fire was similar in that a borehole GAG delivery gave the best outcome. Both these were achieved with one surface fan operating and maintaining minimum pit ventilation and seam methane dilution. A third scenario fire, a Mains belt fire, utilised the GAG positively through use of an alternative Portal for docking. These examples showed that the audit was a success in that it highlighted successful approaches to use of inertisation where the previous approach was inadequate.

On the other hand Mains belt and Development heading belt) scenario fires were placed such that alternative approaches to inertisation were ineffective because pit layout means excess dilution affects the GAG exhaust quality which can be brought to the fire.

Recommendations arising from the Oaky North analyses were as follows:

1. GAG docking stations should be fabricated for all ventilation intake openings to the mine. The existing apparatus at the Travel Decline should be supplemented by docking points at the Highwall portals, any pit boreholes of appropriate diameter and future main shafts. In effect each docking point can deliver to a restricted geographic zone within the pit; multiple points allow the appropriate point to be utilised.
2. Segregation strategies simulated at pit bottom areas have shown that distribution of inert gases to separate Mains headings can be improved. They were useful for fires

located inbye from the pit bottom in the Mains but were less effective for the fires located a long way further inbye and in the longwall production and development panels (due to increasing dilution through stoppings).

3. It was recommended that a borehole with a diameter of at least 1 m should be considered at the beginning of each panel for delivering inert gases to each longwall production or development face. These boreholes can also be used for other purposes such as delivery of ballast or emergency extraction of people out of the mine. They may be used for other services. Incorporation of remote controlled doors should be considered to give control over which gateroad should be used to carry the inert gases into the panel.

In the inertisation evaluation of Oaky No 1 Mine, ten major case study scenarios based on the modelling of fires with introduced inertisation in a number of high priority different points geographically spread within the longwall mine layout are discussed. Possible alternative strategies for successfully inerting the fires have been examined and conclusions drawn to the success or otherwise of these approaches. Approaches focus on use of alternative portal docking points, increased underground segregation and possible use of boreholes to delivery GAG exhaust directly to the fire seat.

These ten fire simulation exercises have produced scenario results in three categories:

1. Those in which satisfactory inertisation can be achieved from use of the mine's current single docking point at the Main Intake Shaft. This applies to four of the fire scenarios examined.
2. Those in which a better and satisfactory inertisation strategy can be achieved from use of a docking point other than the Main Intake Shaft. This applies to three scenarios examines. The other alternatives for docking were the Main Drift Headings B or C. It was recommended that GAG docking stations should in future be fabricated for all ventilation intake openings to the mine and currently for Drift Headings B and C.
3. Those in which an unsatisfactory inertisation outcome was achieved from use of the Main Intake Shaft and where the alternative reappraisal led to an unsatisfactory outcome. This applied to three scenarios examined. Investigations which were outside the scope of this report could be undertaken for these three scenarios to determine whether use of another access point to the mine, particularly a specially excavated borehole would provide a satisfactory inertisation outcome.

Oaky No 1 mine has all current intake air portals close together. This means that some parts of the mine with active workings are at considerable distance from inertisation docking points on access intake airways. Strategically placed boreholes near active workings where priority fires may occur could be placed advantageously to allow inertisation when required.

It was recommended that Oaky No 1 mine examine how use of boreholes or other approaches could effectively allow satisfactory inertisation of priority fires locations and particularly the three specific scenarios highlighted.

General recommendations arising from the Oaky No 1 Colliery analyses are as follows:

1. GAG docking stations should be fabricated for all ventilation intake openings to the mine. The existing apparatus at the Main Intake Shaft should be supplemented by docking points at the Drift Headings and any future pit boreholes of appropriate diameter and future main shafts. In effect each docking point can deliver to a restricted geographic zone within the pit; multiple points allow the appropriate point to be utilised.
2. Segregation strategies simulated at points along the various Mains have shown that distribution of inert gases to separate Mains headings can be improved. Current segregation is less effective for fires located a long way inbye the mine and in the longwall production and development panels (due to increasing dilution through stoppings).
3. It was recommended that a borehole with a diameter of at least 1 m should be considered at the beginning of each panel for potential delivery of inert gases to each longwall production or development face. These boreholes can also be used for other purposes such as delivery of ballast or emergency extrication of people out of the mine. They may be used for other services. Incorporation of remote controlled doors should be considered to give control over which gateroad should be used to carry the inert gases into the panel.
4. Scenarios in which no satisfactory inertisation strategy was apparent should be further examined to determine the merits of locating a borehole or shaft in the vicinity of the fire to enable satisfactory outcomes.

The fire simulation exercises at Oaky North and Oaky No 1 mines demonstrated that it is possible to efficiently evaluate possible inertisation strategies appropriate to a complex mine layout extracting a gassy seam and determine which approach strategy (if any) can be used to stabilise a mine in a timely fashion.

A final chapter has focused on borehole design parameters. Analyses have been established applicable to Australian conditions based on the complex fluid flow theory that describes the dynamic, hot, pressurised exhaust carrying a superheated vapour. Determinations have been made of the relationships between borehole back pressure and GAG thrust relationships and the best approach to vary the jet engine thrust to overcome this bask pressure. These mathematical relationships can now be applied to investigate the possibility of using GAG in small diameter boreholes for either production inertisation or fire fighting purposes. This would be a verification exercise taking the equations describing GAG exhaust fluid behaviour based on the steady flow energy equation and comparing the theoretical predictions of GAG exhaust fluid behaviour with actual measurements of pressure, quantity and temperature at various locations downstream from GAG exhaust trials proposed.

Mine fires and heatings are recognised across the world as a major hazard issue. New approaches allowing improvement in understanding their use of inertisation techniques have been examined. The outcome of the project is that the mining industry is in an improved position in their understanding of mine fires, use of inertisation and the use of modern advances to preplan for the handling of possible emergency incidents.

11.1. Additional Work

Based on the findings from this project, it is proposed that a study on production or proactive use of inertisation and particularly the GAG inertisation unit should be undertaken. The study should aim to examine the possibility of a wider and proactive application of GAG in Australian mines responding to or recovering from mine fires or inertisation of sealed mine workings or spontaneous combustion heatings or elimination of the potential explosibility of newly sealed goafs.

This project should take many of the findings from the current project report. Three of the main conclusions from this project are the objectives of this proposed project

- Positioning of the GAG inertisation units is a major determinant of potential success for most efficient suppression of a specific fire. Studies undertaken with most Australian underground collieries have concluded that the current situation is not well placed to effectively inert most colliery priority fires.
- There is a need to examine attainable designs for GAG inerting using panel boreholes under Australian conditions with current drilling technology. Part of this is to calculate design considerations to overcome backpressure. There is a limit to the ability of the GAG jet engine to deliver exhaust down smaller dimension borehole. The objective will be to define the
    - Hole designs (diameters and depths) that can deliver directly without assistance of any fan,
    - Hole designs that can deliver with modifications to the jet engine to improve thrust to overcome back pressure required for this delivery to be attained, and
    - Specifications of boreholes design parameters that cannot achieve delivery even with full GAG jet thrust.
- There is a need to examine the use of the GAG for production or proactive uses in a wider application in Australian mines responding to recovering from mine fires, spontaneous combustion heatings, elimination of the potential explosibility of newly sealed goafs or inert mines or mine sections on closure. Some of the current uses of low flow inertisation facilities could be more effectively undertaken with the GAG unit.

Any use of the GAG must examine its interaction with the complex ventilation behaviour underground during a substantial fire and fire simulation exercises will be undertaken using

Ventgraph software. Inertisation users in Australia and in particular GAG operators such as Mines Rescue organisations need the answers to these questions for future planning. In particular detailed designs are needed by operating mines. Borehole drilling into operating mines has become commonplace in recent years and designs that allow multiple use for ventilation requirements, delivery of road base, potential man escape and delivery of inert gases provide a step forward for the industry.

The current project has been examining the aspect of positioning of the inertisation units which is a major determinant of potential success for most efficient suppression of a specific fire. Priority fire locations at mines with a developed and current Ventgraph simulation model have been examined as to the ability of a GAG inertisation unit to inert a fire in the mine recovery stage. In the study it was assumed that the GAG would be docked at a prepared position designated by the mine (most commonly the current fabricated docking installation). The conclusion is that the current situation is not well placed to effectively inert most colliery priority fires.

These simulation exercises undertaken with a wide range of Australian mines focused attention to the situation that many potential underground mine fire sources cannot be successfully inertised with the GAG docked at the current specified point. This inability to deliver GAG output is particularly so for fires in extended areas of workings or in panels. Two important conclusions are

- Successful delivery of GAG output from units on the surface must consider other (that is alternative to Mains Travel or Conveyor Heading portals) delivery conduits directly into workings near the fire through existing or purpose drilled boreholes.
- During a fire the stopping of the main surface fan or fans will lead to rebalancing of pit ventilation and in some cases potential explosions through air reversals bringing poorly diluted explosible seam gases or fire products across the fire site.

A major project outcome will be that the GAG unit moves from a specialise facility only using in training or emergencies to one which is part of the production process. In this way the unit will be used frequently and will by necessity evolve in its applications and usage. More of the industry workforce will be trained in its usage, as it will be frequently brought to mine sites. This will reduce the real cost of training and it can be written off against the production particular use.

Some of the current uses of low flow inertisation facilities could be more effectively undertaken with the higher flow GAG unit. Sponsoring mines will have both a detailed design developed for specific borehole locations appropriate to their mine plan and simulation scenarios developed on how GAG exhaust could be utilised in elimination of the potential explosibility of newly sealed goafs or in combating goaf spontaneous combustion heatings.
The work program for project will be undertaken in two stages with second grant application to ACARP following successful completion of stage 1.

Stage 1

- Design for automation of GAG operation should be finalised and implemented. The automated GAG unit should be tested over a 10 day surface trial to establish back pressure versus thrust relationships and other operating parameters. A series of test are required to configure the control system for backpressure and can be combined with training sessions.
- To investigate the possibility of using GAG in small diameter boreholes, a verification exercise should be undertaken comparing the theoretical predictions with actual measurements of pressure, quantity and temperature at various locations downstream from GAG exhaust trials proposed.
- Determinations should be made of the relationships between borehole backpressure and GAG thrust relationships and the best approach to vary the jet engine thrust to overcome this bask pressure.
- Ventgraph simulation on mines of supporting companies should be used to examine critical issues such as location of the GAG and boreholes for high priority fires or related issues. Evaluations should be undertaken against real mine situations with the support of mine operators.

Stage 2

- A second phase should be proposed as a follow on to undertake a physical trial with a GAG coupled to a mine borehole to do the job as pre-tested initially in surface trials. This is very likely to be supported by a mine undertaking final closure and sealing of a mine section.

11.2. Completion of the Project

The ACARP Grant that has funded this study set down a list of activity stages that would be undertaken by the project to place the Australian mining industry in an improved position in understanding of use of inertisation in response to and recovery from mine fires.

- The status of the industry in its use of mine inertisation has been established. The technical specifications of units currently in use have been givens and various application described as set down inn chapter 2. Frequent liaison has occurred with mines rescue personnel in NSW and Queensland. A review of other inertisation sources that may be available to the mining industry has not occurred, as there is no documented evidence of any mine application of other sources in recent years.
- A review of international use of inertisation in modern mining practice particularly in the US and Europe has occurred as set down inn chapter 2.
- General mine simulations have been undertaken to assess how inertisation sources can be utilised in a fire emergency. A number of different situations have been examined as laid down in Chapter 3.
- Undertake simulations of the effects of common fire causes and fire progress rates with inertisation units simulated at more than one mine “docking” surface point to help mines

decide on optimal placement. Detailed aspects that were targeted for examination in a focused section of the report included:

- Location of the unit for high priority fire positions; e.g. portal docking position, special boreholes.
- Time required for inertisation output to interact with and extinguish a fire.
- Effects of seam gas on fire behaviour with inertisation present.
- Changes that can be safely made to the ventilation system during inertisation including switching off of some or all fans.

Simulations require a detailed study of a number of mine layouts to identify optimum portal placement for inertisation units for various underground fire locations. Comprehensive exercises have been undertaken over 15 different scenarios at Oaky North and Oaky No 1 Mines as described in chapters 6 to 9.

- Liaison has occurred throughout the project with the Polish program authors (by frequent email contact and visits by the Polish authors to Australia and visits by Australian researchers to Poland) to enhance the findings of the project and make inertisation related modification to the VENTGRAPH program from the project findings. Specific issues that were addressed included:
  - Variation in inertisation units outputs by quantity.
  - Approaches for including other inertisation devices apart from the GAG in simulations.
  - Discussions on GAG jet engine "fan" characteristic curves to allow calculation of ability of GAG to deliver through small diameter boreholes.
  - Ability to model additional seam gases.

  Changes that have occurred are set down in chapter 5.
- Preparation of teaching material on theory of fires, mine simulation and development of emergency management plans has occurred. It has been made use of in various areas of technology transfer as set down in chapter 12.
- Technology transfer from the project was considered vital and the various means by which this has occurred are comprehensively described in chapter 12. Significant presentations have occurred at mine sites in addition to industry seminars and conference deliveries. A bank of teaching material is available from these various efforts.

In addition to the specified outcomes significant and relevant development have occurred.

- The calibration exercises on VENTGRAPH for fire simulation and inertisation usage that have occurred are described in chapter 4.
- A rigorous mathematical thermodynamically based analysis has been developed to give understanding to the issue of delivery of GAG exhaust down small diameter boreholes as set down in chapter 10.
- The conclusion section in chapter 11 includes some recommendations on additional work that could follow from the project to answer a number of issues current in the industry. This additional work is related to the issues of borehole delivery and greater production usage of the GAG unit.

12. TECHNOLOGY TRANSFER

Technology Transfer has encompassed both presentations of papers at conferences, publication of papers in refereed Conference Proceedings and publication of papers in refereed Journals. It has included attendance at specialised Workshops. Furthermore discussions and presentations were made at various mine sites within Queensland and NSW to all levels of management, engineers, safety personnel and crewmembers.

12.1 Papers and Presentations

The following papers and presentations were given on the topic of fire simulation and inertisation during the grant's currency or in the immediate period before.

1. DEMONSTRATION OF VENTGRAPH MINE FIRE SIMULATOR
   ADS Gillies
   Queensland Department of Natural Resources and Mines Chief Inspector's Chief Executives' Meeting, Brisbane, December 2003

2. CASE STUDIES FROM SIMULATING MINE FIRES IN COAL MINES AND THEIR EFFECTS ON MINE VENTILATION SYSTEMS
   ADS Gillies and HW Wu
   Proceedings Fifth Australasian Coal Operators Conference, Ed. N. Aziz and B. Kininmonth, Aus. Inst. Min. Metall., Melbourne, pp. 111-125 February 2004.

3. LESSONS FROM TWO UNDERGROUND COAL FIRES IN AUSTRALIA USING RETROSPECTIVE NUMERICAL SIMULATION STUDY
   A.M. Wala, A.D.S. Gillies and H.W. Wu
   SME Annual Conference, Denver, February 2004

4. EFFECTS OF MINE FIRES ON MINE VENTILATION, GASES AND INERTISATION
   ADS Gillies and HW Wu
   Queensland Department of Natural Resources and Mines Inspectors' Meeting, Mackay, May 2004

5. EFFECTS OF MINE FIRES ON MINE VENTILATION PARTICULARLY IN GASSY MINES
   ADS Gillies and HW Wu
   Mine Managers' Association of Australia Seminar, Belmont NSW, May 2004

6. CASE STUDIES FROM APPLICATION OF NUMERICAL SIMULATION SOFTWARE TO EXAMINING THE EFFECTS OF FIRES ON MINE VENTILATION SYSTEMS
   H.W. Wu, A.D.S. Gillies & A.M. Wala
   Tenth US Mine Ventilation Symposium, Anchorage, Balkema, The Netherlands, pp. 445-455, May 2004.

7. SIMULATION OF MINE FIRES AND GAG USAGE
   ADS Gillies, HW Wu and R.S. Hosking
   Queensland Mines Rescue Service Inertisation Seminar, Mackay, May 2004

8. USE OF MINE FIRE SIMULATION FOR EMERGENCY PREPAREDNESS
   A.D.S. Gillies, H. Wu, D. Reece and R.S. Hosking
   Queensland Mining Industry Health and Safety Conference, Townsville, Queensland Resources Council, pp. 13-22, August 2004

9. AUSTRALIAN MINE EMERGENCY EXERCISES AIDED BY FIRE SIMULATION
   A.D.S. Gillies
   Third School of Mine Ventilation, Zakopane, Poland, 283-308, October 2004.

10. SPONTANEOUS COMBUSTION AND SIMULATION OF MINE FIRES AND THEIR EFFECTS ON MINE VENTILATION SYSTEMS
    ADS Gillies and HW Wu
    Coal Operators' Conference, Aus. Inst. Min. Metall., Melbourne, 225-236, April 2005.

11. INTRODUCTION OF SIMULATION SOFTWARE EXAMINING THE EFFECTS OF FIRES ON MINE VENTILATION SYSTEMS IN AUSTRALIA
    ADS Gillies, HW Wu and A Wala
    Eighth International mine Ventilation Congress, Brisbane, AusIMM, July 2005, pp 317-324.

12. AUSTRALIAN MINE EMERGENCY EXERCISES AIDED BY FIRE SIMULATION
    A.D.S. Gillies, H.W. Wu and A.M. Wala
    Archives of Mining Sciences, Polish Academy of Sciences, Krakow, Poland vol 50, issue 1 2005 pp 17-47.

13. FIRE SIMULATION ASSISTS MINE EMERGENCY TRAINING EXERCISES
    A.D.S Gillies and H.W. Wu
    31st Safety in Mines Research Institutes Conference, Safety in Mines Testing and Research Station, Brisbane, 254-260 October 2005.

14. QUEENSLAND MINE EMERGENCY LEVEL EXERCISES ASSISTED BY FIRE SIMULATION
    A.D.S Gillies and H.W. Wu
    Eleventh US Mine Ventilation Symposium, State College, Pennsylvania, Balkema, The Netherlands, 351-358, June 2006.

15. ISSUES IN USE OF INERTISATION OF FIRES IN AUSTRALIAN MINES.
    A.D.S Gillies and H.W. Wu
    Queensland Mining Industry Health and Safety Conference, Townsville, vol 1, 53-66 August 2006.

16. INERTISATION OF FIRES AND HEATINGS IN AUSTRALIAN MINES,
    A.D.S Gillies and H.W. Wu
    Proceedings, Fourth School of Mine Ventilation, Polish Acad. of Sci. Mine Ventilation Section of the Mining Committee, Cracow, October 2006 139-149.

17. GAG INERTISATION OF FIRES AND USE OF BOREHOLES,
    A.D.S Gillies and H.W. Wu
    Queensland Mine Rescue Service GAG Seminar, Mackay, 6 – 7 December 2006.

18. INERTISATION OF FIRES AND HEATINGS IN AUSTRALIAN MINES,
    A.D.S Gillies and H.W. Wu
    Archives of Mining Sciences, Polish Academy of Sciences, Krakow, Poland vol 52 issue 1 2007

12.2 Workshops

Workshops have included the following.

Two day Workshops to engineers from mines using fire simulation and inertisation software. These were held in Brisbane as follows

- December 2004
- December 2005
- March 2007

QMRS Workshops

The Queensland Mine Rescue Service organised Workshops in May 2004 and December 2006 at Mackay. Presentations of different aspects of inertisation were given at both of these.

Furthermore four days of Training on fire simulation and inertisation were given to QMRS Managers in 2005

NSW Workshop

Bruce Dowsett of Centennial kindly organised a NSW workshop at the beginning of the grant activities in April 2005. The purpose of the Workshop was to explain its aim and purpose and elucidate comments from participants.

Attendance at Inertisation Workshop, 4 April 2005, Newcastle

PERSON	COMPANY	POSITION	COMMENTS
Proud John	Mines Rescue	Training Coordinator	
Healey Paul	DPI	Inspector	Integrated into mine preparedness systems; establish responses.
Anderson Ian	DPI	Inspector	
Cowan Graham	DPI	Inspector	
Porteous Richard	Xstrata	Tech Support Manager	Rep Glenn Lewis. Establish contract interest within Xstrata.
Linde Gerard	United Colliery	Mining Engineer	Keen to apply; re-contact
Leggett Ray	DPI	Inspector	
Stoddard Ron	CFMEU	Check Inspector	
Davis Roger	Centennial-Mannery/Munmorah	Tech Services Manager	Showed interest
Shields Greg	Centennial-Mandalong	Ventilation Officer	Methods comparison, identify gas dangers. Showed interest
Cornford Peter	Centennial-Myuna	Ventilation Officer	Multi seam applications, other inertisation
Dowsett Bruce	Centennial	Group OHS Coord	Will continue to monitor interest in the Xstrata Group
Beikoff Steve	Xstrata-West Wallsend	Tech Services Manager	Inert limitations, mine factors required. Showed interest
Hempellstall John	Centennial	Group Manager OHS	
MacPherson David	Excel, Chain Valley	Manager	Showed interest
Justen Marc	Xstrata, Beltana	Ventilation Officer	Minimum standards, preparedness difficult to achieve
Bird Murray	Mines Rescue	Superintendent	Good training tool; program cannot do everything.
Glashoff Tony	Centennial-New Stan	Ventilation Officer	Unable to attend but interested
Bergin Peter	Centennial		Unable to attend but interested
Bracken Steve	Centennial		Unable to attend but interested
Mcalary Neville	Xstrata Head Office		Unable to attend but interested. In South Africa
Lewis Glenn	Xstrata Head Office		Unable to attend but interested, Rep by Richard Porteous
Myors Andrew	Centennial		Unable to attend but interested
McCreadie Lindsay	Mines Rescue Singleton		Unable to attend but interested
Enright Ken	Mines Rescue Singleton		Unable to attend but interested

Participants at the April 2005 workshop were give a Questionnaire.
The Questionnaire sheet and a summary of responses are produces below.

INERTISATION AND MINE FIRE SIMULATION QUESTIONNAIRE

Name: Company:
Email: Phone:

ACARP Grant: INERTISATION AND MINE FIRE SIMULATION USING COMPUTER SOFTWARE

Duration: 1 April 2005 – 31 March 2006

Purpose: The primary objective of the study is to use the Polish mine fire simulation software VENTGRAPH to gain better understanding of how inertisation (GAG, Mineshield, Nitrogen Pressure Swing Adsorption (Floxal) and Tomlinson Boiler and other) units interact with the complex ventilation behaviour underground during a substantial fire. Critical aspects targeted for examination under the project grant include location of the unit for high priority fire positions, size of borehole or pipe range required, underground segregation requirements, time required for inertisation output to interact with and extinguish a fire, effects of seam gas on fire behaviour with inertisation present and main fan management. A second major aim of the project is to take findings from the exercises tied to the above objectives to develop inertisation related modifications to the program in conjunction with the Polish program authors.

From an operating mine's point of view, what would you like to see the grant achieve?

- Technical evaluation of available inertisation facilities
- Development of approaches for simulation of effects of inertisation on mine ventilation systems under emergencies
- Appraisal of how mines should pre-prepare for use of inertisation e.g.. docking point, segregation, prioritising fire points etc
- Assistance to mine site for pre-preparedness for use of inertisation

INERTISATION AND MINE FIRE SIMULATION QUESTIONNAIRE RESULT SUMMARY

Of the 17 people attending the seminar, 7 participants responded in detail to the questionnaire. A summary of some comments and suggestions from the responses is as follows.

From an operating mine's point of view, what would you like to see the grant achieve? (comments or suggestions made in participants in ***italics and bold***).

- Technical evaluation of available inertisation facilities
  - ***Yes***
  - ***Limitations and applicability***
  - ***Multi seam applications***
  - ***Comparison between methods of inertisation***
- Development of approaches for simulation of effects of inertisation on mine ventilation systems under emergencies.
  - ***Yes***
  - ***More features required***
  - ***Other types of inertisation/combinations***
  - ***Simulation software capable of identifying dangerous gas mixtures/scenarios i.e. Recirculation.***
- Appraisal of how mines should pre-prepare for use of inertisation e.g.. docking point, segregation, prioritising fire points etc.
  - ***Yes, minimum standard recommendations not necessarily those of QLD but what will work for GAG, Mineshield, and PSA etc.***
  - ***Software able to save and compare results***
  - ***Ability to reverse changes made***
- Assistance to mine site for pre-preparedness for use of inertisation.
  - ***Integrated into other mine safety systems, e.g.. Emergency preparedness, mine fires, spontaneous combustion management***
  - ***Establish "standard" scenarios with prepared responses***
  - ***No, too many mine sites, life-of-mine plans extensive and not enough time and money.***

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2005

The Ability of Current Gas Monitoring Techniques to Adequately Detect Spontaneous Combustion

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The Ability of Current Gas Monitoring Techniques to Adequately Detect Spontaneous Combustion

D Cliff[1]

ABSTRACT

This paper investigates the adequacy of current gas monitoring techniques to adequately detect spontaneous combustion in underground coalmines. Despite being in the 21st century spontaneous combustion continues to occur in underground coalmines sometimes being detected only at a very advanced stage. Control of the incident is often then very expensive and time consuming.

The adequacy needs to be assessed not only from the point of view of the analysis technique be it tube bundle, gas chromatograph or real time sensor but also the number, location and sampling frequency of the monitoring locations.

Recommendations are made to optimise monitoring processes and recognise limitations of current techniques.

INTRODUCTION

There are three key questions that need to be addressed when designing a mine environment monitoring system:

- What are you trying to monitor?
- Where are you going to monitor? and
- How are you going to monitor?

Determining the answer to the first question will define the boundary conditions for defining the answers to the second and third questions.

WHAT TO MONITOR?

The focus of this paper is monitoring for the detection of spontaneous combustion, however mines are required to monitor for a range of situations including, safe working conditions for the workers, outburst prevention, equipment fires and statutory monitoring requirements, eg ERZ in Queensland.

Classically monitoring for the detection of spontaneous combustion has focussed on gas monitoring and exceeding of predetermined maximum allowable values for gas concentrations (eg carbon monoxide) or derived indicators such as Graham's ratio. Most of these indicators have been derived based upon either laboratory testing or events in underground coalmines that occurred many years ago. Often the conditions in these mines bore no similarity to modern underground coalmines. Historically for example, spontaneous combustion events would often occur in the pillars of roadways and were detected by smell or a rise in CO make. Now the majority of incidents occur in the goaf some distance behind the longwall where there is no externally defined ventilation circuit. Thus it is unreasonable to expect that textbook definitions of indicators can be routinely applied without significant modification and testing for relevance.

Events in the recent past such as at Southland, North Goonyella and Dartbrook indicate that this detection process is less than perfect. This is due to the size of the area to be monitored and the inability to sample within goafs. The lack of defined airflows also hampers early detection. Indeed due to the difficulties of monitoring for the presence of heatings in the goafs of modern longwall mines there needs to be a shift from detection of a heating to detection to prevent a heating.

1. MAusIMM, Minerals Industry Safety and Health Centre, Sustainable Minerals Institute, The University of Queensland, Brisbane Qld 4072.

In each of these cases a heating developed in the goaf of a longwall panel some distance behind the face. The heating was detected when gas samples were taken through a seal into the goaf that revealed abnormal CO and H_2 concentrations. Initially there was little indication of the location of the actual heating.

In two of the cases the application of inert gas into the goaf controlled the heating. Unfortunately in the third case the heating developed so rapidly that it became a raging fire and sealing at the surface was the only option, after inertisation was tried.

The gestation period of the heating in each case is unknown except that a maximum value can be established from the time the goaf was established. In two cases there was no indication of a worsening situation, in part due to the absence of regular gas monitoring through the seals. Local conditions, such as water blocking access to the seal prevented sampling in one case.

In one case it was only after sampling from a line of seals that it was determined that the heating was remote to the original detection point, indeed on the opposite side of the goaf, fed by air from the face. The treatment of the heating was protracted and it is likely that several lesser intensity oxidation occurrences initiated subsequent to the original heating.

For another case after sampling along the gate road into the goaf at various seal locations, the seat of the heating was determined to be close to a particular gate-road seal, and a surface borehole was able to intersect the heating allowing the application of inert gas directly onto it.

In the third case there is still today no definite evidence to locate the source of the heating. In each case however there is no way of knowing the genesis of the heatings in terms of what caused that particular area of goaf coal to abnormally oxidise and not the millions of tonnes of other coal in the goaf all around it. Circumstances at that point must just have been right for it to propagate. The initiation of the event in each case probably occurred months beforehand and the oxidation stewed away until conditions favoured acceleration. In two of the cases this was caused by sudden influx of additional air due to seal failures. In the third case it was probably simply a case of the longwall had been stationary for a number of weeks and air was able to continually flow to the heating site, under conditions that favoured abnormal oxidation.

Spontaneous combustion is a complex process and the chemistry of the process is still not well characterised. Laboratory experiments at SIMTARS (see for example Cliff *et al*, 2000) and UQ (Beamish, Barakat and St George, 2001) clearly show the complexity involved when coal reacts with air. Figure 1 depicts a 'typical' bituminous coal molecule. Coal is of course not a simple molecule rather it is a complex mixture of a range of large organic molecules containing carbon, oxygen, hydrogen, nitrogen and sulfur. Add to this impurities such as carbonates, pyrites and salts, stir in seam gases (methane and/or carbon dioxide) and water and you get coal as we know it. Some parts of the coal are far more reactive than others.

For example when methane is oxidised to carbon dioxide it goes through a series of intermediate compounds – methanol to formaldehyde to formic acid to carbon dioxide.

$$CH_4 – CH_3OH - HCHO – HCOOH – CO_2$$

The hardest step to achieve is the first step; methane is very unreactive and needs a lot of help (energy and catalysis) to begin

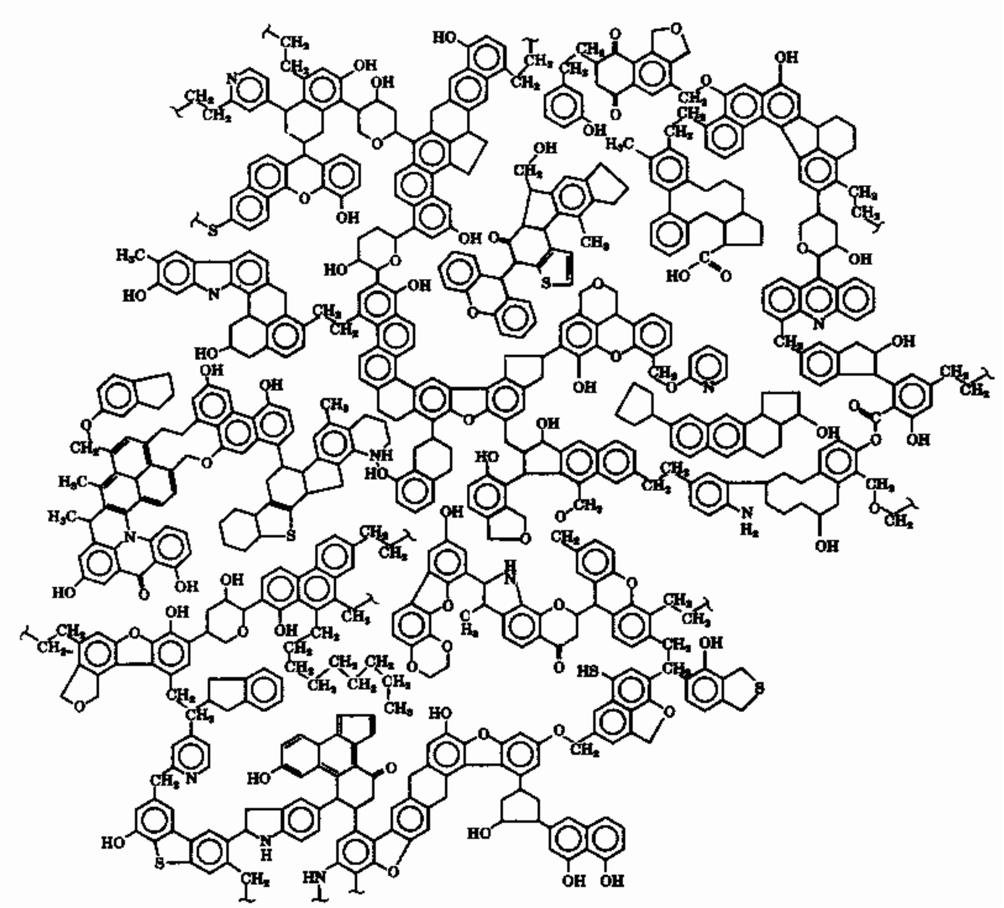

FIG 1 - Example of molecular structure of coal (Shinn, 1984).

the process. The further to the right the process proceeds the easier it becomes. Thus acid functional groups are very reactive and alkyl functional groups are not. In Figure 1, the bits around the molecule labelled – CHx are thus unreactive whereas those containing oxygen are more reactive. Not surprisingly low rank coals contain more of the oxygenated species than high rank coals and hence have a higher inherent reactivity.

Of course being a mixture of many chemical components it means that the oxidation chemistry is also complex. Figure 2 below indicates a simplified model of the oxidation process. Even in this model each reaction step has its own temperature dependence as well as individual dependence on the concentration of the reactants. Coal reacts as a solid and thus the effective surface area available for reaction is an important factor. If oxygen cannot get into the coal to reach the reactive components of the coal then reaction cannot occur. In other words the presence of water and seam gas within the pores of the coal reduces the effective surface area of the coal available to react and hence the potential for the coal to heat up and proceed to spontaneous combustion is also reduced.

Similarly if the most reactive components of the coal macromolecule have already reacted, then the rate of oxidation is substantially reduced, ie if a coal has been exposed to air for a long time, the reactive components will have reacted and the heat will have dissipated to the atmosphere, the residual 'weathered' coal will be unreactive. A more detailed description of the chemistry of coal oxidation can be found in Cliff and Bofinger (1998).

Figure 3 illustrates two tests carried out on Dartbrook coal samples in the large-scale (16 tonne) reactor at SIMTARS (Cliff *et al*, 2000). It can clearly be seen that the coal apparently lies dormant for many days and then suddenly the oxidation process accelerates out of control in a few hours. This translates to negligible gas concentrations and ratios suddenly becoming huge. In the case of the run of mine test, the CO make went from less than 1 L/min to over 100 L/min in less than 24 hours.

This is consistent with the laboratory observation that for every ten degrees increase in reaction temperature there is a doubling in the nett reaction rate and thus gas evolution rate. The dormant period appears to align with the dehydration of the coal and thus the energy being generated by the oxidation process is being absorbed by the energy requirements to volatilise the water out of the coal. Once this process is complete then the energy is channelled instead into heating the coal.

What this all means is that given the difficulty in detecting an active heating we should focus on preventing a heating from occurring. Monitoring strategies defined by an early response should be triggered by such things as:

- The detection of oxygen in areas of the goaf where it should not be. This does not immediately cause trouble but it will be the catalyst if this condition remains in place for any length of time. Remedial action to reduce the oxygen supply can avert a heating. Such action could include proactive inertisation, tightening of seals and reducing the pressure difference across the face of the longwall.
- The ability of oxygen to pass into areas of particular coal in the goaf for longer than normal, eg if the longwall stops for any length of time or is reduced to slow production rates.
- Pressure differences across seals that are not what is expected – this of course presumes that you know what to expect. Abnormal pressures differences often indicate leaking seals and air ingress into goaf areas.

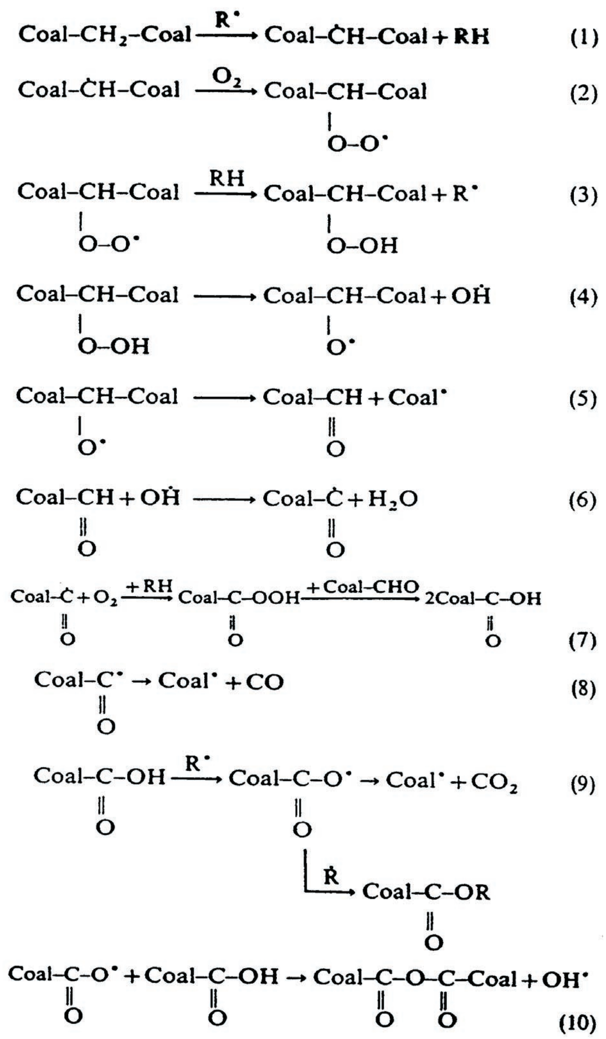

FIG 2 - Coal oxidation reaction scheme (Wells and Smoot, 1991).

- The detection of unusual trends in gas behaviour particularly carbon monoxide. For example, generally as the longwall retreats away from a gate road seal, the oxidation behaviour of the coal goes through a maximum and tails away, due to the reduced availability of oxygen and the weathering of the coal. A peak in CO is typically seen one or two cut throughs behind the face. If a peak is found further back or the smooth trend behaviour from seal to seal is not observed then something unusual is happening.
- Knowing the temperature profile across the face and around the goaf so that abnormal temperatures can be identified early. Temperature is very important as it affects the reaction rates as discussed above.

In the past too much reliance has been placed on small-scale laboratory testing such as R70 determinations, or in gas evolution testing. These have limited application to the real world, as the laboratory conditions bear no resemblance to those conditions found in a longwall goaf. R70 tests for example are carried out on small samples of dried crushed coal, which has been degassed. Often the tests are carried out under conditions where the airflow through the sample is much higher than would be found in the goaf. This means that the balance of reaction mechanisms depicted in Figure 2 above will differ from that in the goaf and hence give different gas evolution behaviour.

Medium and large-scale testing using run of mine coal and more realistic airflows often give very different results. For example the two-metre column work of Beamish *et al* (2002 and 2003), has been able to demonstrate that significant levels of hydrogen can be generated at temperatures much less than 100°C. These results are consistent with an oxidation pathway where coal reacts with air in the presence of water vapour, and internally rearranges itself to generate carbon dioxide and hydrogen. A separate oxidation pathway appears to generate a mixture of carbon monoxide and carbon dioxide.

WHERE TO MONITOR?

Monitoring needs to be undertaken to ensure that normal behaviour can be characterised and that abnormality can be detected as soon as possible. Too often mines collect inadequate

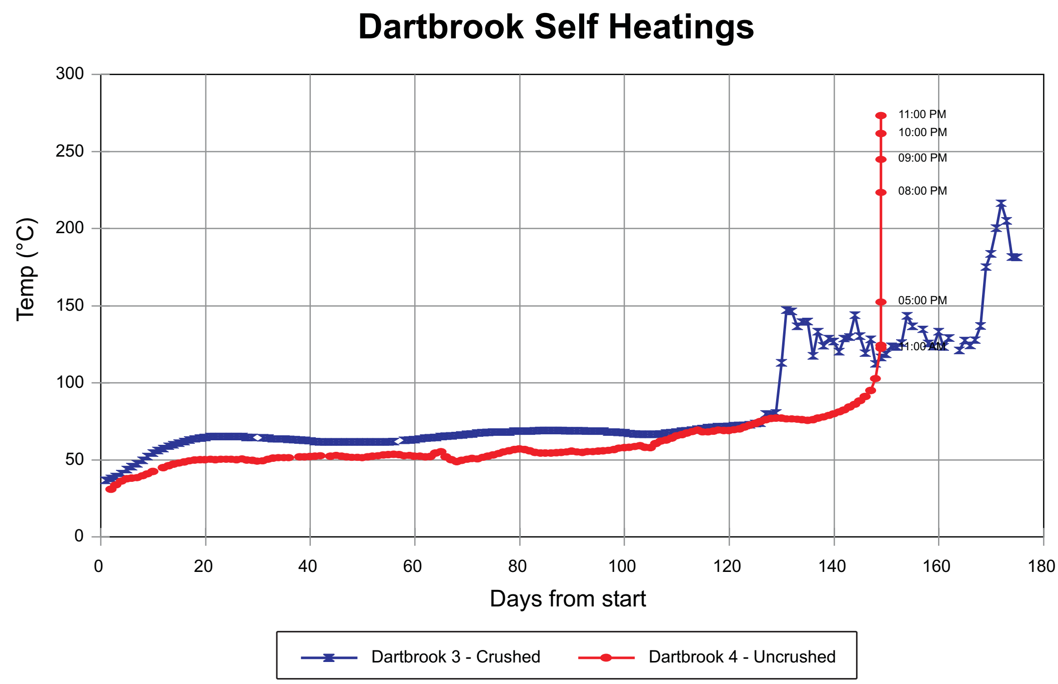

FIG 3 - Temperature time plots for large-scale oxidation tests (Cliff *et al*, 2000).

amounts of data from far too few monitoring locations. This means that when something abnormal is detected, the situation is often serious and evacuation of the mine is the only option. Typical goaf seal behaviour needs to be determined including:

- pressure differentials across seals and around goafs as a function of distance from the face and other factors such as the change in the pressure difference across the face;
- gas evolution and derived indicators as a function of distance from the face, this is especially important where factors such as goaf drainage and or back bye ventilation is used to reduce seam gas impacts on the face; and
- longwall return concentrations and derived indicators such as CO make as a function of operating parameters including size of goaf, rate of retreat, etc.

Monitoring is a multi-tier process. Initially characterisation of normal goaf and return roadway behaviour will be an intensive campaign until sufficient data has been gathered to give the mine confidence that it knows what to expect. Once this initial characterisation is complete then the monitoring can be tailored to check that the expected behaviour is occurring.

HOW TO MONITOR?

Frequency and complexity of sampling will depend on what is being monitored. Continuous monitoring of panel returns is required by regulation. Seal sampling can be undertaken by a mix of tube bundle sampling with analysis by a bank of infrared analysers for CO, CO_2 and CH_4, paramagnetic for O_2, and bag samples with analysis by gas chromatograph for the seals immediately inbye the face and perhaps just bag samples taken on a regular basis for those seals toward the rear of the panel or if there is spare tube bundle capacity, these tubes could be sampled less frequently than the more important (more likely to change) tubes. It may not be necessary to sample every seal if there is no indication of abnormality in terms of pressure differentials or oxygen presence. Gas chromatography gives the opportunity to directly analyse for the presence of hydrogen and higher hydrocarbons as well as check the accuracy of the tube bundle system. The older the coal the harder in general it is to resuscitate it for abnormal oxidation.

DETECTION

Much has been written about the use of indicators for detecting spontaneous combustion. There is only one real law for spontaneous combustion monitoring – *there is no universal indicator.*

There are many indicators that have been used to detect spontaneous combustion including:

- Carbon monoxide concentration. This is very unreliable in isolation as CO is produced at all temperatures by coal oxidation and extensive oxidation over a long time may well produce the same concentration as a small much more intensive oxidation event.
- Hydrogen concentration. This is also unreliable in isolation as the work of Beamish *et al* (2003) and Nehemia, Davidi and Cohen (1999) have demonstrated.
- Ethane. Ethane is most commonly present as a minor seam gas typically between 1/100 and 1/1000 of the methane present. *Do not use it as an indicator of spontaneous combustion.*
- Ethylene concentration. At high temperatures (> 200°C) the coal will pyrolyse and produce a raft of unsaturated and saturated hydrocarbons. The presence of significant concentrations of ethylene (>10 ppm) is a reliable indication that abnormal oxidation has occurred. Typically by the time this occurs the CO and hydrogen concentrations are orders of magnitude higher than this and rising.
- CO make. This is only valid in roadways with defined, known ventilation. Absolute numbers only have meaning when they are calibrated against the actual mine performance and operating conditions (see above). Real time air velocity/air quantity sensors are readily available and allow for real time monitoring of the ventilation flows as well as determining makes.
- Graham's ratio. This ratio can be a useful indicator of advanced oxidation however it is possible that effects of a small intense heating will be hidden in the effects of a large-scale low-level oxidation. This will cause Graham's ratio to underestimate the intensity of any oxidation process. It is thus important to set triggers based on normal behaviour at the mine rather than textbook values.
- CO/CO_2 ratio. This ratio suffers from the same problems as Graham's ratio and also interference from any CO_2 that is present in the seam gas.

There are many other indicators that have been suggested but none of them offer anything significant to the above, as they suffer from the same problems (Cliff, Hester and Bofinger, 1999) and may only serve to confuse the diagnosis. Some indicators such as CH_4 to CO_2 can be used to help identify anomalies and locate leaking seals or abnormal ventilation circuits in goafs.

Care needs to be taken when using inertisation techniques as they upset the parameters on which a number of computer programs calculate ratios as they use preset factors for such things as the ratio of oxygen to nitrogen in inlet air. This in turn will invalidate oxygen deficiency calculations and distort Graham's ratio, artificially reducing it. Equally importantly introducing additional flows into goafs can cause existing flow paths to alter and direct different gas atmospheres to monitoring locations with consequent effects on the interpretation.

Trigger action response plans (TARPS) for spontaneous combustion should be established to initiate when something abnormal is detected, hopefully indicating a precursor to advanced oxidation rather than actual advanced oxidation. This allows preventive action to be initiated without impacting on the production of the mine. TARPS should not simply be for evacuation or major concern, they should initially be advisory and necessitate action by perhaps just the ventilation officer and his support crew. The response to a trigger should be appropriate to the risk the trigger reflects. Why evacuate the mine when CO exceeds 100 ppm in the goaf? Is there a flammable atmosphere there? What is the source of the CO – is it extensive oxidation or intensive? These are questions that would modify the response to the trigger. In this modern era there is no need to use simple triggers relying on the measurement of one gas. Mine environment monitoring systems are capable of providing a lot of information and it should be utilised to assist in the decision making process.

CONCLUSION

In summary, prevention is better than cure, especially where there is no guarantee that a heating can be detected at a stage early enough to control it quickly and easily.

Comprehensive monitoring systems need to be established to establish normal mine environment behaviour and understand the factors that can affect gas concentrations in all areas of the mine, including monitoring pressure differences around the mine, air flows and temperatures. Proper maintenance and personnel skilled in understanding mine monitoring systems and interpretation of mine atmospheres must support these systems.

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