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Coal Mine Outburst Mechanism, Thresholds and Prediction Techniques

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August 2006

ACARP

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COAL MINE OUTBURST MECHANISMS, THRESHOLDS AND PREDICTION TECHNIQUES

AUGUST 2006

CLIENT: AUSTRALIAN COAL ASSOCIATION RESEARCH PROGRAM (ACARP)

FOREWORD

This report presents case studies of outbursts both in Australia and overseas.

In an endeavour to provide a basis of prediction as to whether an outburst will occur it takes the approach of examining the total energy that may be released in an outburst. The sources of energy considered are the strain energy that may be released in failure of the coal and in the release of gas. Two modes of gas release are considered, one from pore space and one from diffusion. In the latter case a new model is developed to describe the potential energy release from diffusing particles. In the Australian context it is considered that the elements of energy release due to gas dominate.

The critical factors that contributing to energy release in an outburst are:

- Gas Content/Gas Pressure
- Diffusion Coefficient
- Sorption Isotherm
- Particle Size

The less critical factors are:

- Free pore space
- Stress
- Stiffness

A determination of the total potential energy that may be released from an outburst is considered to be a better method of determining the outburst risk than current single parameter methods in use. It is proposed that this should be developed to replace current systems.

Such a development is likely to enable mining in gassy but hard coals but may impose different restrictions on mining fine gouge material.

Ian Gray

3 August 2006

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1 Introduction

This report has been prepared as a document which describes outbursts and their mechanism in an environment where most Australian Coal miners have never seen an outburst. Its first function is hopefully therefore to educate. Its second function is to suggest ways by which outburst conditions may be predicted and to advise on methods to prevent them occurring.

The reason that outbursts are extremely rare in Australia is the success that the mining industry has had in preventing them. Gas drainage and a conservative protocol has virtually eliminated the occurrence of outbursts. However the cost of this is sometimes quite great in as far as areas that cannot be drained cannot be mined, or can only be mined by shotfiring. These limitations have severely slowed or halted development in some cases. Hence an attempt is made in this report to put forward methods to assist in determining whether outbursts can occur under varying conditions.

2 Description of an Outburst

An outburst is a violent expulsion of coal and gas from a working face. Sometimes rock is also dislodged in the outburst.

Outbursts are hazardous through the mechanical effects of particle ejection and by asphyxiation from the gas produced. The violence of an outburst has frequently moved a continuous miner back several metres from the face in a heading. In a number of cases the dislodged coal, which is frequently of small particle size, engulfs the operators preventing them from escaping while the gas released asphyxiates them.

Outbursts normally only occur on advancing headings though there have been cases where they have occurred from a longwall face even though they may have been otherwise described. To my knowledge, no case of outbursting has ever occurred on pillar recovery.

There have been reported a number of cases where outbursts have occurred subsequent to shotfiring. Gray (1980, I) reports cases at Akabira and Sorachi Mines in Hokkaido, Japan where crews were caught by outbursts occurring after the shot was fired. An outburst at Sorachi occurred four hours after the shot was fired catching the crew unaware.

3 Case Studies

Australian coal mines have suffered over one thousand outbursts. In every outburst case the details are slightly different and engineers and geologists have pondered the question as to why the outburst occurred. Studies have involved detailed geological mapping, examination of coal microstructure, and physical tests. Nearly always there is something that is different locally, whether this local difference is the cause of the outburst is frequently not known, though in many cases a significant geological feature is associated with the outburst site.

Outbursts in the Bulli seam at West Cliff (Marshall et al, 1980) and Tahmoor Collieries are prime examples of outbursts that have occurred on structures in NSW. In Queensland, outbursts on structures have occurred at Leichhardt Colliery, Blackwater (Moore and Hanes, 1980) and the Collinsville mines (Biggam, et al, 1980). Notable exceptions from those outbursts that have occurred on geological structures are the outbursts that have occurred from solid coal at Leichhardt (some 350 outbursts) and Cook Collieries (Gray, 1980 II), (Moore and Hanes, 1980).

Overseas many outbursts have occurred some of which vastly exceed the size of any that have occurred in Australia. Hargraves (1980) reports an outburst at No 1 Morrissey Colliery in Canada which occurred in 1904 and produced 3 500 tonnes of coal and 140 000 cubic metres of gas.

3.1 The West Cliff Experience

Marshall et al (1980) describe very well the nature of outbursts at West Cliff Colliery. The mine was operating at the time in the Bulli seam at a depth of 480 m with a seam gas pressure of greater than 3 MPa. The seam is 2.5 m thick and consists of a bituminous coking coal with a vitrinite reflectance of about Ro max = 1.25. The seam gas was of a composition varying from substantially methane to substantially carbon dioxide.

The outbursts reported by Marshall invariably involve a continuous miner approaching a shear zone on a fault. A particular shear zone direction (110 deg) produced far more outbursts than another (080 deg). The shear zones had thicknesses of 200 to 1400 mm and generally but not universally the thicker the zone the more severe the outburst. Severe outbursts ejected up to 140 tonnes. Parallel to the shear zone were usually a series of joints that might be up to 2 to 3 metres from the zone. This jointed coal was usually dislodged with the outburst.

On approaching the shear zone a haze, thought to be due to cooling with gas expansion would sometimes be visible from the continuous miner and then an outburst would occur. Under work practice at the time the operator would use the continuous miner as a shield. It was found to be particularly effective to have the head raised to contain the outburst. This practise was also in place at Tahmoor Colliery.

Figures 1 to 6 are reproduced directly from the paper by Marshall et al. They give a good indication of the occurrence of outbursts at West Cliff Colliery.

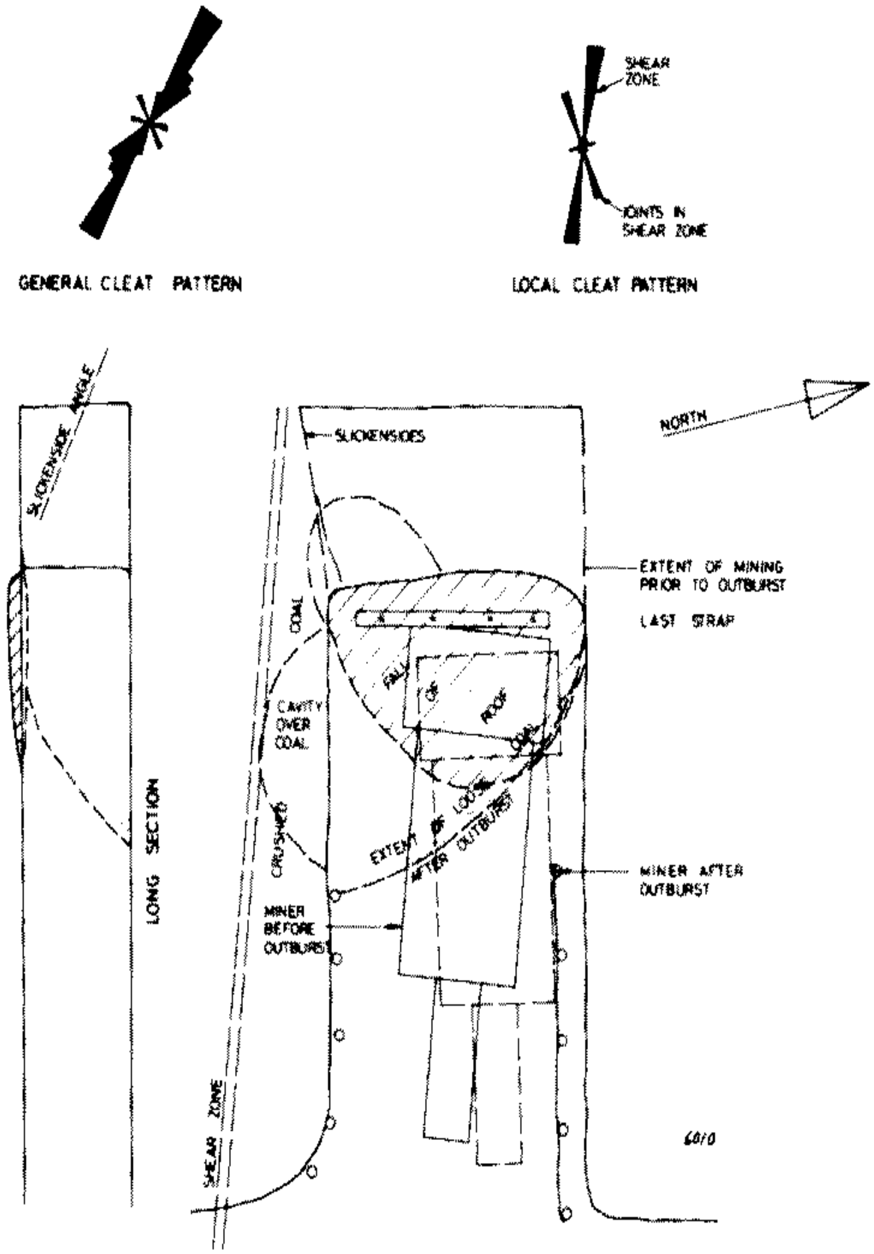


Figure 1: Outburst No. 3 West Cliff Colliery,Bulli Seam. Drivage with shear zone parallel to roadway. (Marshall et al 1980|)

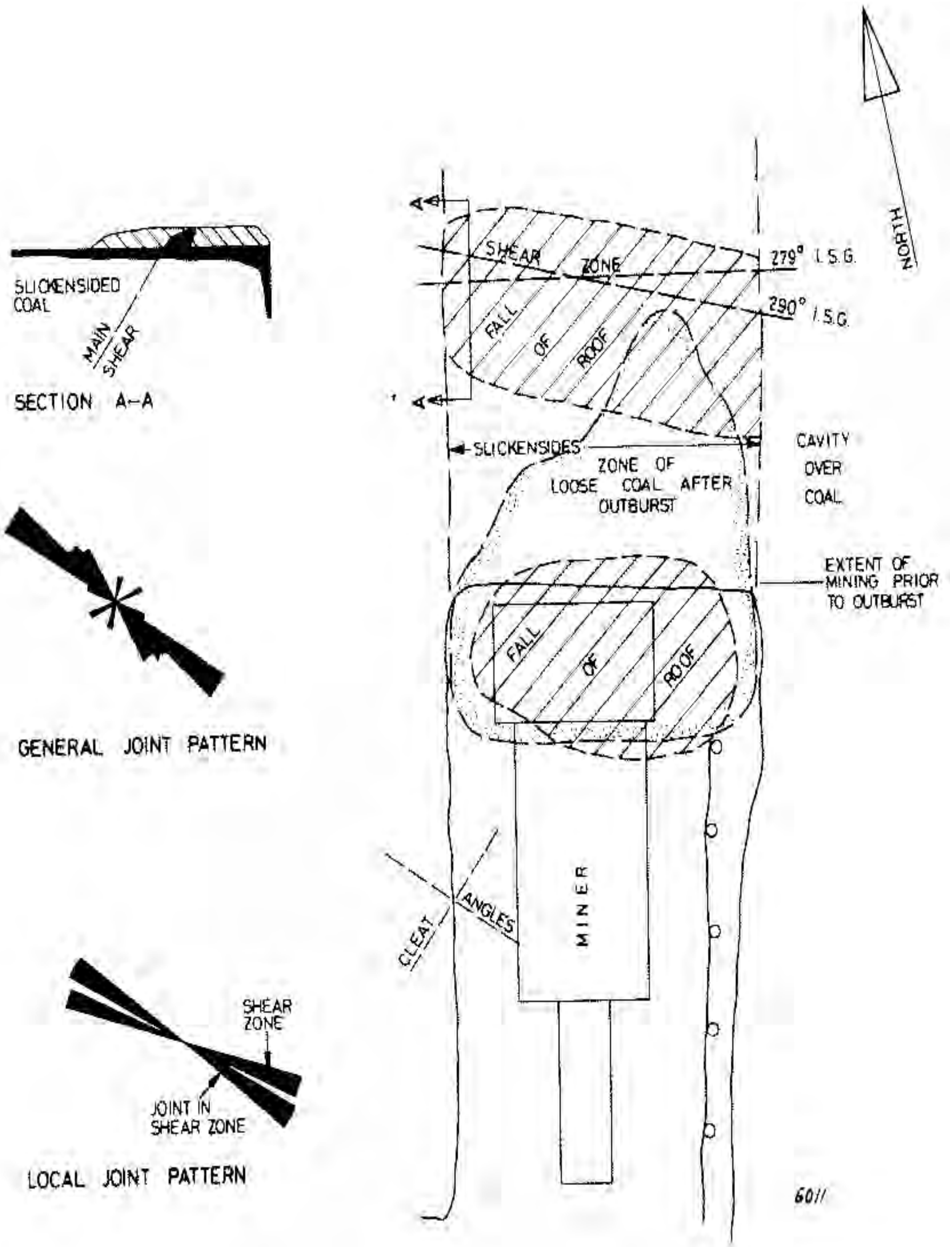
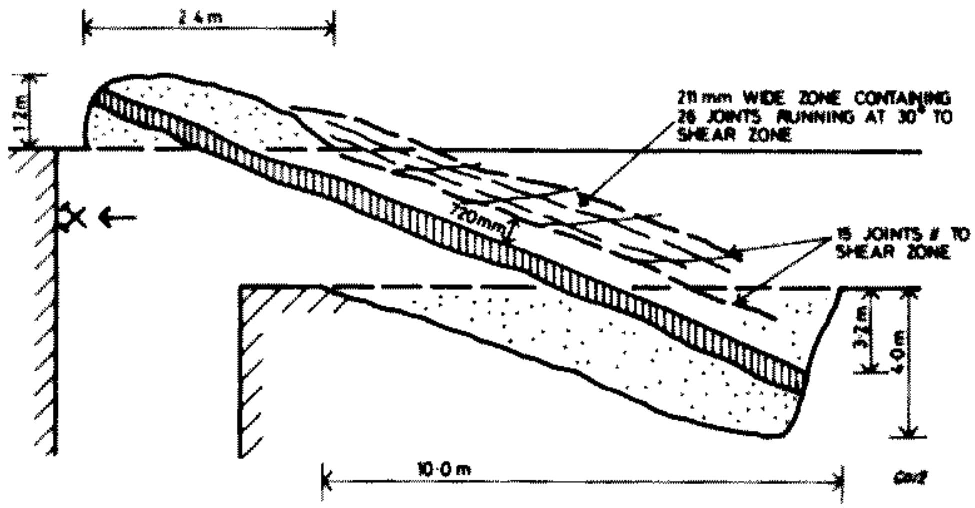


Figure 2: Outburst No. 1 West Cliff Colliery, Bulli Seam. Drivage with shear zone intersecting at right angles to the roadway.; (Marshall et al 1980)



<u>Figure 3: An example of outburst and associated failures. Outburst No. 85.; (Marshall et al 1980)</u>

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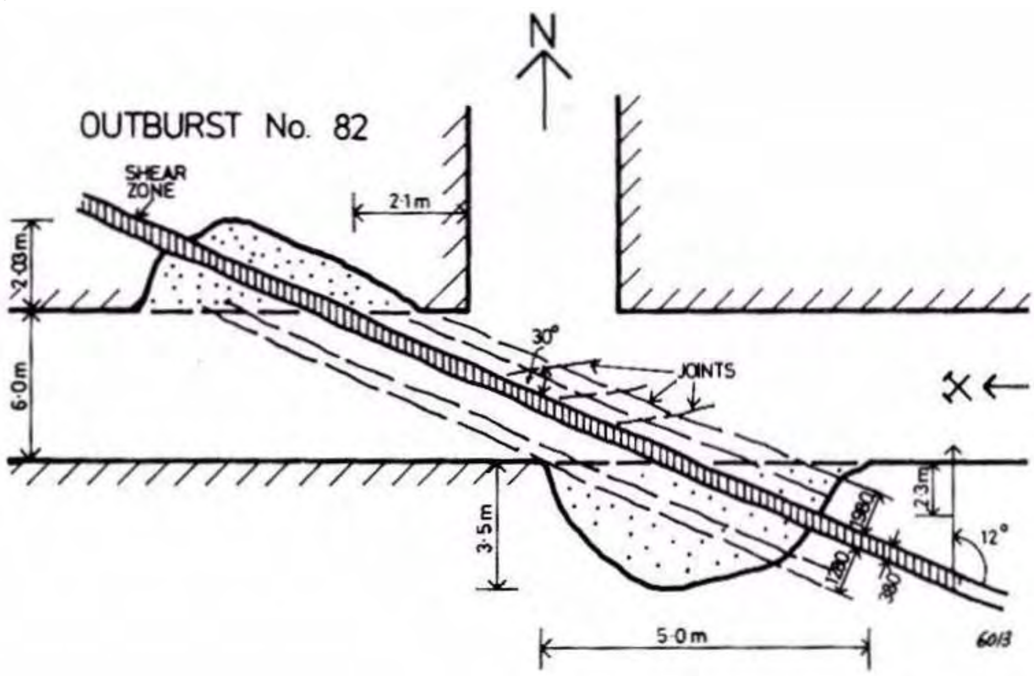


<u>Figure 4: An example of outburst and associated failures. Outburst No. 82.; (Marshall et al 1980)</u>

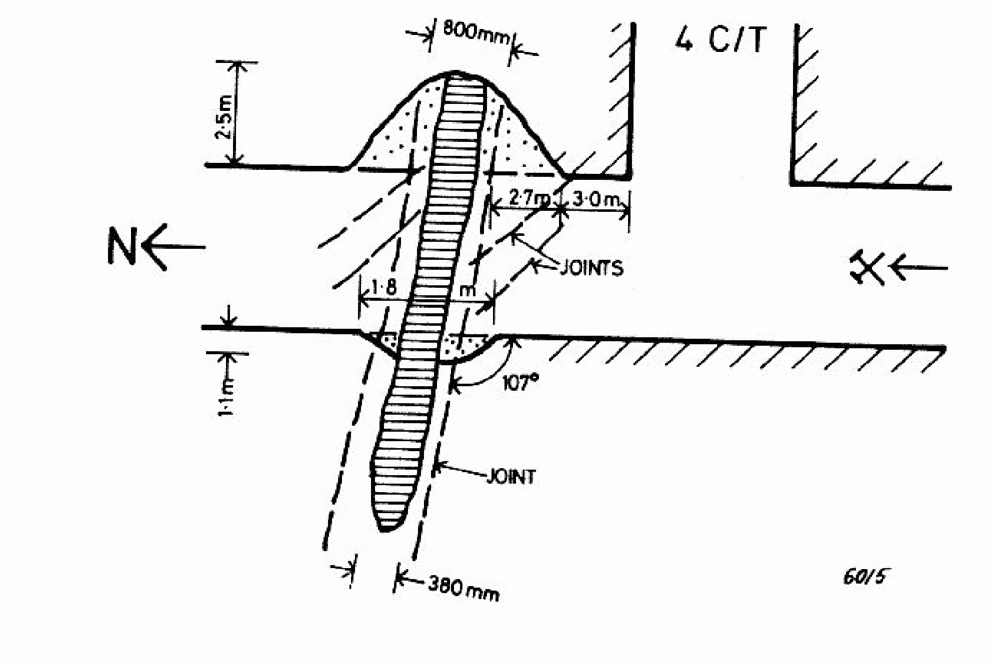


Figure 5: An example of outburst and associated failures. Outburst No. 26. (Marshall et al 1980)

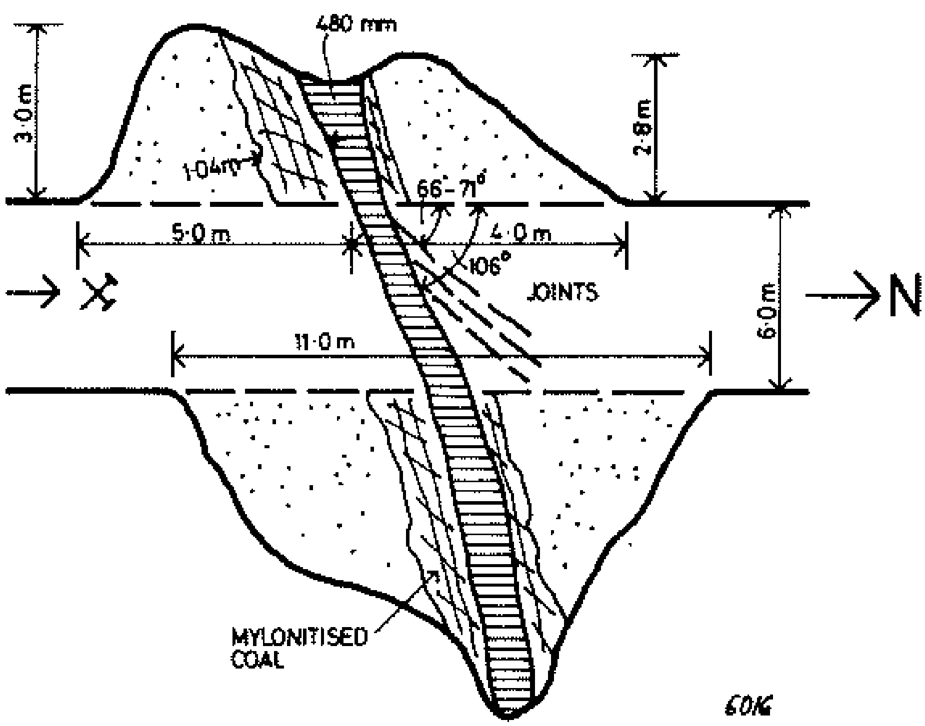


Figure 6: An example of outburst and associated failures. Outburst No. 20.; (Marshall et al 1980)

Outbursts effectively ceased at the mine with the introduction of guided drilling to ensure good gas drainage practice. However the mine had two outbursts from the longwall face in 1998, which were reported by Walsh (1999). These formed cones that pushed approximately 17 tonnes of the upper part of the seam into the AFC and left behind it the appearance of burst cones in the face. The outburst occurred when the longwall was taken beyond a gas-drained area which contained CO_2 at a gas content of approximately 20 m^3/tonne. Such longwall burst occurrences are however rare though the Author is aware of their occurrence in the Kuzbas coal area of Siberia.

3.2 Leichhardt Colliery – Solid Coal Outbursts

Leichhardt Colliery was located in the Bowen Basin some 16 miles south of Blackwater. It mined the 6 metre thick Gemini seam at 400 m depth. The seam consists of a low ash (6%) high quality coking coal of vitrinite reflectance of 1.24% consisting roughly of equal parts vitrinite and inertinite. Uniaxial compressive strength testing revealed a core strength between 5 and 20 MPa with Young's modulus of 2.3 GPa and a Poisson's Ratio of 0.32 when approaching sample failure.

The coal was noted for its high frequency and very directional main cleat system which varied from 040 to 100 degrees in orientation. A very minor butt cleat existed which was mineralised. The coal had a gas content of some 16 m^3/tonne. On mining, many induced fractures formed with a spacing as close as 1mm which frequently enhanced and/or joined the cleating and made identification between mining induced and natural cleat fractures quite difficult (Hanes and Shepherd, 1981).

Leichhardt Colliery had an unusual stress and permeability regime. The coal exhibited a very and directional marked cleating. The principal stress direction appeared to follow the direction of the main cleat with a 2 to 3:1 horizontal stress ratio (as measured in the sandstone roof). In drained areas of the seam the vertical stress was found to be higher than the horizontal. This stress regime is thought to be significantly different from the pre drained state. Gas drainage holes drilled across the cleat produced very little gas at first but would after two months produce a significant quantity. Holes drilled parallel to the cleat would only produce gas at occasional joints. The ratio of permeability perpendicular to the cleat compared to parallel to the cleat is considered to be of the order of 100:1. Initial permeability across the cleat was by back analysis of drainage thought to start at about 0.1 millidarcy and to rise with drainage to something approaching 500 millidarcys. This extraordinary change in the permeability is considered to be a function of the close nature of the cleating and shrinkage that occurred in the coal with drainage.

Most of the mining was conducted by a Joy 10CM continuous miner, with a few pillars developed by an Alpine AM50 road header. A limited amount of development was conducted by shotfiring in the eastern development and after the fatal 1978 outburst, in the 1 South Development. The mine never progressed beyond this stage of development because of numerous problems. There were many outbursts, roof (coal) bursts, floor heave and ribside failures. Rectangular roadways had to give way to arched roadways and then with shotfiring there was a return to the rectangular roadway configuration.

The outbursts invariably occurred from the face on the side that was perpendicular to the main cleat. The mode of failure was one whereby part of the face would often show a bulged or ringed structure akin to a cut onion (face). As the face bulged further, the outburst would occur. The mining crews became able to pick such outbursts by the ring formation and would often try to induce them with the cutting head of the miner. The outbursts were usually quite small by Leichhardt standards dislodging less than 75 tonnes of material. These small outbursts led to complacency. The largest outburst in the eastern workings occurred on 18 July 1975. This outburst came from the right hand corner of the face and engulfed the miner, its driver and cable hand, who fortunately survived. Over 300 tonnes of material was loaded out in the subsequent clean out. A plan of the outburst is taken from Moore and Hanes (1980) and is shown in Figure 7.

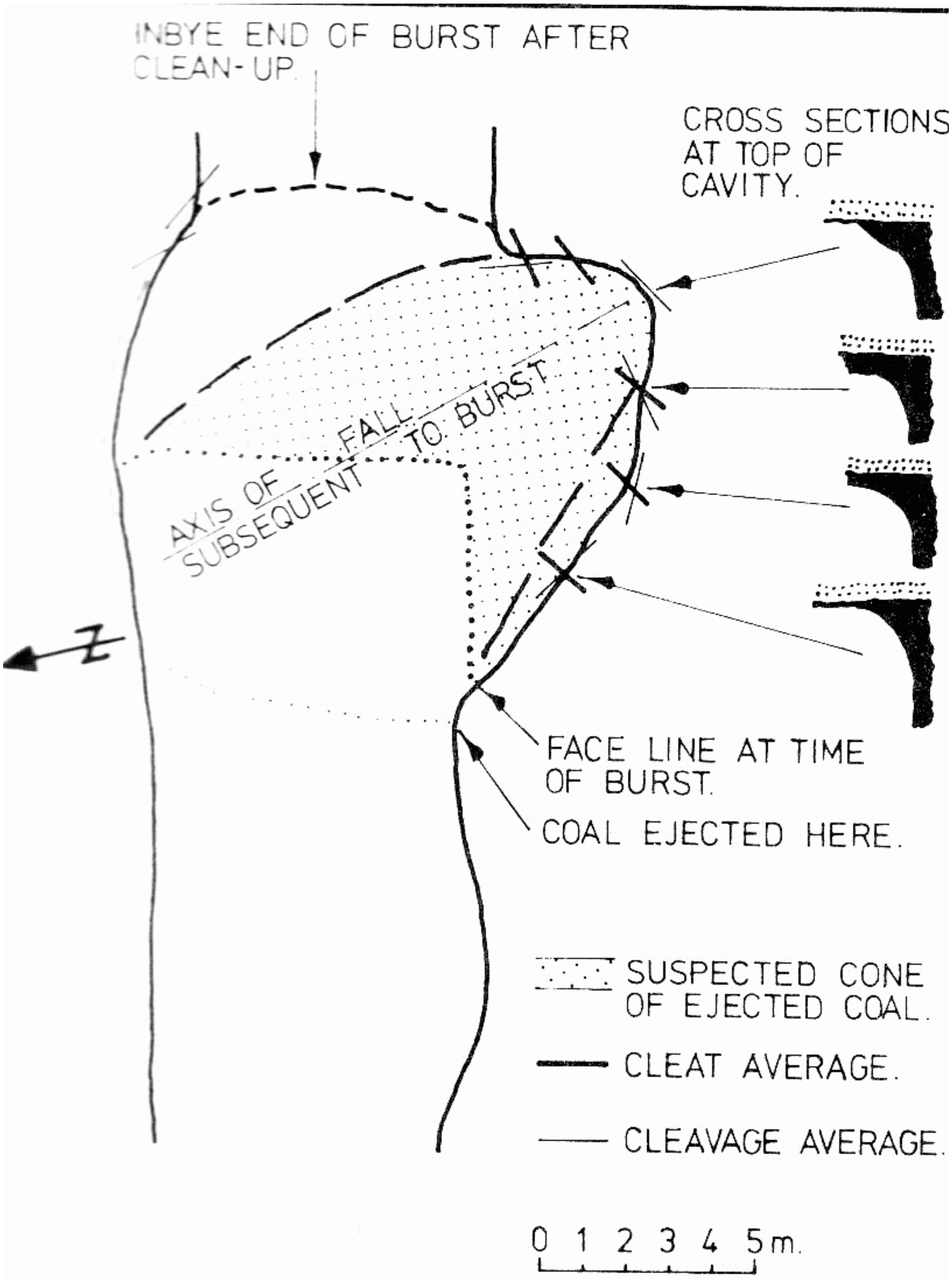


Figure 7: Leichhardt Colliery Eastern Outburst , Moore and Hanes (1980)

These solid coal outbursts were of a conical form and occurred with the axis of the cone approximately perpendicular to the main cleat. My interpretation of these is that failure would occur within the coal causing multiple induced cleavage planes. The coal between each of these planes would start to buckle and gas pressure would drive the plates of coal outward. It was possible to count hundreds of induced cleavage planes in the walls of many of the outburst cavities and in the ribs. Figure 8 shows a section of such an outburst cavity.

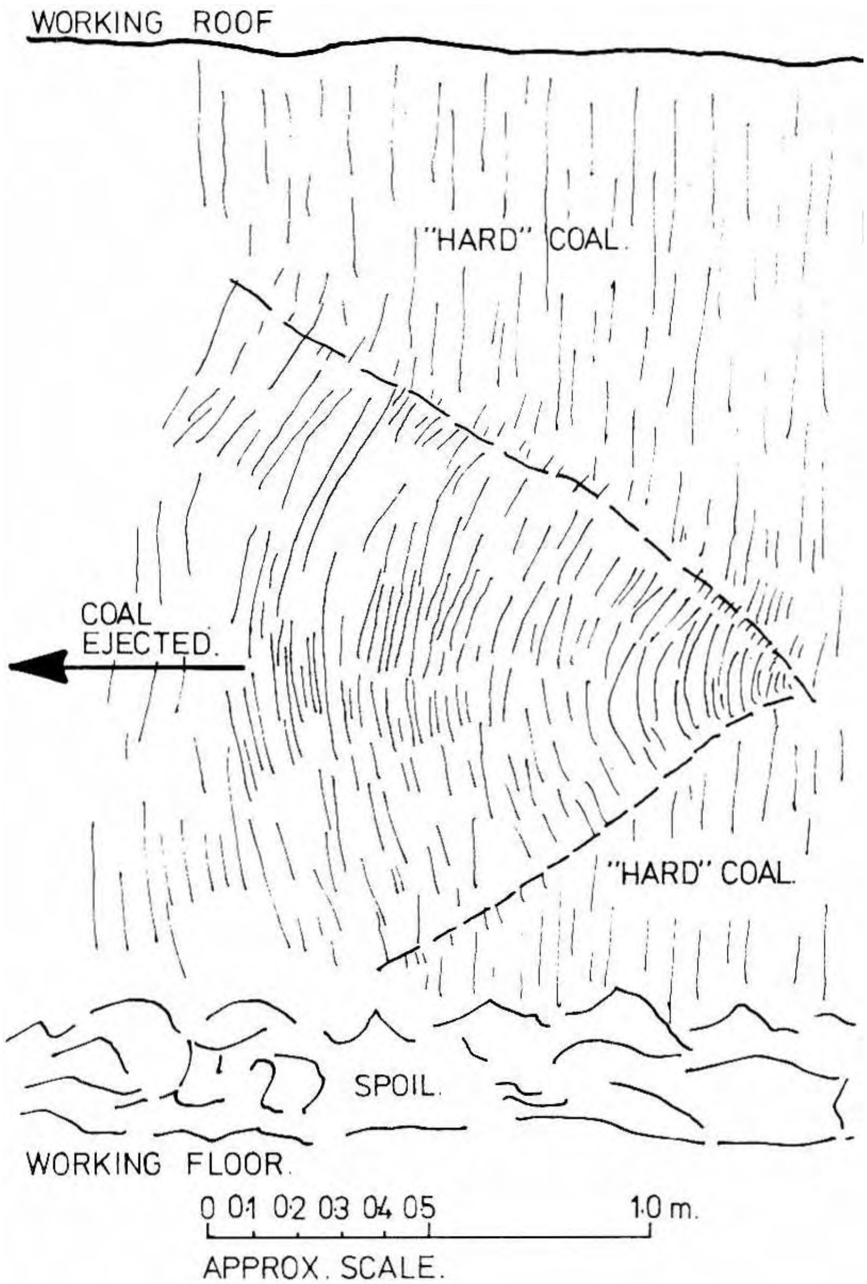


Figure 8: Leichhardt Colliery Outburst Cavity – Induced Cleavage Planes, Moore and Hanes (1980)

After well over 200 outbursts while using mechanical mining equipment, mechanical mining was banned as a consequence of the 1978 fatality and the mine reverted to shotfiring in the southern area. This was apparently successful; however measurements revealed that this success was simply due to natural gas drainage having occurred along the cleat between the previously existing southern and eastern areas of the mine. As mining progressed beyond the area drained between the headings and down the cleat line, gas pressures rose dramatically and normal solid coal outbursts recurred. This situation is shown in Figure 9.

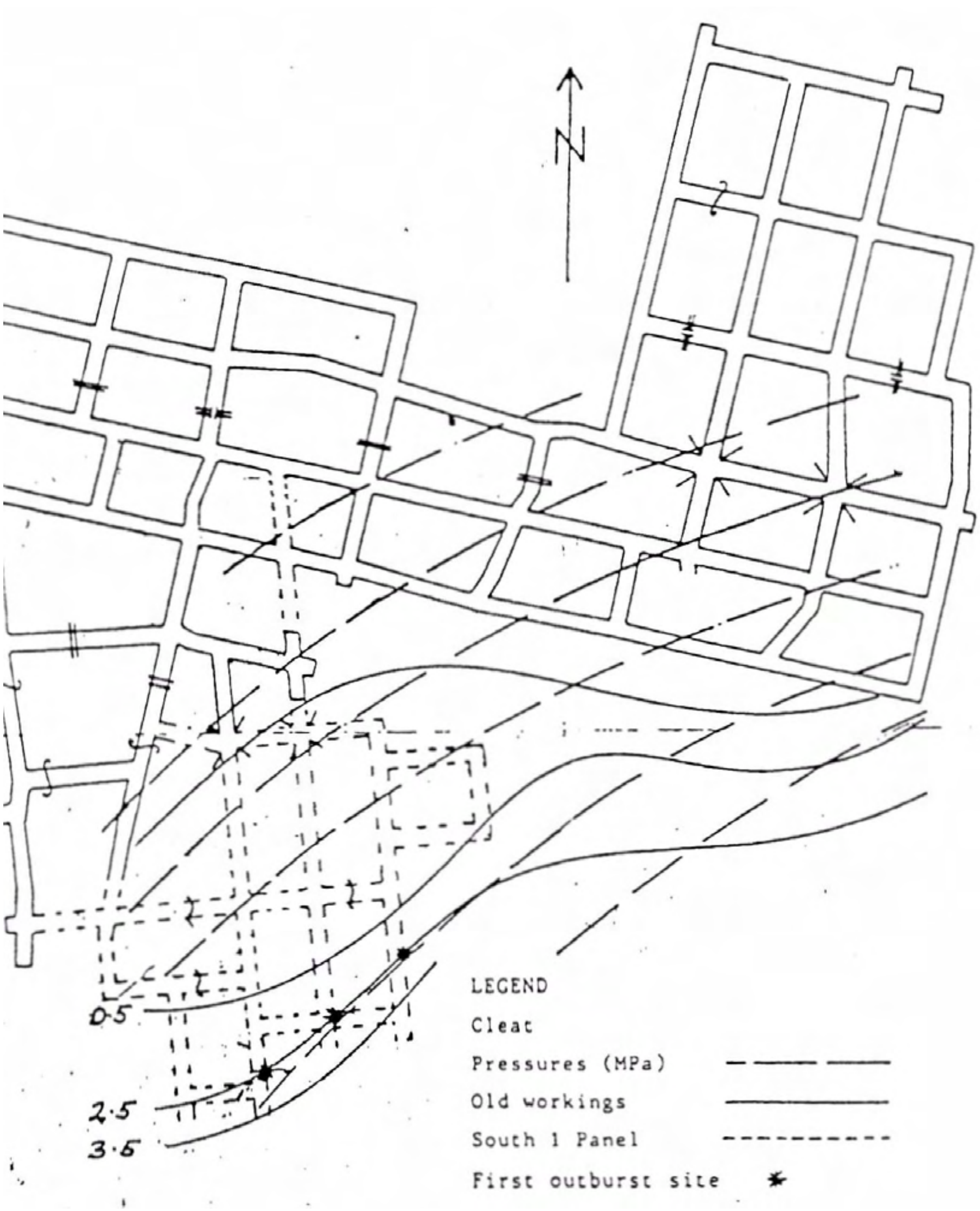


Figure 9: Leichhardt Colliery Outburst – Cleat Line Pressures, Gray I. (1983)

As can be seen, outbursts commenced on approximately the 2.5 MPa seam fluid pressure contour. This may not correspond to actual face pressure but is not too far in error as face gas pressure measurement revealed. Three holes were drilled from 1 South Panel by Jeff Wood and myself following an outburst. The procedure was to drill to a depth and then to insert a 1 metre long packer to within a metre of the bottom, set it and time pressure build up. On stabilisation of the pressure the packer was removed, the hole drilled deeper, and the process repeated. The pressures measured are shown in Figure 10. The influence of the cleat on pressure is clear as the hole drilled along the cleat had much lower pressure. This measurement reinforced the concept of 2.5 MPa being the threshold pressure for outbursts at the mine. In retrospect the operation probably put us at serious risk as we might have triggered an outburst in the process of hand drilling. At the time the risk was seen as being the same as shotfiring crews experience every round.

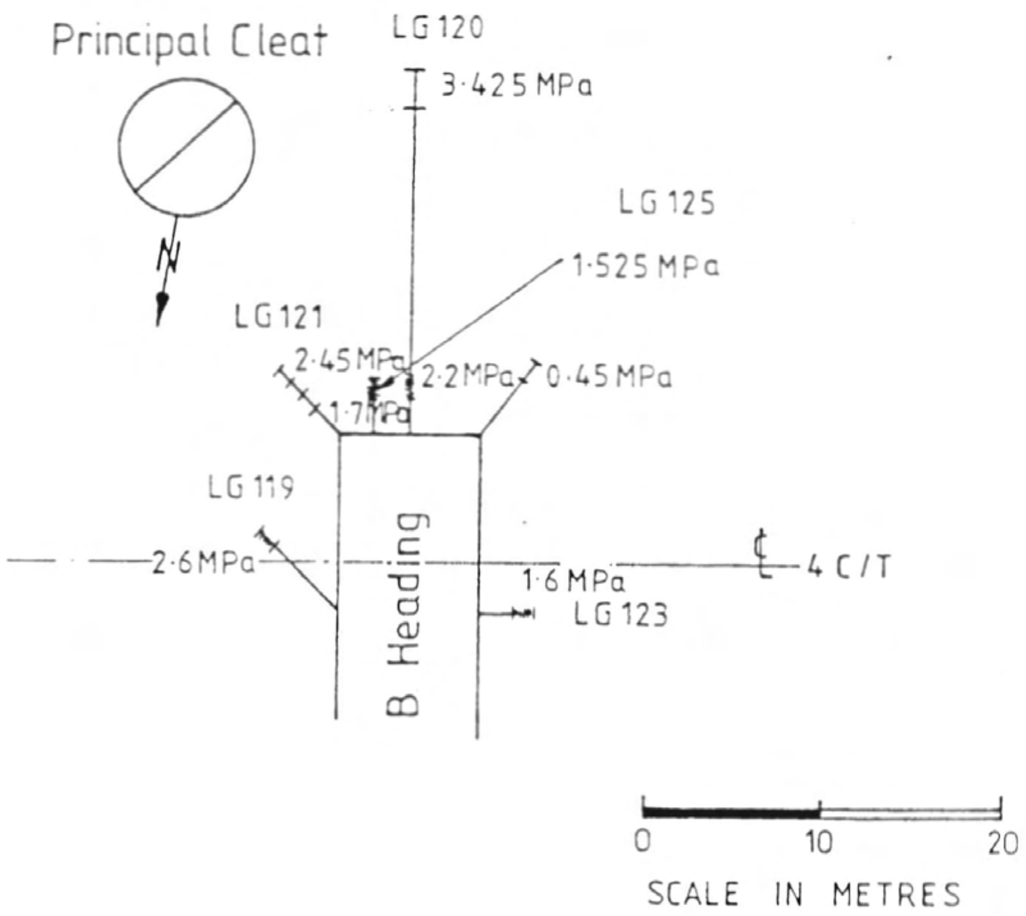


Figure 10: Leichhardt Colliery – Cleat Pressure Stabilisation, Gray (1983)

Interestingly Cook Colliery developed similar outbursts on a small scale in its deeper areas.

Several measurements were made of stress change in front of an advancing face. These were undertaken using uniaxial stress change cells. These showed no significant stress change until the cells were exposed by the next shot.

Table 1 gives the particle size distribution of broken material from a normal Leichhardt Colliery Outburst site.

Rank 1.24				
Macerals %	Vitrinite 35	Exinite 1	Inertinite 59	Mineral Matter 5
	Sizing (mm)		**% Retained**	
	+12.7		31.0	
	6.35		26.6	
	3.18		18.1	
	1.00		15.0	
	0.50		4.8	
	0.25		2.6	
	0.125		1.0	
	0.0		0.9	
Apparent relative density 1.215 gm/ml				

Table 1: Particle Size Distribution, Normal Outburst, Leichhardt Colliery, Gray (1980 II)

3.3 Leichhardt Colliery – December 1978 Outburst

In December 1978 an ouburst took place in the north of the mine. This was unlike all other outburst to have occurred at Leichhardt Colliery both in size and nature. It could be regarded as being more conventional in that it involved disturbed mylonitic coal. This disturbed coal was part of a reverse fault that had affected the seam. The fault caused approximately 30% of the seam to be either mylonitic or brecciated. A plan view and view of the ribside of the outburst cavity is shown in Figure 11. The outburst ejected 500 tonnes of rock and coal as it excavated a 21 metre long cavity ahead of the mining face. The face had been pre-drilled to approximately 20 metres with five 100 mm diameter holes the day before the outburst. A 20 tonne outburst occurred when mining proceeded directly on to these after drilling. This was cleaned up and as mining recommenced the next day the major outburst occurred. It excavated itself back into the face following the mylonitic material until it choked itself off by filling the cavity full of broken material. An estimated 10 000 to 12 000 cubic metres of methane and up to 1500 cubic metres of carbon dioxide were released with the outburst. Those surviving the outburst reported several distinct thumps as the material was ejected and entrained in the gas stream. Two men were killed in the outburst.

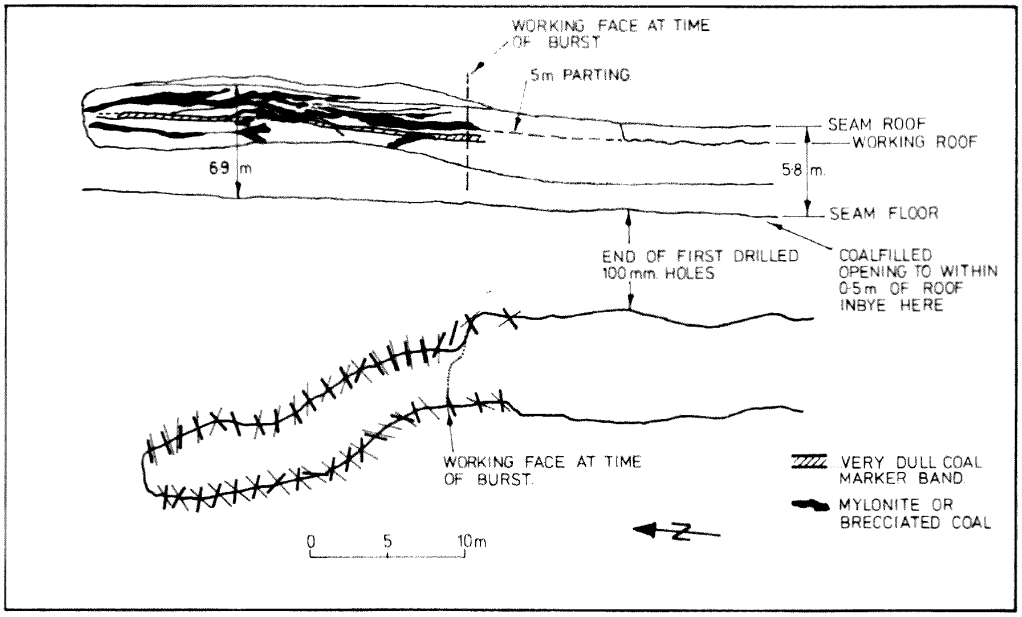


Figure 11: Leichhardt Colliery December 1978 Outburst Cavity, Moore and Hanes (1980)

Table 2 gives the particle size distribution of brecciated material from the December 1 outburst at Leichhardt Colliery while Table 3 gives the particle size distribution of the mylonite taken from the ribside of the outburst cavity. The nature of the brecciated material is finer than the normal outburst material (Table 1). The mylonite (Table 3) is significantly finer still.

Rank 1.23				
Macerals %	Vitrinite 51%	Exinite 0	Inertinite 43	Mineral matter 6
	Sizing (mm)		**% Retained**	
	+12.7		7.2	
	6.35		20.0	
	3.18		23.0	
	1.00		25.4	
	0.50		10.8	
	0.25		6.6	
	0.125		3.1	
	0.0		3.9	
Apparent relative density 1.24 gm/ml				

Table 2: Particle size distribution of brecciated material from December 1978 outburst. Gray (1980 II)

Rank 1.28				
Macerals %	Vitrinite 56%	Exinite 1	Inertinite 38	Mineral matter 5
	Sizing (mm)		**% Retained**	
	+12.7		12.3	
	6.35		14.2	
	3.18		16.2	
	1.00		21.2	
	0.50		11.8	
	0.25		10.8	
	0.125		6.1	
	0.0		8.0	
Apparent relative density 1.124 gm/ml				

Table 3: Particle size distribution of brecciated material from December 1978 outburst. Gray (1980 II)

3.4 Collinsville Outbursts

The Collinsville mines have had an outburst problem since 1954 when an outburst in the State Mine killed 7 men by asphyxiation in an outburst of 900 tonnes. The outburst was associated with a fault and dyke. The outburst set a pattern for less severe ones that followed in Dacon No. 3 Mine and No. 2 Mine in that it involved faulting and igneous activity and appeared to be largely gas-driven by carbon dioxide seam gas.

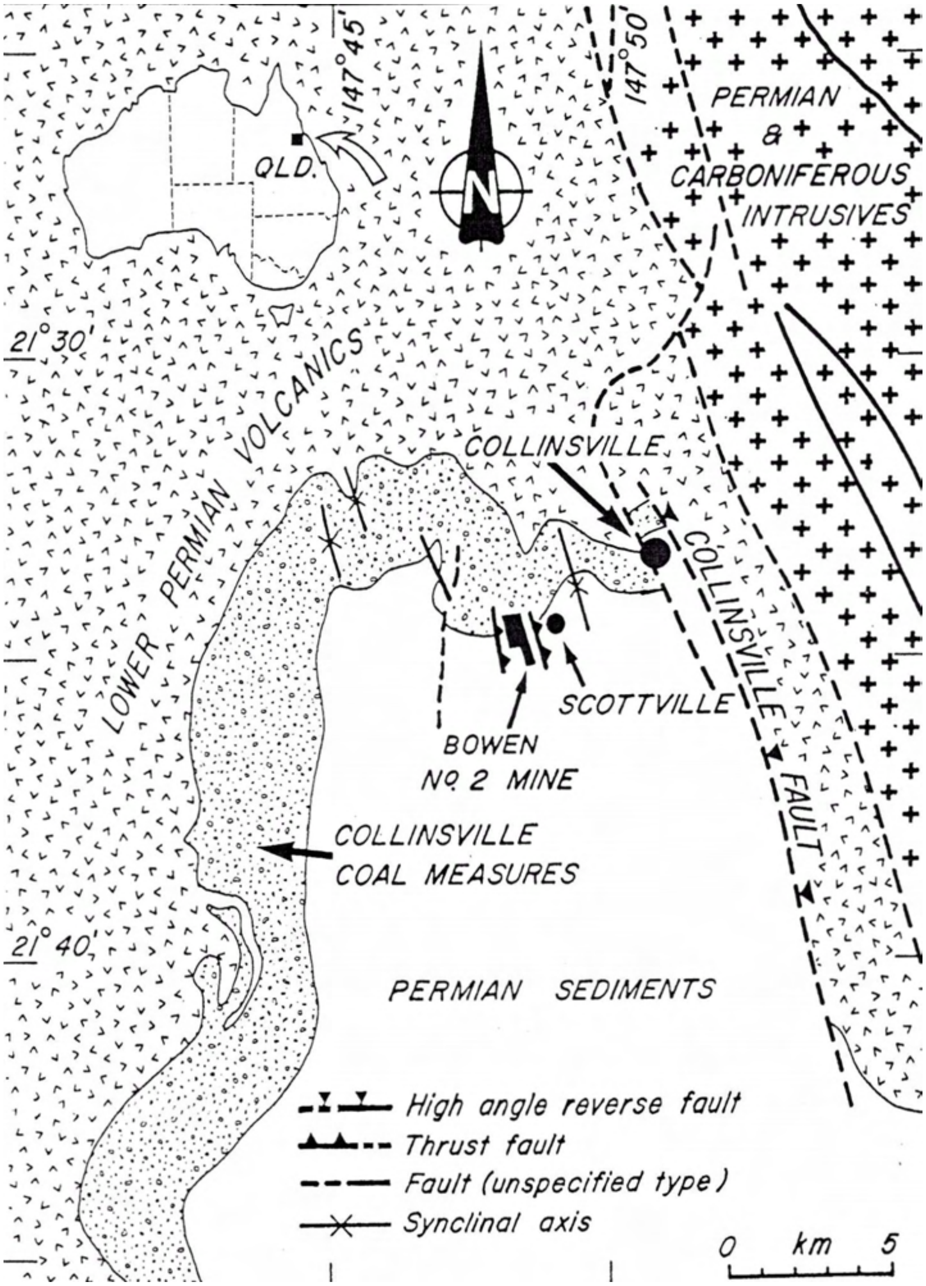


Figure 12: Geological Setting of Bowen No. 2 Mine, Shepherd et al (1980)

3.4.1 Collinsville Geology

Collinsville is situated at the northern end of the Bowen Basin and dips from the outcrop, southward. There are a series of seams in the Collinsville coal measures. The geological setting is shown in Figure 12. The Bowen seam is mined at No. 2 mine. It is reasonably well separated from the other commercial seams, the Blake and Garrick. The seam thickness varies but is approximately 6 metres. Towards the southern (lower) dips area of the mine, a stone band divides the seam. The composition of this varies but a carbonaceous muddy siltstone of approximately 0.5 to 1.0 metre in thickness was found at the outburst sites. Above this, the seam was 3.0 metres thick and below it 2.0 to 3.0 metres of coal could be expected. The seams are affected by igneous intrusions and local rank variation through the thickness. Rogis (personal communication 1980) has proposed the movement of hot fluids in the seam as an explanation for the local rank variations. These rank variations are evidenced by Ro max to varying from 1.10% to 1.31% at one location. Igneous activity is also primarily responsible for the seam gas which is predominantly carbon dioxide. Gould and Smith (1980) have evidence of this from their isotopic study of the coal. The seam gas composition varies with methane making up most of the balance. This has presumably come from the normal coalification process.

Work done by the mine using both core desorption technique and the Hargraves' emission index has shown that the gas content of the coal varies with location. Depth has a bearing on this but local variations also exist. These local variations are thought to be due to coal type changes and igneous intrusions. Cross bedding is frequently found in the coal. This leads to rapid coal variation with position.

In the mine area there are a number of geological faults. Running approximately North-South are several major reverse faults combined with other lesser faults. These faults have been the main site of outbursts in No. 2 mine. The outbursts have been characterized by ejections of mylonite. Floor heave occurred during mining in the dips area and this was considered to be a gas-related problem. Faults and outburst sites are shown in Figure 13.

In the upper areas of the mine the roof is massive sandstone but tends to become more shaley with depth.

A dominant cleat exists in an approximately east-west direction. The butt cleat is less well developed but is more obvious than its equivalent at Leichhardt Colliery.

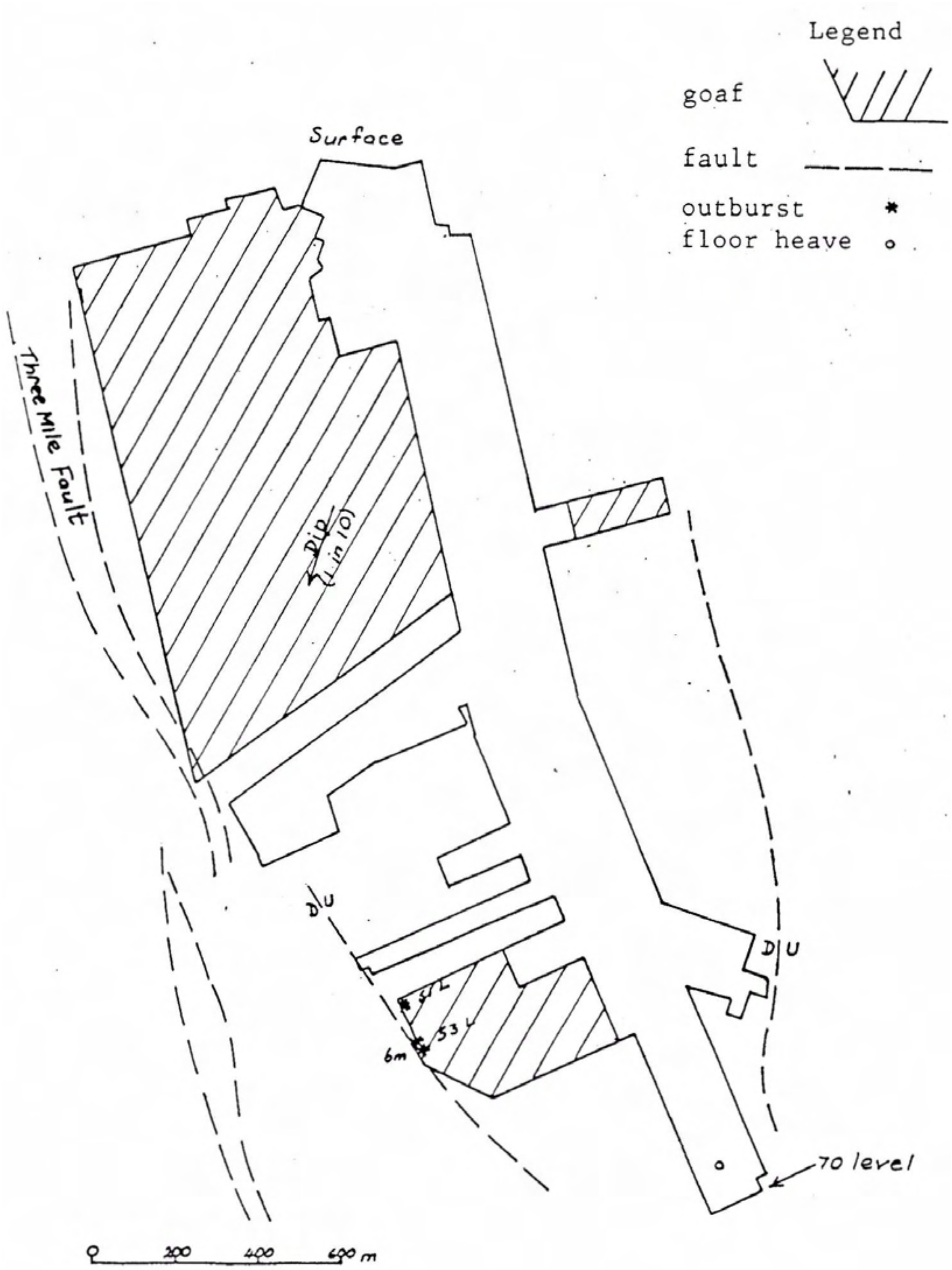


Figure 13: Bowen No. 2 Mine, Collinsville Faults, Outburst and Floor Heave Sites. Gray (1983)

3.4.1 Collinsville Mining

Mining into the early outburst in the State Mine was being undertaken by shot firing. When mining resumed in the area of the 1954 outburst, a procedure of inducer shot firing was adopted. This involved firing either simultaneous or millisecond delay rounds with men being well clear of the working area. This approach was successful in inducing outbursts (Hargraves 1963).

In No. 2 Mine mining has been with a continuous miner. Some attempts were made in the dips area to use pulsed infuser shotfiring with ICI Hydrobell explosive and packers as stemming following practice at Metropolitan Colliery (Ward 1980). This involved using a borehole filled with waterproof explosive and water under pressure. This was found to be unsatisfactory as misfires often occurred, and hence was abandoned. The bulk of mining in gassy areas had been controlled by the Hargraves' emission value. When this exceeded 1 cm^3 gas/gm of sample coal, mining was halted, the face and floor drilled up to 40 metres in depth and emission values retested until they dropped below 1 cm^3/gm. Mining could then proceed.

3.4.3 Outbursts at No. 2 Mine

Several outbursts and a floor heave/outburst from beneath the stone band have occurred in the mine. These have been small in comparison with outbursts in the State Mine. The outbursts occurred on a thrust fault of 6 metres throw as shown in figure 4.8. This fault is a double thrust set and is surrounded by a large amount of mylonite and brecciated coal. Shepherd et al (1980) report increased cleat intensity up to 200 metres outbye of the fault. The outbursts occurring in 53 Level West (53LW) were all gassy, of fine material and small size. The first in 53½L followed pulsed infusion shotfiring and a recorded Hargraves' emission value of 1.1cm^3/gm. It took the form of a virtually silent excavation into soft mylonitic or brecciated coal. The best description that can be given to the outburst is one of similarity to a piping failure in an earth dam or embankment. The gas clearly eroded a path back into the coal until the outburst choked itself off behind the head of the miner. This is shown in Figure 14. The outburst produced 25 tonnes of coal.

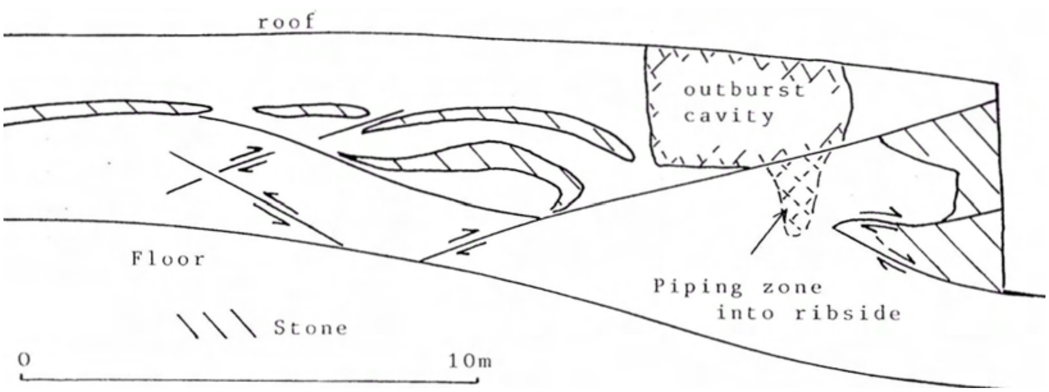


Figure 14: Sketch of 53 ½ Level outburst in the six metre throw thrust fault. View of southern ribside. Gray (1983)

Another outburst occurred in the 53L area of the mine. This could almost be regarded as slump as a soft boggy face rolled out and over the head of the miner.
Later in the next panel above 53L, 51L, several small outbursts occurred. The last of these moved about 30 tonnes of material. The outburst itself resembled a blast from a shotcrete gun of fine particles. Subsequently the roof area above the outburst collapsed. As a small gas drainage exercise had been carried out in the vicinity of the outburst, the gas pressure was known to be 400kPa or less. A floor heave during the drivage of 67L in the mine involved a release of gas and the breakage of the stone band. The miner was raised and pushed sideways. Twelve metre long floor holes which penetrated the stone band were drilled prior to the floor heave occurring. Adjacent to the floor heave site was a substantial remnant of a river channel in the roof.

3.5 Japanese Outburst Experiences

In 1980 I had the opportunity to visit Japan on a 3 month study visit arranged between the Japanese Ministry of International Trade and Industry (MITI) and the Commonwealth Government. The purpose of that visit was to study Japanese methane drainage practice. Outburst occurrences in Japan were also documented (Gray I., 1980 I).

3.5.1 Sunagawa Colliery, Hokkaido

This mine suffered from severe outbursts for many years before the drainage program described in section 3.2 was adopted. Even with the implementation of this program, workers at the University of Hokkaido have monitored small (60 tonne) outbursts.
Because of the weak coal (1-3 MPa UCS) and high gas pressures measured to 4.4 MPa at 710 metres depth, outbursts occurred readily. The most common outburst location was on crosscut drivage from rock into coal. The seam most commonly affected was No. 8. Outbursts could extend up to 20 metres above the crosscut.

Drainage practice in use in 1980 probably removed about 40% of seam gas and caused a drop in seam pressure to approximately 2 MPa (Gray 1980 I). Use of drivage by inducer shotfiring appeared to control any present outbursts safely.

3.5.2 Akabira Colliery, Hokkaido

Geology

The colliery is in the Northern part of the Ishikari Coalfield. Four coal bearing measures with 23 coal seams of 50 metres thickness in total exist in the lower part of the Tertiary strata which mainly consist of shale and sandstone and overlie Cretaceous strata in unconformity. The Akabira Fault is the major geological structure in this area and the general strike shows N-S direction. The inclinations of the east and west wings of the syncline are about 60º and 40º respectively, and become gradually steeper towards the surface. The deepest coal seams lie at 1500 metres below sea level. Workable coal seams in 1980 were No. 11 upper, No. 11, No. 10, No. 9, No. 8 and No. 7 seams. The seams range from 1.7 to 4.5 metres thickness and inclinations of 2º to 68º.

The faulting makes mining conditions very difficult, all being located by underground drilling. Most of the faults are reverse though the Akabira fault has both normal and transcurrent components. Most of the faults have been normal in the past but have been changed. This is revealed by tuff marker bands. The major fault gouges are typically 10 to 15 metres thick and within this zone contain a mixture of rock and coal from many levels. The strata surrounding these zones are deformed though reasonably competent.

Within the fault some larger blocks of harder material remain; however, much of the material is intensively slickensided. The coal within the zones has a layered structure of multiple slickensiding and can easily be crushed by hand. (It is very similar to the mylonite found from an outburst at Leichhardt Colliery, Queensland, December 1st 1978.)

All the outbursts which have occurred at Akabira have been associated with faults. To the north-east an andesite volcanic cone exists while to the south-west a basaltic plug exists. Near the latter a neighbouring mine had to stop mining due to water make. This mine had many outbursts. No effects of the neighbouring volcanic activity are apparent to mine engineers in Akabira.

Cleating has not been studied in relation to gas drainage behaviour. In areas accessible to the author a principal cleat across the syncline and reverse faults was apparent as was a butt cleat which could possibly be associated with the transcurrent movements. Coal strengths are the highest encountered in the Ishikari Coalfield being in an “Australian Range”, i.e. 0 – 20 MPa with an average of 13 MPa. Shales are in the range 30 – 50 MPa and sandstones 80 – 90 MPa. Distinct strata of ‘Gambai’ or loose powdery coal exist within the coal seam.

Mining

Where advance mining was practised it was covered by stress relaxation drilling. This involved drilling holes of 100, 245 or 250 mm diameter ahead of the face using augers. Gas release occurred rapidly during and just following drilling because considerable breakage of coal occurs. Once in-seam drivage has been achieved mining may proceed by longwall or step-cut (Kakuchi) methods. The latter was a form of hand worked longwall in a dipping seam with a stone packed goaf.

Outbursting

All outbursts at Akabira mine have been in faulted areas, the worst cases being associated with the major faults. For this reason the major fault areas are totally avoided in extraction, and are treated with caution on advance working. Some rock outbursts have occurred in development in rock. It is common for the oily smell of higher hydrocarbons and aromatic compounds to be noticed before such occurrences. This led to gas analysis work in an endeavour to identify outbursts by this means. Some attempts have been made to measure gas pressure in cross measure work. These yielded varying pressures over a close group of holes. The holes were sealed using cement grout caulked standpipes. Akabira has concluded from this that gas pressure is not uniform and furthermore it is not a good indication of gas outbursts as outbursts occurred in the low pressure areas. The maximum gas pressure recorded was in a cross measure hole that was grout sealed for 14 metres. The borehole was used in an unsuccessful attempt at hydrofluoric acid injection to increase flow; the pressure measured at 515 metres depth was 2.55 MPa.

The worst gas outburst occurred on approaching the central fault in 1958. It released 3600 m^3 of gas and expelled 700 m^3 of rock and coal dust.

Jamming during auger drilling was regarded as an indicator of a fault and therefore an outburst-prone zone. Many outbursts occurred some time after shotfiring, catching crews erecting support. In the seven years to 1980 safety precautions incorporating large stress relaxation boreholes and stress relaxation have been adopted and there have been no outbursts.

3.5.3 Minami Oubari Colliery, Hokkaido

This colliery suffered a fatal outburst in 1979 following drivage into pre-drilled ground. The outburst was associated with two small faults of 200 mm and 300 mm movement. The normally weak coal in the mine broke and eroded easily to allow 660 m^3 of coal and 47 000 m^3 of methane to be released. The most significant finding about the outburst was that flow from advance boring was low. An explosion and fire followed the outburst during rescue attempts.

3.5.4 Yubari Shinko Colliery, Hokkaido

This outburst closely mirrors the one at Minami Oubari Colliery. It occurred in 1981 during in-seam drivage into pre-drilled ground. The boreholes were in this instance cross-measure holes through the seam from underseam galleries. Low flows were measured from these. The outburst which followed a fault involved 4 000 tonnes of coal and rock. An explosion and fire followed the outburst.

3.6 Summary of Knowledge Gained from Case Studies

A range of outbursts have been described in this Chapter. They have varied from outbursting in solid unfaulted coal at Leichhardt Colliery through the outbursts at Sunagawa where unfaulted but very weak coal was involved in the outbursts, to the other cases where ejected material was fault gouge.

In all cases gas has been implicated as the main factor in propelling the material. Of all the outburst cases, normal outbursts at Leichhardt colliery would appear to be the closest to rockburst condition as they were associated with a hard, apparently unfailed rib and face, and occurred in a buckling form which could be associated with expected face stress patterns. However, in areas measured to have low seam fluid pressure, outbursts did not occur. These areas were too far from other workings to be in a de-stressed state. The implications of this are that all outbursts are primarily gas-driven. Certainly the large outbursts rely on gas to provide clearance for the material generated so that they do not choke off on improperly expelled material.

Very large outbursts (more than 500 tonnes) all tend to be associated with faulting. The material in the faults is smaller and more easily entrained in a gas stream and more capable of supplying that gas from pore space and increased surface area to desorb gas.

Some outbursts are mentioned as having occurred at very low levels of gas pressure. The most interesting of these is the 51L outburst in Bowen No. 2 mine, Collinsville. This shows that a carbon dioxide outburst from mylonite may occur with a gas pressure lower than 400 kPa. The ease with which loose material may be propelled by gas makes it unlikely that gaseous ejections can ever be totally stopped by gas drainage alone.

4 Theoretical Aspects

4.1 Outburst Initiation

The occurrence of an outburst is preceded by failure of the coal. Failure in itself is nothing remarkable as it occurs in mining all the time. The difference in an outburst is that the failed material is ejected with energy and with gas. A gradation exists between a rockburst where no gas is emitted to outbursts which are totally gas driven. The gas contributes in a major way to the expulsion of the coal and is generally thought to be the main contributor to total energy release in the majority of outbursts (Gray, 1980 II).

Failure is by definition a state whereby the effective stress in the material (coal) exceeds the strength of the material. Effective stress is the total stress minus fluid pressure. The fluid pressure may be either gas or water in a coal seam however very little dilation of the coal mass is required for the effect of water pressure to be relieved while gas pressure will be sustained through desorption. For failure to continue the removal of material from the failure site is required otherwise the outburst will choke off as confinement is developed.

Coal is a variable material both with location and with ply within the seam. Various coals have significantly different stress strain characteristics. In uniaxial compression most coals fail in a brittle manner and break up into multiple particles. This applies particularly to the brighter coals. This breakage is a function of the coal type and rank. The particle sizes generated on breakage are important. The finer the size of particle created in the failure the more important it is in outbursting as fine particles have the capability to desorb gas more quickly and be removed from the outburst zone quickly.

Coal toughness is an issue of particular importance to solid coal outbursts. Toughness is by definition a measure of the energy required to cause a unit area increase in fracture area. Commonly this is thought of as a crack tip propagation issue and much has been written on the subject in mechanical engineering and hard rock breakage literature.

If we think of coal breakage under gas pressure then we must think of a fracture propagating whilst it is being filled with gas which is produced by the desorbing coal. The coal is of course subjected to other stresses which will also tend to cause breakage. How the coal breaks up is a function of the coal structure and its toughness. The tougher the coal the harder it is to break up. Tough coals absorb energy from the work being done to them in failure.

If coal is already broken into many fragments by the faulting process then it has negligible cohesion and can be expected to behave mechanically as a soil with the major exception that it may produce gas.

4.2 Outburst Energy Release and Fragment Propagation

The propagation of an outburst and indeed its violence may be considered to be directly related to the surplus energy that the coal has following failure. How this energy is dissipated is entirely dependent upon the geometry of the outburst and the roadway. The energy may be absorbed in inter-particle collisions or it may direct the coal outward into the roadway. Predicting these factors is probably beyond reasonable expectations as it requires a detailed knowledge of the geology and geometry of each potential outburst case. Getting some estimate of potential energy release should however be a goal that is sought after.

4.2.1 Potential Energy Release Due to Strain Energy

Stressed coal contains elastic potential energy. This stored energy may be mathematically represented as the integral of the stress with respect to strain over the volume being considered. In addition to the strain energy in the failing coal itself the surrounding material may impart energy to the failed coal from the outside.

Let us consider a potential outburst form that is a cylinder perpendicular to the face. The cylinder face is stressed radially and there is no face confinement. The mechanism of failure is crushing due to the radial stress exceeding the strength of the coal. Let us also assume that gas removes the coal from the cylinder as it fails maintaining a free face. These are sweeping assumptions but they still permit us to look at the energy involved.

It can be shown that the strain energy per unit volume of a biaxially stressed coal face is as shown in equation (1). This is simply the integral of stress and strain.

$$W_d = \frac{1}{2}\varepsilon_r \sigma_r = \frac{1}{2}\sigma_r{}^2\left(\frac{1-\nu}{E}\right) \qquad (1)$$

Where:

W_d = energy elastically stored per unit volume under biaxial states conditions
σ_r = uniform radial stress field
ν = Poisson’s ratio
E = Young’s modulus
ε_r = Radial Strain

The coal surrounding the failing cylinder may also impart energy to the core by elastic release. This energy is given in equation (2).

$$\frac{W_r}{Vol} = 2\sigma_r^{\ 2}\left(\frac{1-\nu^2}{E}\right) \quad (2)$$

Where:
W_r = Energy due to elastic wall contraction on unloading
Vol = Volume of cylindrical hole
σ_r = radial stress
ν = Poisson's ratio
E = Young's modulus

Consider now the case where the coal has a uniaxial compressive strength of 12 MPa, a Young's modulus of 2.0 GPa and Poisson's ratio of 0.3. At the limit of failure of the face the elastic potential energy contained in the core is 0.025 MJ/m^3. The energy that may be imparted from the elastically collapsing cylinder is 0.13 MJ/m^3. It is probable that these energies may not be simply added but that the failure of the coal core will absorb energy from the surrounding collapsing cylinder and that the potential energy release is somewhat less than the sum of the two energies (0.16 MJ/m3).

4.2.2 Energy Release Due to the Expansion of Free Pore Gas

Most coals are water saturated in their virgin state. However mining leads to some drainage which initially displaces water because of the relative permeability characteristics of the coal. This water displacement leaves gas filled pore space within the coal. If the coal is solid coal then the volume of the pore space is extremely low. However the potential for significant gas filled void space exists within gouge material and, to a lesser extent in coal fractured by mining stresses.

Gas stored freely in pore space contains potential energy. This energy is immediately available if the pore space is not confined within coal solids. That is the gas exists in cleats or indeed between fragments of coal in brecciated or gouge material. On failure the gas may expand adiabatically delivering its energy. If we consider this energy being delivered to a piston then we may calculate it readily.

Equation (3) describes the potential energy available from adiabatic expansion of gas.

$$W = \frac{P_1 V_1^{\gamma}}{(\gamma - 1)} V_1^{(1-\gamma)}\left(1 - \left(\frac{P_2}{P_1}\right)^{\frac{\gamma-1}{\gamma}}\right) \quad (3)$$

Where:

W = work performed
P_1 = initial pressure
P_2 = final pressure
V_1 = initial volume
V_2 = final volume
γ = ratio of specific heats $\frac{C_p}{C_v}$

If we consider a void space of 5% in a mylonite then we have an initial volume of 50 litres per cubic metre of material. Figure 15 shows the available energy from this gas during adiabatic expansion. The value of 0.33 MJ at 4.0 MPa gas pressure is quite significant and double the value likely to be available from elastic strain energy.

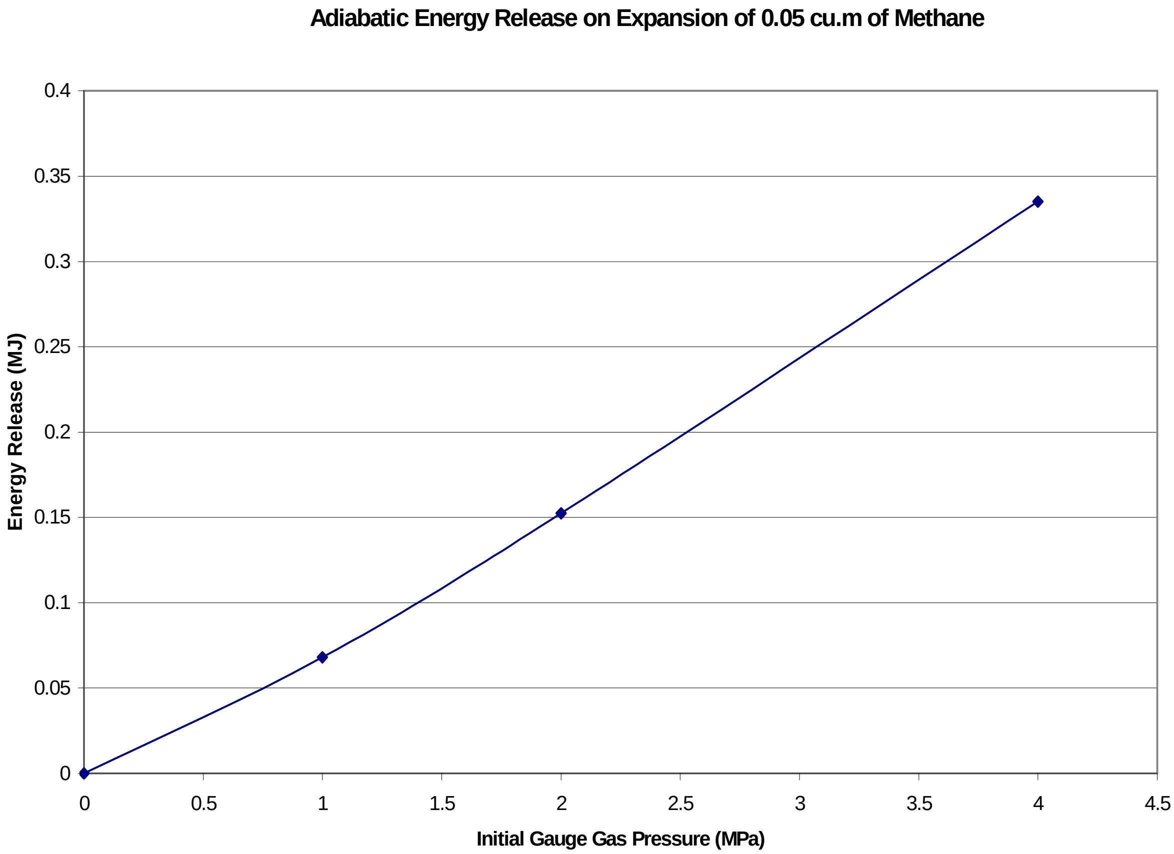


Figure 15: Energy available from 50 litres of gas expanding adiabatically to atmospheric pressure. Note 50 litres per cubic metre corresponds to a porosity of 5%.

4.2.3 Energy Release Due to the Desorption of Gas from Coal

Gas is also stored in the coal through the process of sorption. This name covers a number of processes which may be considered to be essentially summarised as mono and multilayer adsorption. This adsorption takes place in micropores within the coal and a process of capillary condensation occurs whereby the gas is actually in liquid form within the pores. Interestingly the pores may expand with the gas contained therein (Gentle, personal communication 2006). The effect of different pore sizes is to cause variations in gas diffusion rate.

The physical effect of this is coal shrinkage during desorption which may have dramatic effects on permeability of coals (Gray, 1983, 1987) and which is physically measured by SIGRA Pty Ltd on core samples.

Diffusion is the process of the movement of a chemical through another substance. The case being considered here is the movement of gas through coal.

Crank (1975) produced a work which mathematically treated the movement of varying substances down a concentration gradient for different geometries. This work is the basis for the assessment of the initial gas loss from core after it is retrieved from a borehole. In this the core is assumed to diffuse radially from the core. A plot of the gas volume desorbed versus the square root of time is projected back to the estimate of time zero to find Q1, the gas lost during core retrieval. This procedure produces surprisingly good straight lines. From the slope of this graph, the total gas content and the core diameter it is possible to calculate a theoretical diffusion coefficient. If the later diffusional behaviour of the core is examined by the same theory then the coefficient of diffusion may be found to drop to a value that is typically a fraction of the initial value. Thus we could think of coal as having a two stage diffusional process. In reality the coal probably has a multi stage diffusional process corresponding to the varying micropore sizes and the spacing between cleats in the coal.

If we think a little more about core diffusing gas then the concept of a uniform cylinder of coal diffusing gas is the exception. In reality the coal is usually cleated and bubbling may be observed at the cleats in the early stages of desorption. In the later stages of desorption the core is either in a canister or dry and diffusing too slowly to see how the gas diffuses. Nevertheless this initial slope of the line of square root of time versus desorbed volume is an important indicator particularly if note is taken of the state of cleating in the core.

Bearing in mind the limitations of the mathematical theory of diffusion it is possible to use the numbers gained from core for an initial diffusion coefficient in estimating the gas release from diffusing particles. Unlike gas contained in pore space gas diffusing from coal particles is not instantaneously available to drive an outburst.

The energy available from expanding gas is by definition the integral of pressure with volume change. Mathematically this concept may be expressed in equation (4).

$$W = \int_{V1}^{V2} P \delta V \tag{4}$$

The energy available from gas being diffused out of a coal particle assumes that the gas comes out at a pressure and then expands to do work. The particle must be able to deliver this gas at a pressure and the gas must be able to expand. As the volume diffused is time dependent then the power of the expanding gas is dependent on time.

It must be borne in mind that in an outburst the breaking coal is not however a piston to be driven by an expanding gas source but is rather a group of moving particles with potential for leakage between. The rate of gas release and the power of the outburst is dependent on time. At small times the gas may act upon the broken coal. At long times the outburst event will be over and further gas production has no effect on the outburst. This short term gas production at pressure from the diffusing coal is extremely important in outbursting.

To be able to assess the energy that can be derived from degassing coal particles let us consider the measurement of diffusion coefficients.

Core Desorption

The rate of gas diffusion may be gained by examining the rate and total volume of gas release from samples of known geometry, Crank (1975). If a sample of broken coal is gathered for this purpose then it will need to be sieved to gain a measure of the sizes while if a core is taken then its diameter needs to be known. In reality it is also necessary to examine the core carefully for fractures and to decide whether its effective diameter is somewhat less than that of the core itself.

The equation (5) describes desorption from a cylinder with a uniform initial concentration.

$$\frac{M_t}{M_\infty} = 1 - \sum_{n=1}^{\infty} \frac{4}{JOR_i} \, e^{\left(-D\left(\frac{JOR_i}{a}\right)^2 t\right)} \tag{5}$$

where $\frac{M_t}{M_\infty}$ is the ratio of desorbed gas over the total gas that may be released

JOR_i are the roots of a Bessell function of the first kind

for the equation $J_O(a\alpha_n) = 0$

D is the diffusion coefficient

t is time

a is the radius of the cylinder

For small values of Dt/a^2 this equation may be approximated to equation (6)

$$\frac{M_t}{M_\infty} = \frac{4}{\sqrt{\pi}}\left(\frac{Dt}{a^2}\right)^{\frac{1}{2}} - \frac{Dt}{a^2} - \frac{1}{3\sqrt{\pi}}\left(\frac{Dt}{a^2}\right)^{\frac{3}{2}} + \ldots \qquad (6)$$

These equations are plotted in Figures 16 and 17. These show plots of Dt/a^2 (Figure 16) or the square root of Dt/a^2 (Figure 17) versus the ratio of gas diffused to gas available to be diffused. In Figure 16 the value of the diffused ratio approaches 1 for large values of Dt/a^2 as represented by equation (5). The upper curve represents the first term of equation (6) while the curve below the full solution represents the first and second terms of equation (6). The lowest line represents the first, second and third terms of equation (6). The values of Dt/a^2 which lead to errors of 5% and 10% for different solutions of (6) to the full solution are in the diffused gas ratio are presented in Table 4.

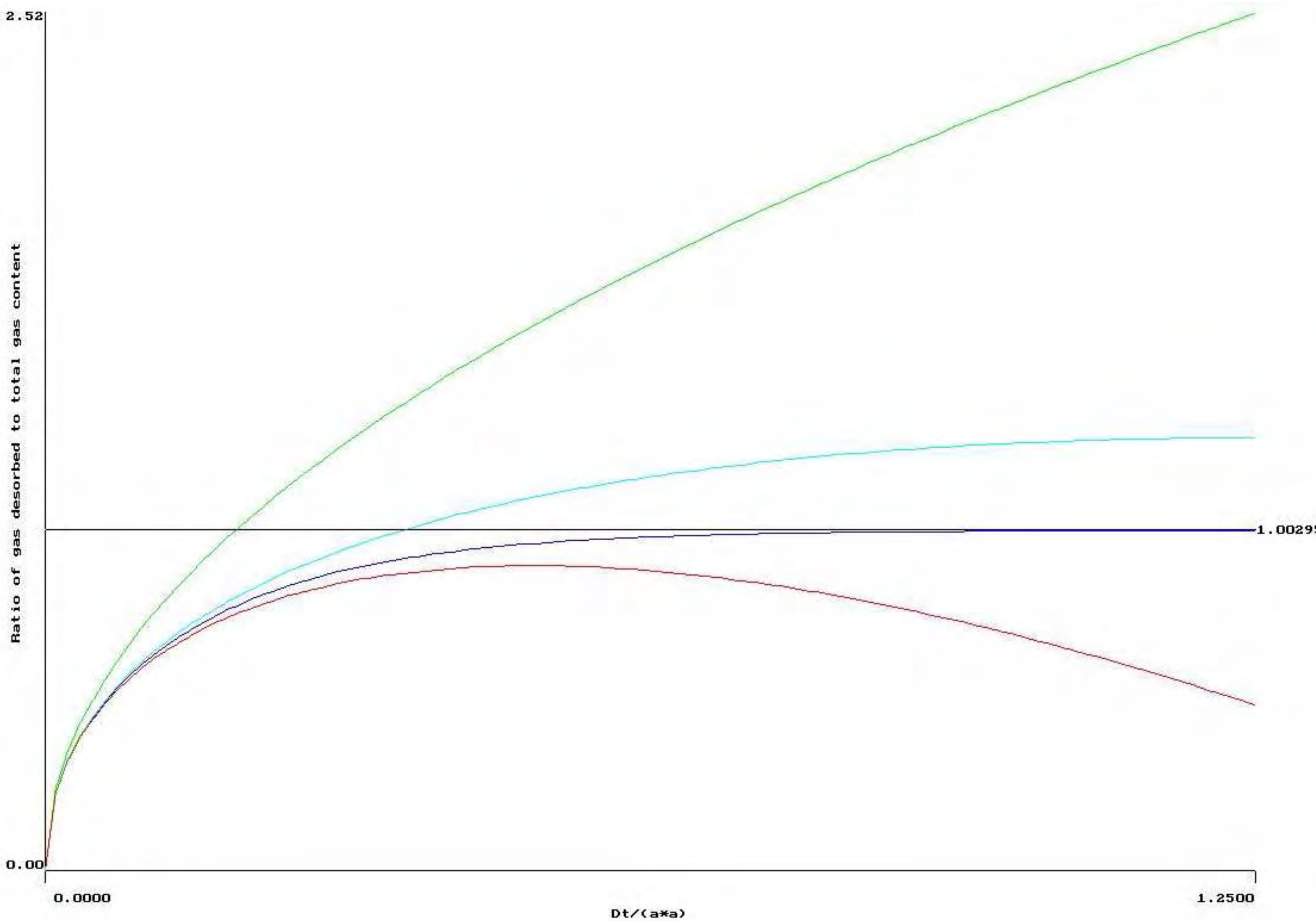


Figure 16: Solution to Diffusion from a Cylinder

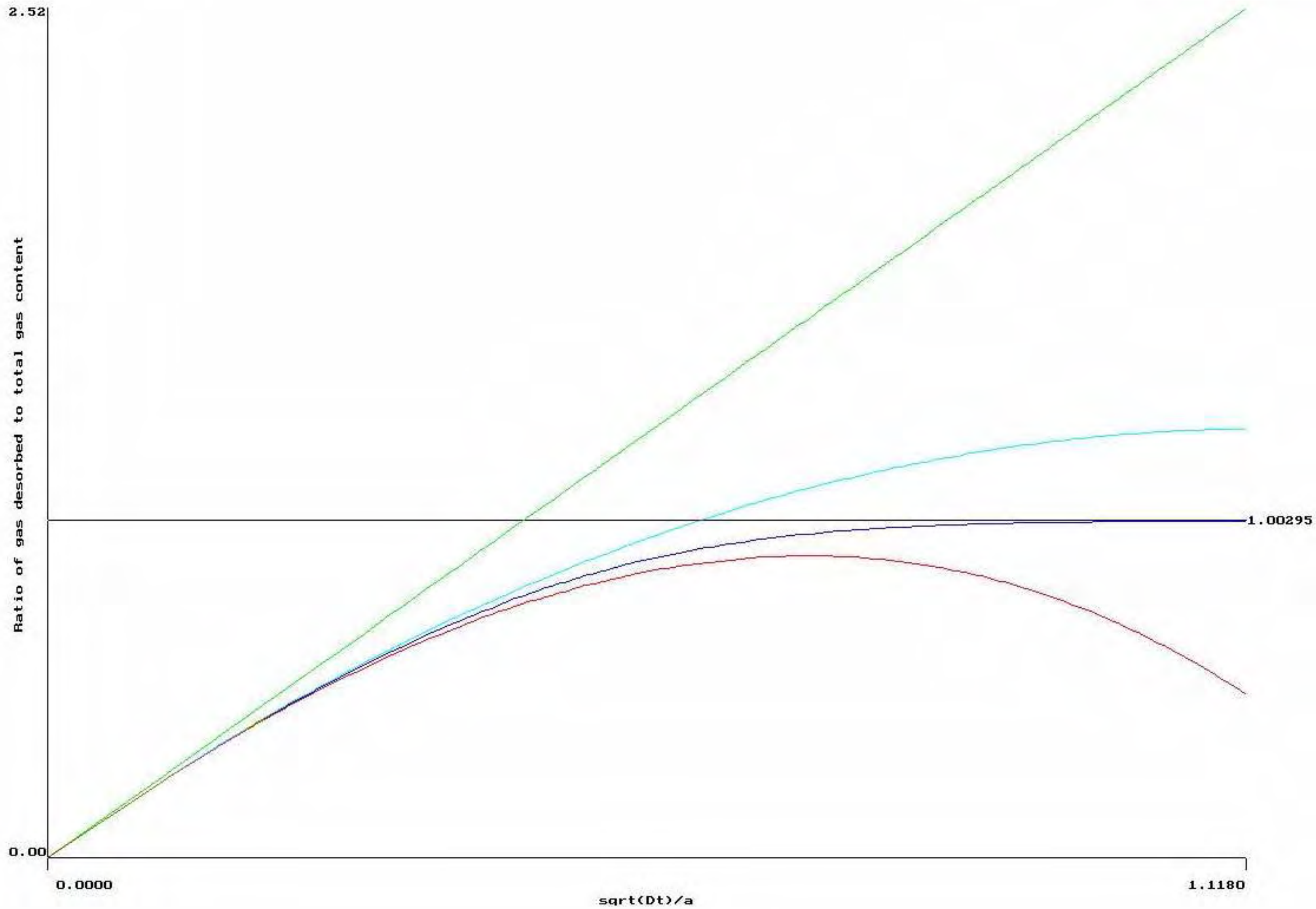


Figure 17: Square-Root Solution to Diffusion from a Cylinder

	1st term	**2nd term**	**3rd term**	**% error**
Dt/a^2	0.0125	0.265	0.4	5
Dt/a^2	0.050	0.40	0.625	10

Table 4: Evaluation of Dt/a^2 for 5% and 10% errors using the 1st, 2nd and 3rd term solutions of equation (6).

From this it can be seen that the first term solution of equation (6) which corresponds to the square root of time plot used to assess the lost gas in core desorption. This is valid within the 10% error range provided that Dt/a^2 is less than 0.05. Manipulation of this first term approximation of equation (6) shows that it is therefore accurate to within 10% provided that the ratio of gas desorbed over gas available for desorption is less than 0.505. For gas desorption ratios less than this it is possible to use the ratio of desorbed gas to gas content, the time and the core diameter to calculate the diffusion coefficient. Alternatively the diffusion coefficient may be calculated from the slope of the initial desorption data by making a plot of the gas desorbed versus square root of time. For HQ size core (61 mm diameter).

In real terms an unfractured HQ core that desorbs 10% of its gas content in 20 minutes has a diffusion coefficient of 5×10^{-9} m^2/s. If the effective diameter of the core due to fractures is 20 mm rather than 61 mm the diffusion coefficient becomes 6.5×10^{-10}. This illustrates the importance of gaining a grasp of the fractures in core when assessing diffusion coefficient.

Diffusion From Particles

The diffusion of gas from spherical particles is described by Crank (1975) in equation (7) which may be reduced to equation (8) for small values of Dt/a^2.

$$\frac{M_t}{M_\infty} = 1 - \frac{6}{\pi^2}\sum_{n=1}^{\infty}\frac{1}{n^2}\, e^{\left(\frac{-Dn^2\pi^2 t}{a^2}\right)} \qquad (7)$$

$$\frac{M_t}{M_\infty} = 6\left(\frac{Dt}{a^2}\right)^{\frac{1}{2}}\left\{\pi^{-\frac{1}{2}} + 2\sum_{n=1}^{\infty} ierfc\frac{na}{\sqrt{Dt}}\right\} - 3\frac{Dt}{a^2} \qquad (8)$$

ierfc is the complimentary error function

In this case *a* corresponds to the radius of the spherical particle.

These equations have been used to calculate gas release from different size particles at different pressures. In addition to the volume of gas released the work that the expanding gas may perform has been considered. This assumes that the gas expands down to atmospheric pressure in an outburst process that lasts up to two seconds in duration and is considered to cease when the power of the expanding gas drops to below 0.1 MW per cubic metre of coal. To permit this simulation a sorption isotherm approximating that of Leichhardt Colliery coal has been used. This is shown in Figure 18. The model is shown schematically in Figure 19.

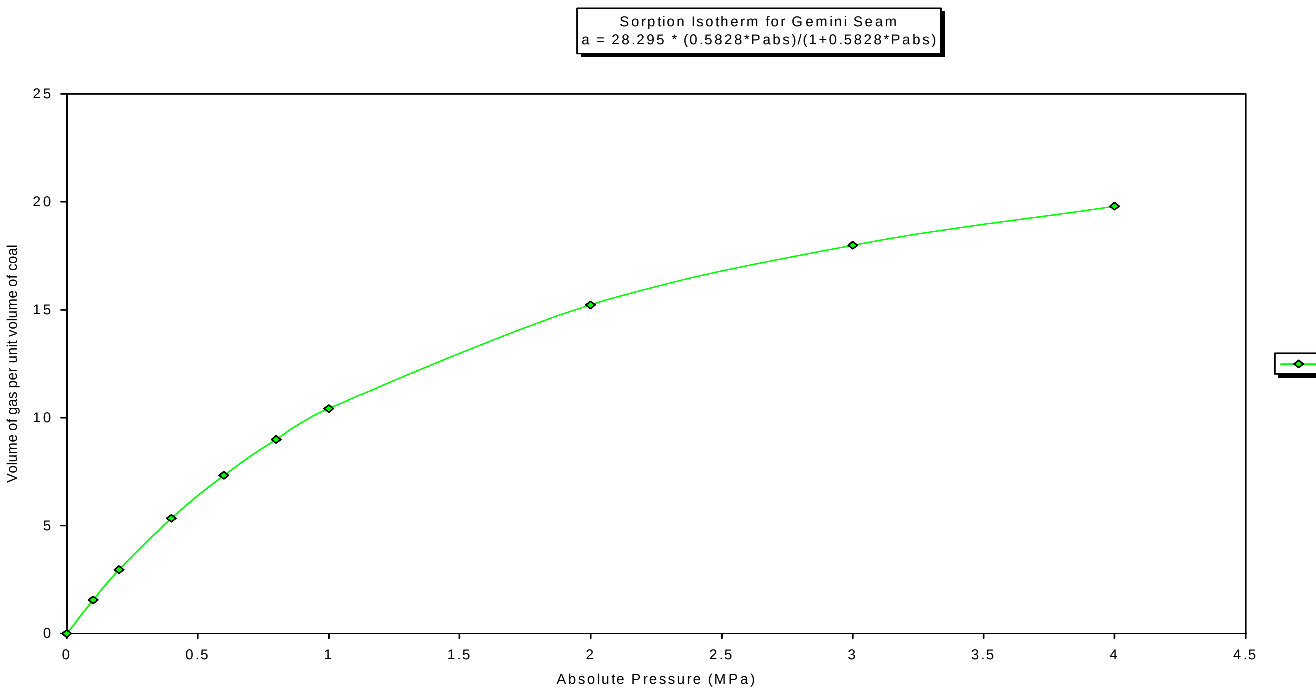


Figure 18: Sorption Isotherm for Gemini Seam, Leichhardt Colliery

Figure 20 shows the modelled energy release from a cubic metre of coal with varying gas pressure and particle size. The diffusion coefficient is quite high at $1\text{x}10^{-8}$ m^2/s. As can be seen the energy release for particles of 0.1 and 0.32 mm is virtually the same. This means that these particles have given up their energy. A particle size of 1.0 mm diameter has given up a substantial part of its gas and energy while that from a 3.2 mm particle is approximately a third of that of the finer particles. Particle sizes of 10 mm and greater simply can not desorb enough gas to pose any concern. The curved nature of the plots is essentially a function of the shape of the sorption isotherm.

Figure 21 shows the modelled energy release from a cubic metre of coal with a diffusion coefficient that is $1\text{x}10^{-10}$, i.e. one hundredth that of the previous example. Here the energy release for the finest particle size (0.1 mm) is very significant but by the time the particle size has increased to 0.32 mm the gas energy release is more than halved and for the case of a 1 mm particle is very small. Coarser particles would produce negligible energy release.

These plots show that the energy release with expanding gas from broken coal may be very significant with energy release values of 1.8 MJ/m^3 at 4 MPa gas pressure for the finer particle sizes. These are very significant levels and are much higher than those coming from a porosity of 5% (0.33 MJ/m^3) and strain energy (0.16 MJ/m^3).

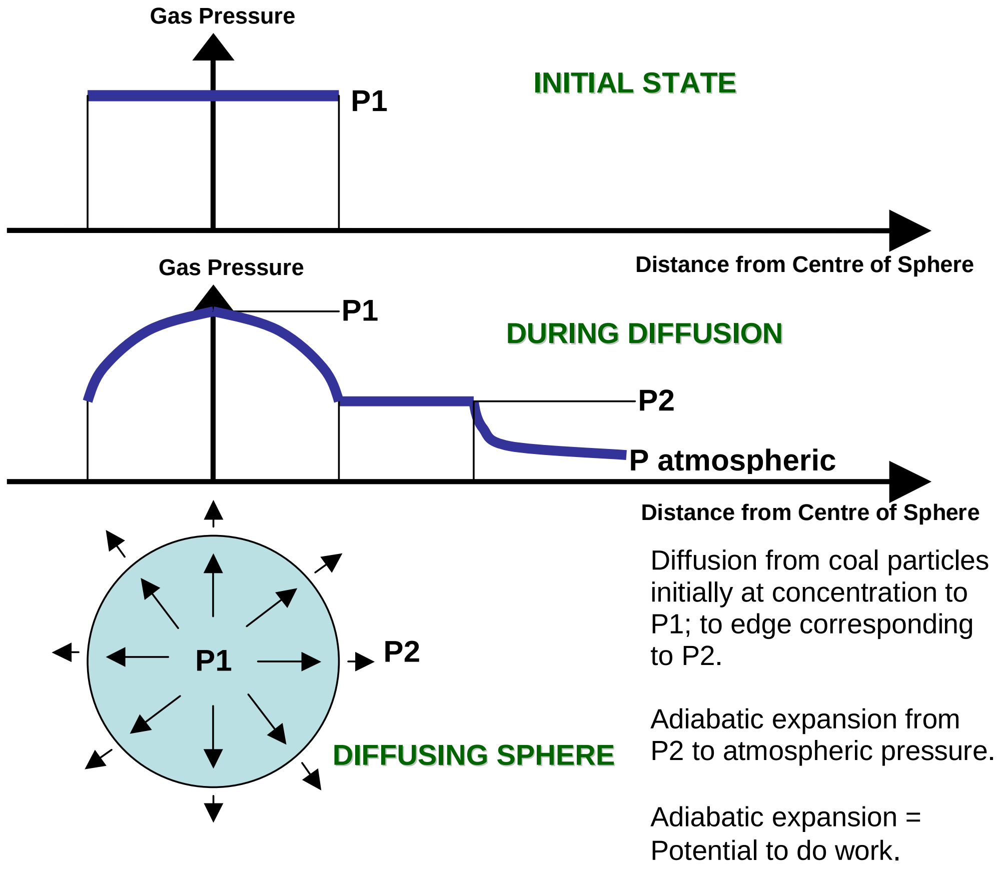


Figure 19:: Spherical Diffusion Model

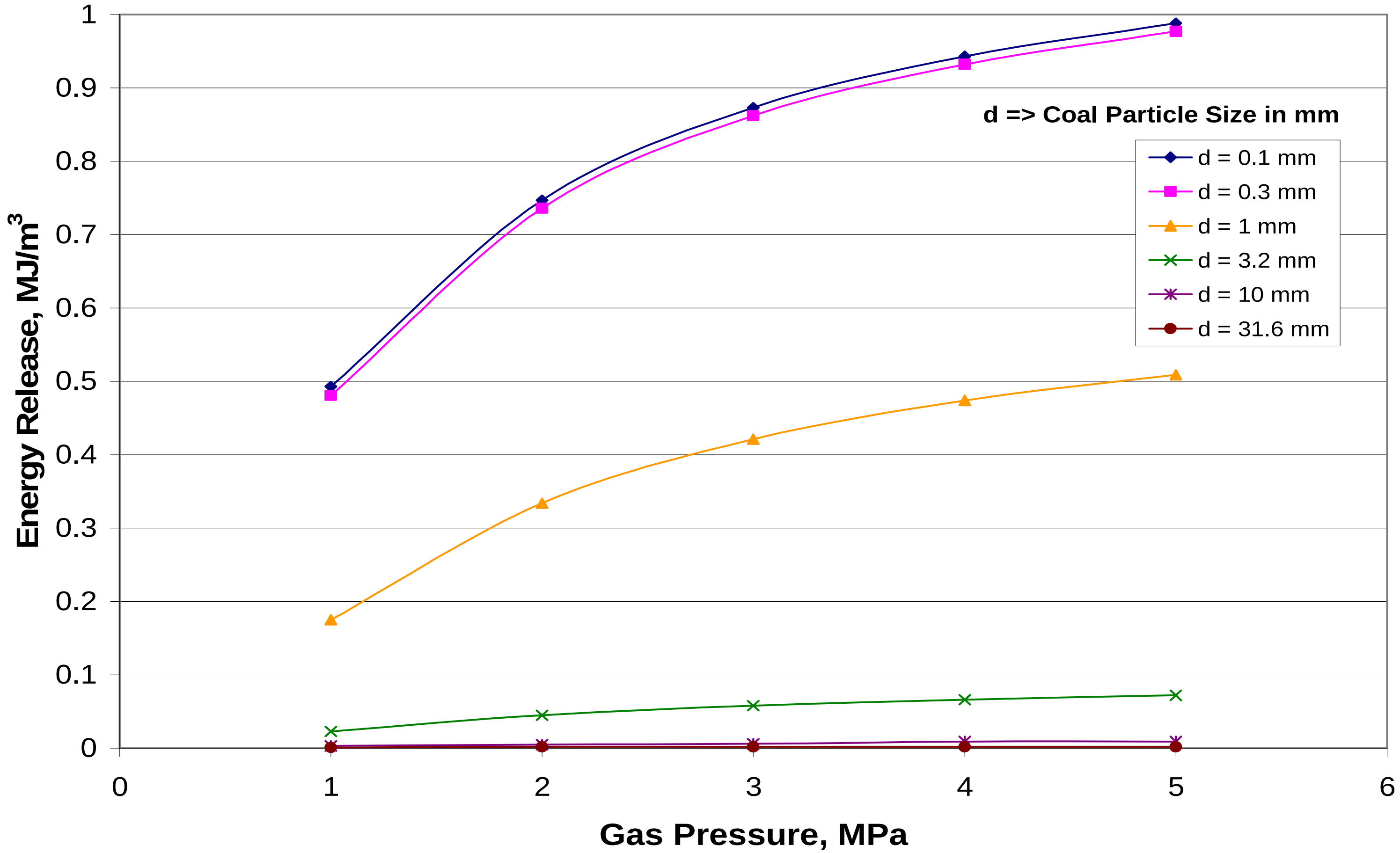


Figure 20: Gas Energy Release, Diffusion Coefficient = 1×10^{-8} m^2s^{-1}

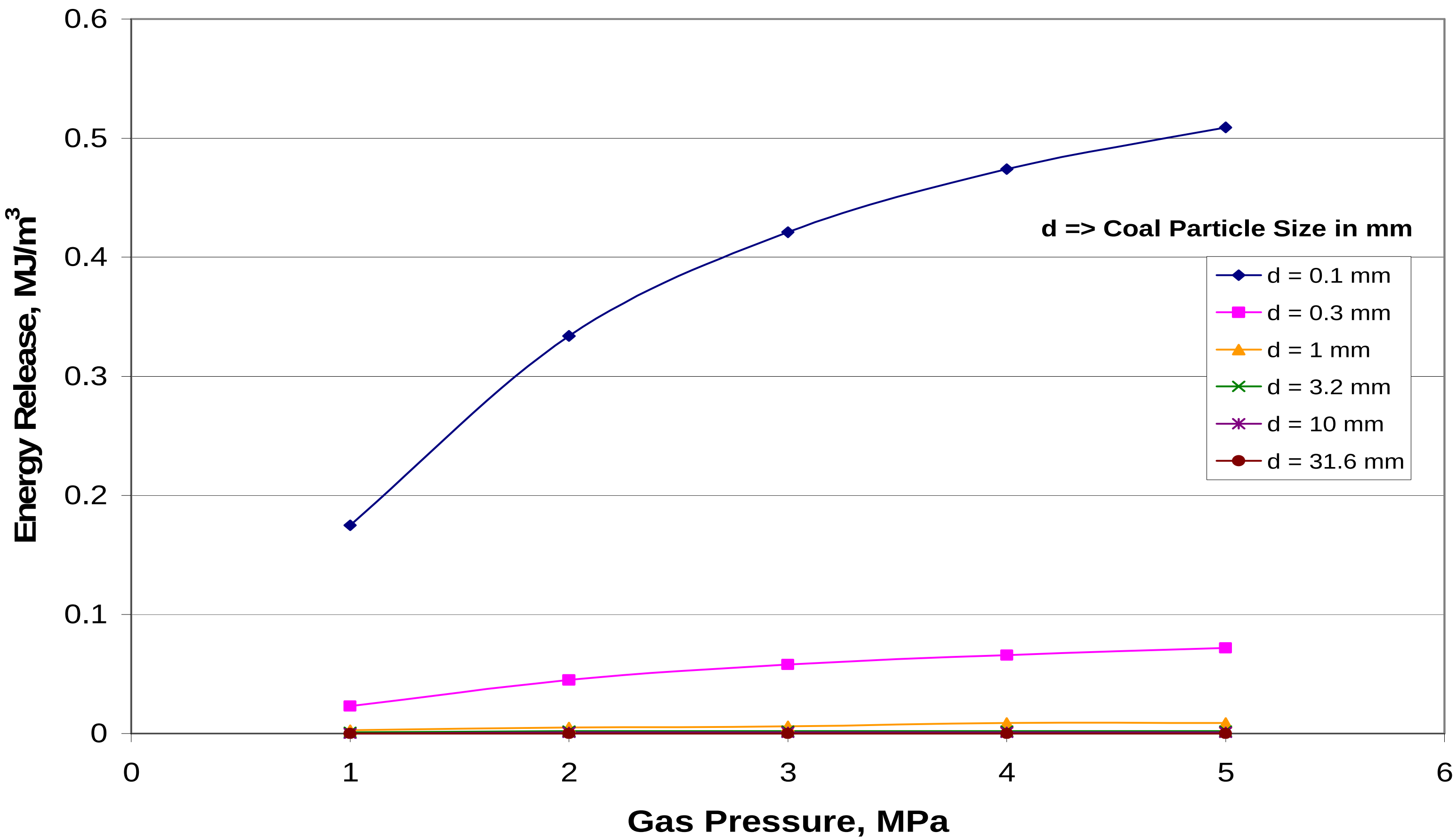


Figure 21: Gas Energy Release, Diffusion Coefficient = 1x10^{-10} m^{2}s^{-1}

4.2.4 Fluidised Movement of Coal In a Gas Stream

The ability of a gas stream to entrain particles is related to the size of the particles, their density , the velocity (squared) of the gas stream and the density of the gas. Particles ejected from an outburst will be projected outward and will in general separate out from the gas flow and fall to the ground. Fine particles may be expected to be held in turbulent flow of gas while heavier particles will not. However heavier particles already at velocity will project further as drag will have less of an effect in slowing them up.

The actual modelling of the movement of particles of different size in a slowing gas stream is a complex process beyond the scope of this report. However the order of importance of particle size and velocity with regards to entrainment can be gained by an examination of the much simpler case of the velocity of gas required to support a particle against gravity. The results of this calculation are shown in Table 5.

Coal Particle Diameter	Velocity Methane	Velocity Carbon Dioxide
mm	m/s	m/s
0.1	2.3	1.4
0.5	5.0	3.1
1	7.1	4.3
5	15.9	9.6
10	22.5	13.6
50	50.4	30.5

Table 5: Velocity of upward gas stream required to maintain coal particle in suspension.

Even the upper end velocity contained in this table is quite realistic in a violent outburst. What is particularly worthy of note are the lower velocities required to suspend a particle in an upward stream of carbon dioxide as opposed to methane. This effect is due to the higher density of carbon dioxide.

4.3 The Importance of Structure

In all outbursts recorded in Australia geological structure is of great significance. This applies not only to the outbursts that have occurred in obvious gouge material within a fault zone but also to the solid coal outbursts that occurred at Leichhardt Colliery. In the latter case the highly directional cleating controlled gas drainage characteristics and the nature of the failure that lead to an outburst.

Most geological structures influence the way in which gas can drain within coal seams. All structures from faults to joints and cleats may supply gas or lead to it draining away. Dykes may act as gas barriers.

The way in which geological structures have altered the coal properties is also very important. This may be by simply macerating the coal to small particles or by weakening the material so that it will break up to form small particles on failure. This may be well illustrated by an example of coal which I took from a fault zone where an outburst had occurred in the Bulli seam at Metropolitan Colliery. The fault fill material was not particles but it was very weak and broke away in large flakes as I pulled it out. It could be very easily crushed within my palm.

In addition to mechanical properties there is the need to consider how structures have influenced the gas type and the chemical properties, particularly the diffusion coefficient and the sorption isotherm.

4.4 What We Need to Know to Determine Whether an Outburst Will Occur.

Outbursts do not occur without failure of coal from all of the stresses that act upon it. Unfortunately relying on no failures occurring in coal mining is not realistic.

The next fact to consider is whether there is gas contained within the coal. If there is negligible gas in the coal then an outburst will not occur though strain related energy releases may eventuate. These strain energies are not very high near an unconfined face but may become high further in from the face. If the face fails suddenly to expose these more stressed coals at depth then these strain energies will need to be considered further.

If there is gas within the coal then we need to determine whether it can lead to an outburst. The fundamental question to be asked is will the gas be released quickly enough that a significant energy release will occur. To answer this question we need to know the gas pressure/content, the diffusion coefficient and what particle sizes need to be considered.

The gas content is regularly measured and the pressure can either be derived from gas content measurements and sorption isotherms or by direct measurement. The diffusion coefficient can be determined to some degree from examining the desorption characteristics of core or particles. The question which may be more difficult to answer is that related to particle sizes.

If we have gouge material in a fault then hypothetically we may be able to measure particle size by sampling. Due to physical difficulties this is not however a practical option for each fault zone. It is also possible that the particle sizing may change during an outburst. This applies to either solid coal or to gouge material.

If an outburst may occur as a result of gouge material being present in a fault then the volume of the gouge material that may be encountered is important.

Finally we need to decide what level of energy release constitutes a hazard.

5. Measurements That Can Be Made

5.1 Gas Content/Desorption Rate Tests

It has long been practice to measure gas contents by taking core either from surface or from underground, placing the core into a canister and measuring the gas release with time. The initial gas loss is calculated from the initial desorption rate and projecting it back to an estimated time when desorption from the core commenced. If a long term desorption is undertaken then it is sensible to fit a desorption curve to this and to extrapolate the results to find the total gas content. If only a short desorption period is permitted then the remaining gas content must be found by crushing the core to reduce it to a small size so that desorption occurs rapidly.

Provided that the core is cylindrical it is possible to calculate the diffusion coefficient from the desorption behaviour. This requires the measurement of core size, gas release rate and total gas content.

If the core is broken or if coal particles are obtained it is still possible to measure gas content and diffusion coefficient. Care must however be taken to get realistic dimensional estimates. In the case of broken coal the fines may emit gas too quickly for a sensible measurement to be made, i.e. most of the gas has been lost before the coal can be put in any form of canister.

To illustrate the effects of particle size on diffusion two figures have been prepared. These show the proportion of gas diffused from spheres of different sizes for two diffusion coefficients, $1x10^{-8}$ m^2/s and $1x10^{-10}$ m^2/s.

Figure 22 shows the case for a diffusion coefficient of $1x10^{-8}$ m^2/s. In this it can be seen that particles smaller than the largest considered (31.6 mm diameter) have lost most of their gas before any sampling system can be practically deployed. This is a real warning for those who sample broken core as they may have lost the bulk of their gas before they get it into a canister. Figure 23 shows the case of a coal with a diffusion coefficient of $1x10^{-10}$ m^2/s. Here the smallest sample that could usefully be sampled is 3.16 mm diameter.

It is currently thought that the likely diffusion coefficients lie between the values used in these graphs, i.e. $1x10^{-8}$ and $1x10^{-10}$ m^2/s. Any sampling that is to be done should accommodate this range of diffusion coefficient and this poses difficulties related to time and particle size. Coals with an outburst risk are likely to produce fines and to have a high diffusion coefficient. This means that obtaining gas contents and diffusion coefficients is difficult in these coals. The gas will have substantially gone from the small particles assuming that they can be sampled.

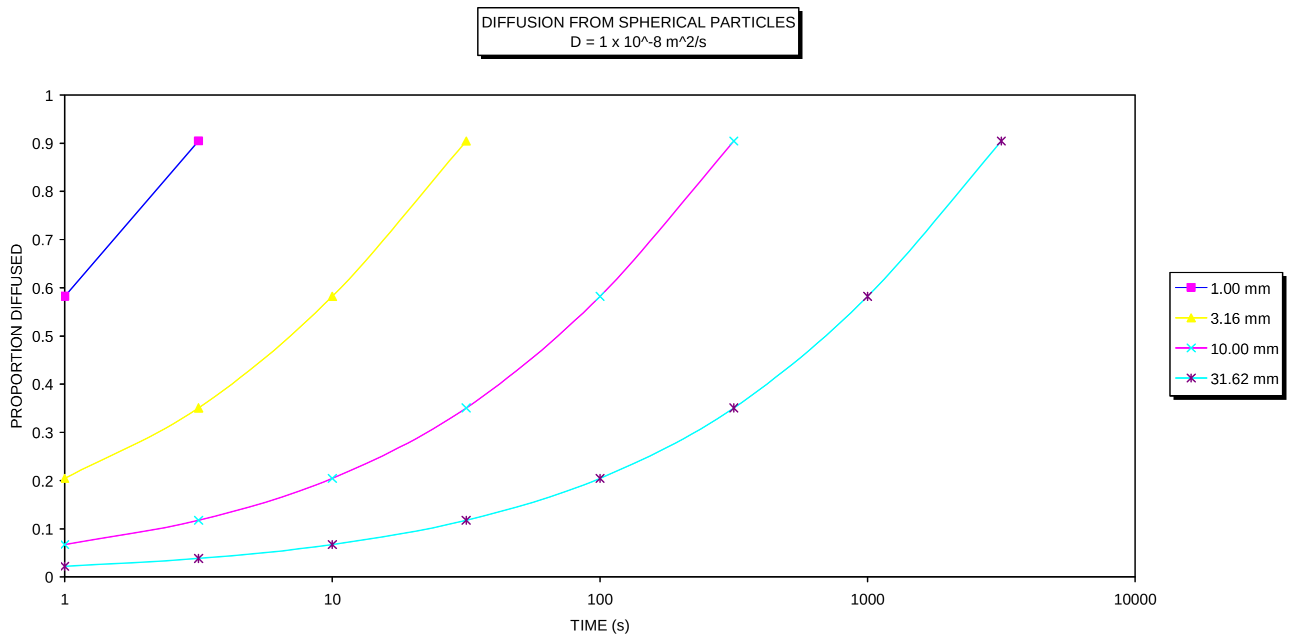


Figure 22: Proportion diffused from spherical particles
Diffusion coefficient = 1x10^{-8} m^2/s

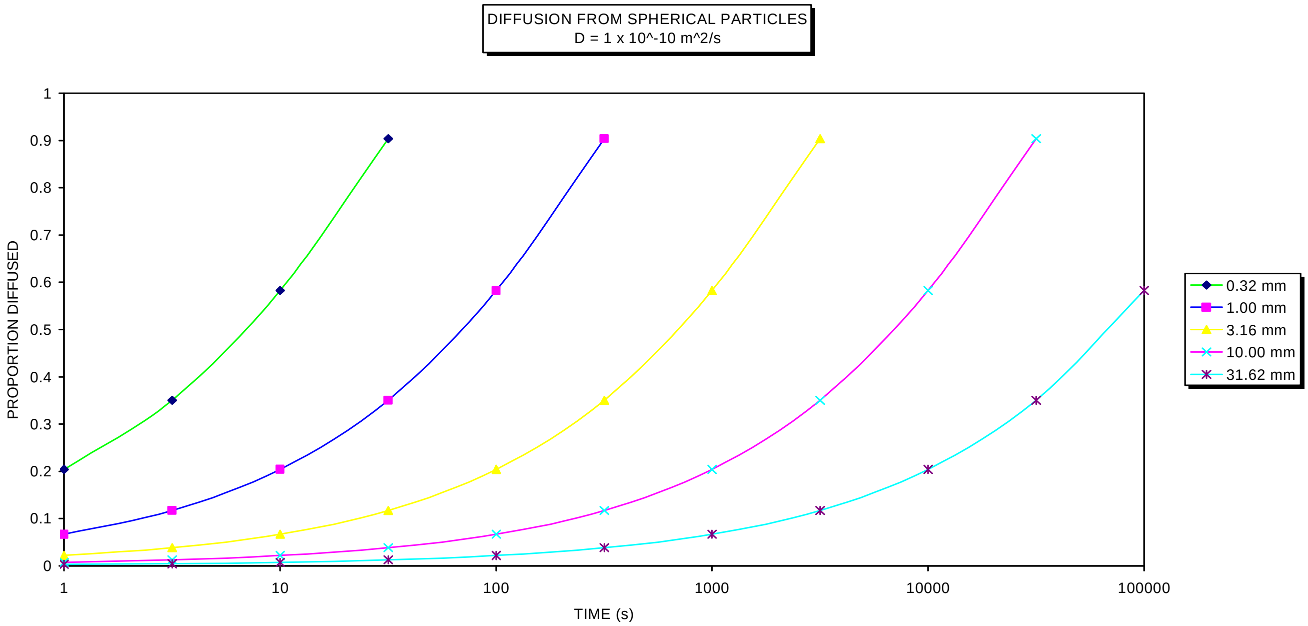


Figure 23: Proportion diffused from spherical particles
Diffusion coefficient = 1x10^{-10} m^2/s

5.2 Index Tests Expanded

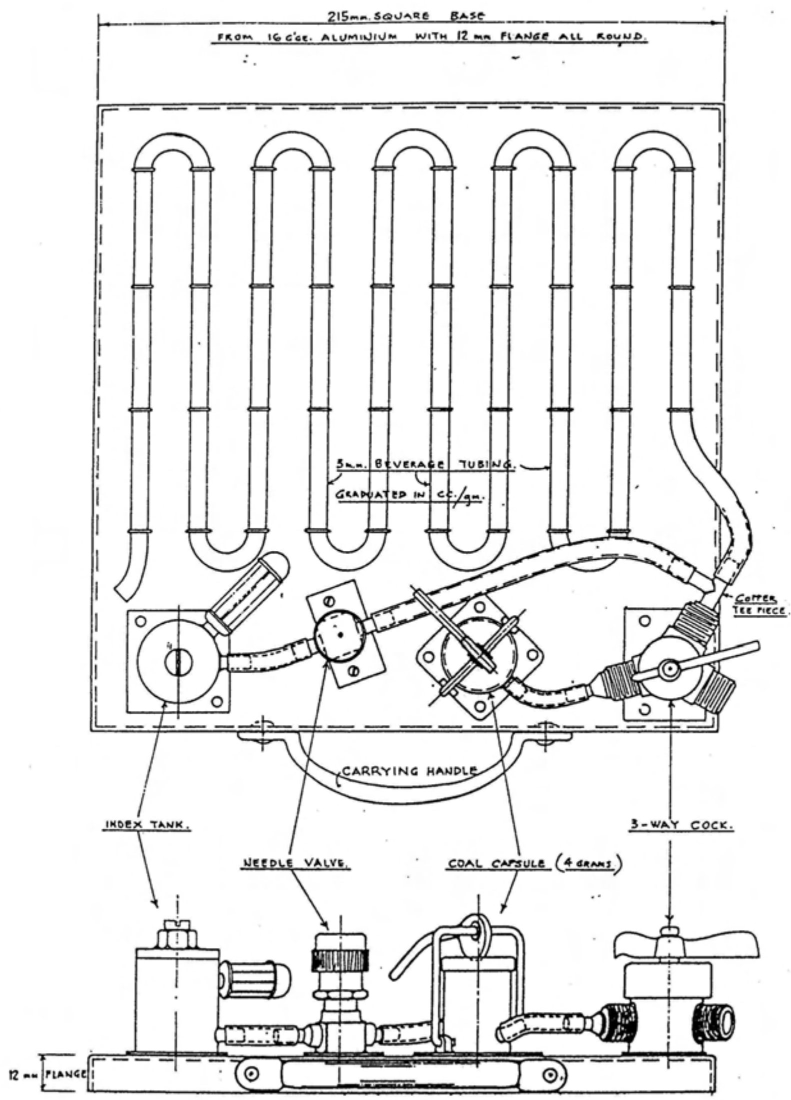
Those who work in the area of outbursting have long wished to find a simple index that will tell them whether an outburst will occur or not. Several index tests have been developed and used with varying levels of success. The lack of universal success of any of these tests is due to the complex nature of outbursting. However it is useful to have a look at what others have done and why they have limitations. One of my prime criticisms of any index test is that it does not provide a measurement of any fundamental parameter such as diffusion coefficient. As such, each test result can only be used for comparison with another test conducted by similar means and cannot be used as a fundamental parameter in its own right. The benefit of fundamental parameters is that they can be used in models such as the energy release model presented earlier.

5.2.1 Gas Content Testing

While this is an attempt at measuring a more fundamental parameter it is has been used as an index of outburst proneness in Australia. This use has been successful in the Australian context but has probably slowed production needlessly in areas of coal that is hard and slow to drain.

5.2.2 Hargraves' Emission Value

Dr Alan Hargraves developed an emission value (EV) meter which is pictured in Figure 24



EMISSION VALUE METER.

Figure 24: Dr. Hargraves Emission Value (EM) Meter

This device was designed to measure the gas release from coal cuttings taken by the use of a hand held auger drill of 43 mm diameter. The auger was used to avoid the introduction of moisture to the coal. The procedure was to drill to 6' (1.8 m) and to clear the cuttings from the hole. The last foot (300 mm) of hole was then drilled, the cuttings taken and sieved so that sample sizes between 0.125 and 0.5 mm could be placed in the EV meter canister. This process had to be completed in one minute which was a very tight time constraint. On initial placement the canister was vented through a three way valve. A droplet of glycol was then injected into the measurement tube and the three way valve used to divert emitted gas into the tubing behind it. The volume of gas released from the coal was then measured over five minutes. The canister held a nominal 4 grams of coal which may have been weighed after testing in some instances. The EV value was the gas emission from the five minute period divided by the mass of the coal and was therefore expressed in terms of cc/gm (equivalent to cubic metres/tonne). The device was used with some success at Metropolitan Colliery and at Bowen No. 2 mine, Collinsville both of which were mines with carbon dioxide as the principal seam gas. The critical EV value was about 1 cc/gm above which measures had to be taken to degas the seam. The EV meter took small samples and the results could be manipulated by the user choosing the coal ply which the sample was taken from. Biggam et al (1980) report a trial at Bowen No 2 mine, Collinsville where 60 readings were taken from a face. These showed a variation from 0.43 to 1.18 cc/gm. Examination of the readings show a distinct pattern and that these variations were a function of location on the face rather than the technique itself. In fact 72% of readings were within 0.2 cc/gm which was remarkably consistent.

The method was also used in mines with methane as the seam gas. Hanes (2006) is of the view that the device was quite successfully used at Leichhardt Colliery with two quite different problems. The first is the reluctance of the crews to use the device and their manipulation of the results by choosing sections of the face from which to take a sample. This was relatively easy to do at Leichhardt as it was known that a sample taken from the corner of the face across the cleat would have a high reading while a sample taken with a drill hole along the cleat would produce a low result. This was consistent with drainage patterns along the face. The potential conflict of sampling results with production bonus arrangements was always an issue. Changing mine culture could have solved this.

The second problem perceived was that the EV invariably dropped with moisture in the coal and that these areas were those that were outburst prone. That outbursts occurred in areas that were moist is quite explicable as these were areas that had not drained and the original seam moisture was still in place. What is particularly interesting though is the apparent drop in diffusion rate with the presence of water in the coal. This requires further investigation.

Hargraves EV meter seems to have had all of the elements of a successful system but with a few major flaws. It combined taking coal of known size at a known time and measuring the emission rate. The major limitations of the test appear to have been associated with the small particle sizes taken by the auger drill and possibly the effects of moisture on the desorption rate. The small particles size matters because even with a slow diffusion rate of $1x10^{-10}$ m^2/s a 0.32 mm sample could be expected to have lost virtually all of its gas within the 60 second period from drilling to the commencement of measuring. If bigger particles could have been tested and the test continued for a longer period then the gas emission characteristics could have been determined and extrapolated to produce lost gas and gas content and in addition the diffusion coefficient.

The Hargraves EV meter as used took just a snapshot of the late stages of diffusion from small particles. The problem with this is that it is not possible to use its results to separate the important parameters of gas content from diffusion rate.

5.2.3 Other Desorbometers

Both Lunarzewski (1995) and Lama (1995) describe a sampling system used in Poland that resembles the Hargraves EV meter remarkably. Here, once again a dry coal sample of size range 0.5 to 1.0 mm and 4 gm mass had to be obtained by auger drilling and placed in a canister within 90 seconds. The volume released was then monitored over the next 120 seconds. The gas release rate was correlated to total gas content. Unfortunately this correlation could only apply to a specific coal and the device otherwise has the same limitations as the Hargraves device.

Noack et al (1995) describe yet another device of this kind called the condenser-barrier desorbometer in which a sample is placed and the initial flow rate measured. This device was used in Germany.

I understand that desobometers are also in use in China.

5.2.4 Laboratory Tests

There are a series of laboratory tests that can be undertaken on coal samples to determine the rate of gas absorption or desorption from coals. Lama (1980) describes several of these. In essence they involve placing a sample of coal in a pressurised atmosphere of gas and measuring the uptake of gas by the sample. Alternatively they involve degassing an already gassy sample and measuring the rate of gas release. The index numbers arrived at by these are somewhat irritating as they are not related to anything fundamental. It would be far better if the result was related to a value of a diffusion coefficient rather than some test value index. This lack of relation of measurement to some fundamental parameter is always a serious limitation. It means that the result cannot be used for any other purpose than comparison with tests done by identical means.

5.2.5 DRI 900

Williams (1997) presents his concept of a Desorption Rate Index. This by definition is the amount of gas that is released from a core of 200 g mass within 30 seconds of being subjected to crushing in a rock crusher owned and operated by GeoGas Pty Ltd, the company owned by Dr Williams. The results are correlated to gas content values from core samples. The DRI 900 value corresponds to a gas release of 900 ml from the 200 gm sample. In his paper Williams shows an empirical correlation of the DRI 900 value to between 7 and 10.3 cu.m/tonne total gas content for methane as the seam gas. He also presents another empirical correlation showing that with increasing carbon dioxide in the seam gas the total gas content drops to 7 cu.m/tonne for a DRI value of 900.

In conducting the test GeoGas crush the sample to a small size so that one could expect that most of the gas in the coal would have been released. Williams however notes that different coals have different total gas contents for the same DRI value.

The test is a combined measurement of the crushability of coal, diffusion coefficient and gas content rolled into one. Weak coals will crush more and become finer while tough coals will crush less. Coals with high diffusion coefficient will tend to release more gas in the given interval as will coals with a higher gas content.

One obvious limitation of the test is that gas loss will occur from core prior to the test being conducted and that this must affect the test results.

The test is used as an outburst indicator with a DRI 900 value being used as the lower threshold indicator of outburst proneness. This is an empirical relation gained in an environment where few outbursts have occurred.

5.2.6 Borehole flow rate

In 2005 I visited mines in the Kuzbas in Siberia and found that they used borehole flow rate as an indicator of outburst proneness. Their measurement technique is to drill a test hole into the face and to insert a packer into that hole as quickly as possible. The flow from the hole is then measured. A flow rate exceeding 4 litres/minute per metre is considered to be an indicator of outburst proneness.

This type of measurement seems to be based upon a model of a diffusing solid where the initial flow rate is at its highest and then declines. As such the measurement is a function of the initial gas content. Alternatively the measurement could be considered an indicator of structure. It seems that the test results are somewhat ambiguous.

Akabira colliery in Hokkaido Japan measured flow from boreholes and when the flow had dropped below a certain level they decided that it would be safe to mine that area. As they were fully aware that in many cases flow would not occur in some areas until some form of stress relief had taken place, usually by the use of large diameter drilling, they presumably protected themselves from falling into the trap of mining low permeability coals with high gas content.

Flow cannot be considered a general indicator of outbursting as in many cases low permeability leads to low gas flow and minimal drainage.

5.2.7 Pressure Build Up in Boreholes

Though the reference cannot be found at this time I do recall reading that in the 1970's some of the German mines used the technique of drilling a hole in the face and inserting a packer, setting it and then watching pressure build up. A pressure rise rate exceeding a certain value was considered to be dangerous.

This is a test that can be useful, but when used by the unwary can lead to problems. The test can yield information on seam pressure which is in many ways as good or better an indicator of outburst proneness than gas content. However the test method has limitations in the quality of the seal that can be developed between the packer and the borehole wall. It also has significant problems in very impermeable coals. Here the flow rate into the hole is sufficiently low that the storage volume in the borehole significantly slows the build up rate. Indeed in one instance at Dartbrook Colliery, it took nearly two months for pressure to build up to seam pressure because of this effect. Being able to recognise these effects takes technical skill and experience.

5.3 Strength Tests

Coal strength can be measured by the processes that are used for other rocks. Such tests include uniaxial compression tests, triaxial tests and point load tests. All these tests are more difficult to conduct on coal because of its weak and often inhomogeneous nature. The measurements that come from them are useful in arriving at Young's modulus, Poisson's ratio and the strength at whatever state of confinement exists. From this the potential strain energy in the coal mass can be calculated. This strain energy may in some cases be quite significant.

5.4 Toughness Tests

Toughness of a material is by definition the amount of energy that is required to create a fracture of unit area. The amount of this energy is clearly dependent on the mechanism by which it is applied. Thus we could expect that the energy required to propagate a fracture in compressional loading is quite different from that supplied by internal fluid pressure expanding a crack.

Most used toughness tests on rocks involve some sort of cyclic loading that breaks down the sample from one sizing to another. Examples of such tests are the Hardgrove grindability test which is used on coal to the Los Angeles test which is used on aggregates. These tests hardly model the circumstance that exists in an outburst where failure is initiated by stress and internal fluid pressure.

However toughness is important as the energy required to fragment the coal in an outburst situation has to come from somewhere. The options are from strain energy or from expanding gas. It is important however to keep in mind the failure strains and to ensure that they are matched to the source of the energy producing the fracture. Thus one might expect that the energy coming from surrounding coal or rocks is in the right strain range as that required to produce fractures in the coal. Unlike this the energy coming from expanding gas derived from diffusion would principally come long after failure.

Any test system currently used to measure toughness on coal is likely to be inappropriate as a direct measure of a fundamental parameter. Therefore either some new test needs to be devised or use must be made of existing tests but with due regard for their shortfalls.

5.5 Fluid Pressure and Permeability Measurement

It should be noted that I have avoided discussing permeability in this report. The reason for this is that whether an outburst occurs or not is not a direct function of permeability. Permeability and other reservoir parameters combined with mine drainage into roadways will determine what gas pressure/content exists near a mine face. This value of pressure/gas content is important in part in determining whether an outburst will occur or not.

The measurement of permeability is however useful in determining whether a state that is conducive to an outburst occurring may eventuate. Virtually all permeability test techniques involve measuring pressures and permit either the direct measurement or an estimation of reservoir pressure by some form of extrapolation.

Any permeability test requires the reservoir (coal) equilibrium pressure to be disturbed by a flow from or into an opening. This must be combined with pressure monitoring either in that opening or in another part of the reservoir. If the test involves a single borehole then the pressure monitoring must be within that borehole. In this case it is best to produce from a borehole and then to seal it and watch the pressure build up.

Most measurements in coal from underground involve two phase flow of gas and water. This makes them interesting to interpret. However production tests always produce more reliable numbers than injection tests despite the complications in analysis.

From underground the most reliable form of test to measure reservoir parameters is a four hole trial. This is illustrated in Figure 25.

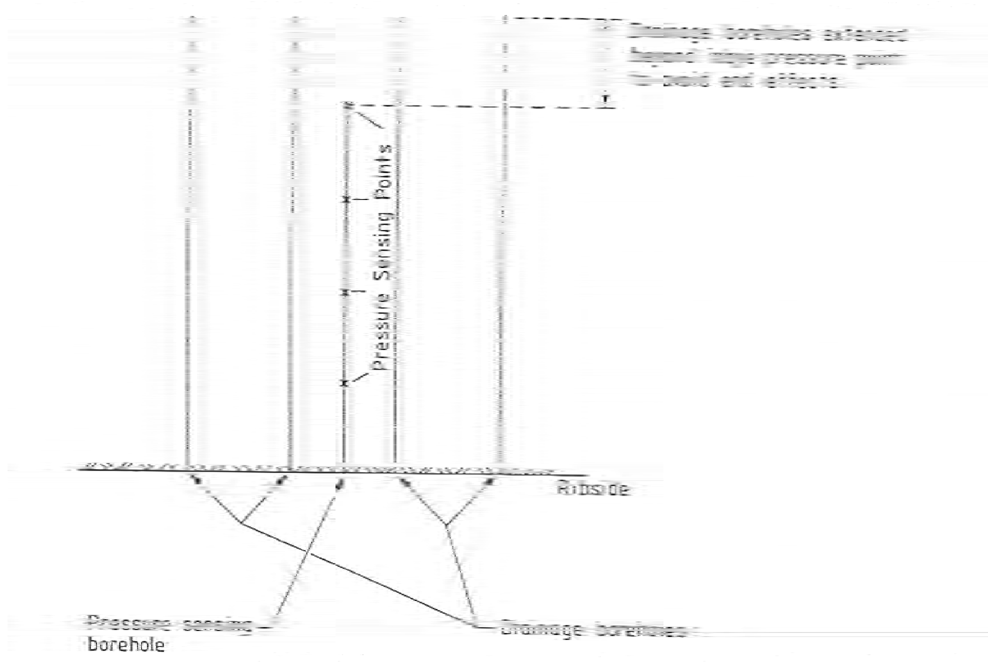


Figure 25: Four hole trial to measure reservoir parameters. Gray (1983)

First pressure sensing points are grouted into a borehole. Pressure is allowed to come to equilibrium and four flanking holes are drilled to drain gas and water from. Each of these is monitored for gas and water flow. In this type of test the outer holes act as a cut off to flow from the outside while the inner two act as part of a infinite drainage pattern at the spacing between them. The pressure sensing points in the middle hole serve to provide both a check on the material balance (gas initially in place - gas drained = gas in place) and as a basis for history matching with a simulator. The multiple holes also give good opportunity for comparing flows to check out seam homogeneity. If an incremental flow test is conducted in each of the four holes then a real picture of seam drainage characteristics can be built up. Ideally this test should be conducted both along and across the main cleat direction so that directional permeability can be assessed. Such test techniques were used at Leichhardt Colliery, Bowen No. 2 mine, Collinsville, and Moura No 4 mine. The only effort at such a test since has been at Dartbrook mine.

A simpler test system for permeability measurement from underground is conducted by drilling a borehole and inserting the equipment shown in Figure 26.

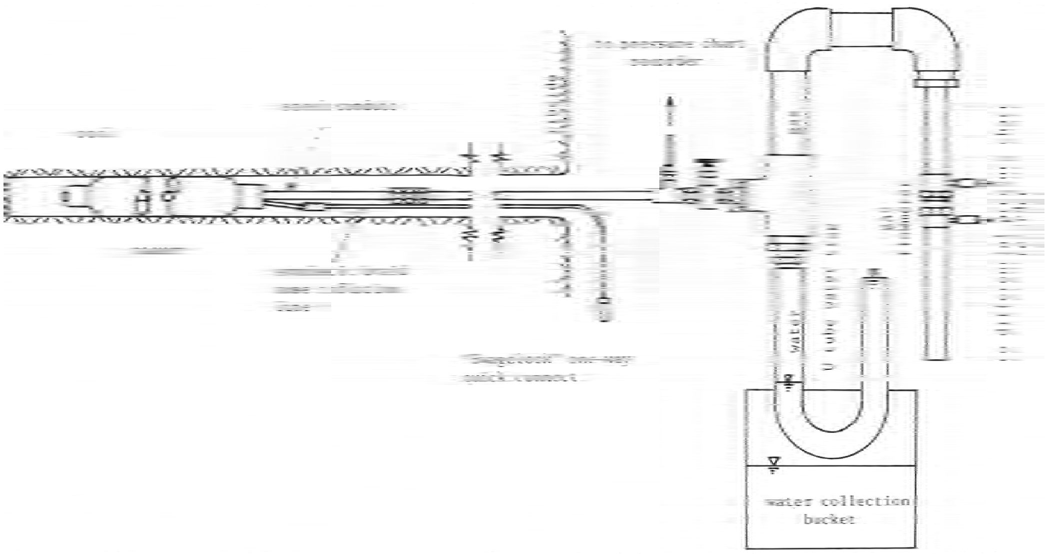


Figure 26:   Pressure Build-Up Test Equipment (not to scale). Gray (1983)

The packer is inflated and the water and gas flow are measured for about 24 hours. The valve is then closed and pressure is allowed to build up. The analysis of this event with the assistance of a simulator provides a basis for assessing permeability.

SIGRA Pty Ltd regularly measure permeability from surface boreholes by conducting drill stem tests and sometimes injection fall-off tests. We have also conducted interference tests.

A Drill Stem Test (DST) involves sealing above the seam and below the seam (if necessary) with packers. A valve at the bottom of the tool is then opened and production is then induced from the coal seam so that water and gas flow into the drill string which has been emptied. When a certain amount of flow has occurred the valve is closed and the pressure build-up is monitored. The bottom hole pressure at seam level is monitored during this test. Analysis involves using flow and pressure build-up information. Injection fall-off tests involve pumping water through a drill string into the coal seam and then closing off the bottom valve and watching the pressure decline.

Interference tests involve either production or injection from a borehole whilst monitoring the pressure changes at the seam in adjacent boreholes. Single hole DST and injection fall-off tests give non directional permeability information whilst an interference test provides the basis for estimating the directional components of permeability.

It must be remembered that the permeability of coals change during drainage. This change in permeability is brought about by changes in effective stress and by two phase effects caused by changes in saturation. Most coals reduce their permeability as the effective stress increases. This is because the cleats narrow as the effective stress rises with fluid pressure lowering. Another effect often reverses this trend. This involves shrinkage of coal with drying and the desorption of gas. The coal shrinkage causes the stress in the coal to lower and the cleats to open. The stress is redistributed to the roof and floor. The two phase effects on permeability may be explained simply in as much as when water occupies the cleats there is no room for gas to flow and vice versa. That is the presence of one phase prevents the movement of another.

Permeability is a measure of the dimension and tortuosity of fluid paths within the (coal) reservoir. It has dimensional units of length squared. Thus a measurement made with either gas or water flow, or a mixture of the two should be the same. In practise there are many factors to be taken into account in measurements. The first is wellbore loss effect. Frequently changed permeability conditions exist around a wellbore due to stress concentrations or simple lack of uniformity of cleating through the wellbore. Secondly the wellbore loss effects can be significantly changed by producing from the hole or injecting. This can be caused by the changes in effective stress brought about by the near wellbore fluid pressures. It can also be strongly influenced by particle build up on the well bore during injection thus increasing the local wellbore loss. If clays exist in the cleats of the coal then injecting a water of different salinity to those naturally existing may cause them to swell and change the permeability quite dramatically. For these reasons production tests analysed on the basis of the fluid mixture being produced are generally more reliable than injection tests though frequently more challenging to interpret.

Great care also needs to be taken of the fact that permeability of coals changes during production. If, as was the case at Leichhardt Colliery, the absolute permeability changes by more than two orders of magnitude during production due to stress changes and shrinkage then one measurement of permeability is simply a snapshot of what it is at that instant in time. Other factors such as shrinkage need to be taken into consideration. These factors become more important in weak, softer and more cleated the coals. Hard coals with high permeability, fewer (but often larger) cleats tend to be less affected by permeability changes.

5.6 Stress Measurement

Reliable stress measurement in coal is virtually impossible because of the cleated and weak nature of most coals. However, knowledge of the stress in coal is quite important from the viewpoint of outbursting.

Stress affects the permeability of coals and hence the way in which they drain in both magnitude and direction. It also affects the way in which coal fails and the energy that may be released on failure. Fortunately the strain energy component of outbursts in Australia does not yet appear to be so great that it is imperative to know stress. It is however still desirable to know the stresses that exist within the coal seam.
Stress measurement in rocks can be categorised in terms of reliability of technique in the order of overcoring, hydrofracture and borehole break out. In coals overcoring is generally impossible because the core fractures. Hydrofracture is limited because of the pre existing cleating within the coal. The determination of the major as opposed to the minor principal stress is always difficult. Borehole break out measurements are only useful if the borehole wall crushes. In coals the additional complication exists that failure is frequently controlled by cleating.

This means that indirect options must be considered to deduce likely seam stresses. These involve the measurement of roof and floor stress. From this the component of stress due to self weight in a zero lateral strain environment can be subtracted leaving what I refer to as the tectonic stress, Gray (2000). Tectonic stress is caused by external strains. These tectonic strains can be calculated and unlike stresses which vary with rock stiffness are frequently shown to be quite consistent across sedimentary strata.

The stress in the coal can then be calculated if an estimate of the Young's modulus and Poisson's ratio is known. This is achieved by calculating the stresses due to self weight and adding to them the stress due to the tectonic strain.

5.7 Open Hole Drilling Measurements

A borehole that is being drilled in coal by open hole techniques is a mini roadway in the seam. Thus it is a very useful model of what might happen in a roadway development albeit with limitations of scale. The other influence that is different is that back pressure may exist in the borehole particularly if the hole is being drilled with water.

It would seem to be sensible to monitor the development of a model roadway in the seam to the maximum extent possible. However this has not come about despite the significant efforts of several individuals and groups within Australia (Lunarzewski 1994).

As outbursts produce lots of fine particles and gas then it would seem these are what should be measured in the first instance. This is quite practical in short holes as gas release can be measured and the cuttings can be collected and their volume and size distribution measured. However as the holes get longer the effect of what is occurring at the end of the borehole becomes somewhat blurred by cutting separation and gas emissions into the hole generally. The situation could be improved by opting for a reverse circulation process in which the cuttings are returned down the centre of the drill string at a higher velocity than in the annulus. This was very successfully done at Leichhardt by the German firm Montan Consulting to measure gas content in about 1974.

In addition to focusing on what is cut or emitted from a borehole a lot can be determined by examining the drilling process itself. The prime measurements that can be made are the rate of penetration, bit thrust, bit torque, rotational speed of the bit and bit load. These were reported by Richard Danell (2000) for the monitored Proram drill. An area of potential outburst structure was detected by it at Appin and on subsequent mining, a small outburst occurred. These measurements were the subject of quite a detailed study in an ACARP project to produce a tool for this purpose. The tool was designed for rotary drilling. Its mode of operation was that torque, thrust and rotational speed would be monitored at ¼ second intervals whist drilling and when the tool stopped rotation it would take a survey and then go to sleep until rotation started again. The drilling measurements were proven to be extremely successful but at the time the survey tool had problems. While the latter have been overcome the impetus was lost and industry opposition to anything other than directional drilling is so great that it was not considered worth while to pursue. The work into this area is reported in Gray (1997) and Gray (2002). The technology can be extended to directional drilling but with some limitations. Any torque and thrust sub must be placed behind the down hole motor. Because of this the signal which comes from the cutting action of the bit is somewhat attenuated, at least in the higher frequencies. Another problem with using torque and thrust measurements is that to be useful the data needs to be transferred quickly and most data transfer systems are quite slow.

Another approach with directional drilling is to use geophysical probes to measure the parameters of the material being drilled. The use of gamma sondes to determine the proximity of the roof and floor seem to be in vogue for horizon control but would seem to offer very little to a miner trying to find a gouge zone. The use of resistivity sondes show promise for detecting larger structures but to detect the finer features it would seem that torque and thrust sensing is probably the best option.

In another approach to getting more information out of an open hole Sigra Pty Ltd built a borehole pressurisation tool. This device was designed to serve several functions. The first of these was to maintain pressure in a borehole during all aspects of the drilling process. This would enable fluid pressure to help stabilise the borehole wall. It was also designed to enable chips to be cut and circulated out of the hole without degassing having occurred. This offered the opportunity to collect a chip sample of coal under water pressure. The options existed to then use these chips to measure sorption pressure, gas content and diffusion coefficient under wet conditions. The tool was built and tested extensively on surface but the opportunity to test the tool underground never came and it has since remained unused. A report on the tool may be found in Gray (1998).

The fundamental problems associated with developing drilling systems for underground that will do anything different is that they require testing. If electronics are involved then the whole issue of intrinsic safety rears its head. If they are simply drilling developments then the problem is one of interfering with drilling production. This is exacerbated if the mine uses a contract driller. Here the issue is one where the contractor has a contract to drill not to develop some other group's technology.

5.8 Core Drilling Measurements

Core drilling seeks to preserve core taken from the central section of the borehole. This is the opposite of open hole drilling. Its use in detecting potential outburst problems is therefore quite different. Instead of looking for fine cuttings in an open hole it is known that in core drilling the fines will be lost. Therefore core loss is a good indicator of potential outburst problems whereas the ability to recover core is an equally good indicator that an outburst will not occur. This last statement can only be considered to apply to core drilling in the absence of fluid pressure in the hole, i.e. from underground. The core may be used for the determination of gas content and diffusion coefficient. Examination of core permits the detection of core loss and the detection of joints that frequently exist around faults even if the core is lost from the fault itself.

5.9 Incremental Flow Testing

Whilst not an indicator of outburst proneness in itself, a knowledge of flow distribution from within a borehole can be extremely useful in detecting areas of low or high flow. In holes up to 60 or 80 m length such tests can be conducted by advancing a single packer up the hole and setting it at increments of say 3 m and measuring flow. The flow per unit length can then be calculated as the slope of the flow versus distance curve.

When the holes get longer this technique loses sensitivity as incremental changes are too small with respect to the background flow. In this case there is a need to build equipment which permits a section of hole to be isolated and the flow measured from it. The equipment can then be shifted and the procedure repeated. The design of such a device has been undertaken several times but the industry has, to date, lacked the enthusiasm to pay for it or use it given that a full incremental flow test may take as long to conduct as the hole takes to drill.

5.10 Detection of Gas Filled Void Space by Geophysical Techniques

Concern exists about pore or void space existing in coals that contain gas at pressure. The reason for concern is that this is instantaneously available stored energy. While large voids are not present in coal seams the detection of dry gas filled gouge material would be extremely valuable. Potential exists to detect such structures by resistivity, radio imaging or seismic. The downside to all such geophysical techniques is the uncertainty associated with the measurement.

5.11 Summary - Most Useful Measurement Techniques

It would seem that the most useful measurement techniques to detect outburst proneness are those that identify the important parameters and permit the detection of structures that may lead to an outburst. In the Australian context we do not seem to have reached the situation where stress is as important a factor as gas and therefore we need to focus on the latter.

Given the proven use of coring it would seem that one avenue to advance the detection of outburst conditions would be to extend coring to a continuous practice and to conduct core desorption properly so that both gas content and diffusion coefficient can be measured. In addition there is the considerable benefit that core is available to be examined. It is important to take careful note of areas where core is not retrieved as this may indicate the loss of gouge material.

The second most useful approach would be to use open hole drilling techniques that enable monitoring of the cuttings and gas emitted from a borehole. If such techniques were combined with behind the bit torque and thrust monitoring then a good system could be put together to locate any structures.

Both the above systems could be regarded as short range sensing techniques to be used with rotary drilling up to maybe 200 m. When detection of conditions is required beyond these distances then recourse should either be made to surface drilling, which will not provide continuous in-seam information, or to more remote sensing techniques. The remote sensing techniques are logically tied to long hole exploration drilling. The scenario of a long hole being drilled with a geosteering package incorporating a torque and thrust sub and resistivity probe is not by any means unachievable. If the hole were drilled using a borehole pressurisation system then the prospect remains to recover undegassed chips of coal from which gas content, sorption pressure and particle size exists.

Whilst not explicitly referred to above the prospect always exists to take lump coal from the back of the miner sieve it to a reasonably coarse fraction and desorb it for a long enough period to determine the diffusion coefficient and initial gas content. The fact that a range of particle sizes would be taken does not prevent analysis as computational techniques exist to determine diffusion rate with mixed particle size distributions. While this technique does not provide any long range protection it does very effectively permit measurement of conditions at the face. This is important in the respect that if coal is being cut that is gassy, has been shown by drilling not to contain geological structure but does not have a diffusion coefficient high enough to pose any risk then mining could be allowed to continue.

6.0 Conclusions

The basic focus of this report has been on examining the potential energy release from outbursts. No attempt has been made to examine carefully how this energy would be expended in an outburst though quick calculations would soon indicate that potential energy releases of 0.5 MJ/m^3 of coal are very significant. According to the model developed, the bulk of this energy comes from expanding gas as opposed to strain energy of the strata.

A model has been developed that looks at the energy available from desorbing coal particles. The critical components to this model are:

- The gas content/pressure
- The diffusion coefficient
- The sorption isotherm
- The particle sizes being considered
- The potential size of an outburst

The first two of these factors can be measured by current techniques supported by improved observation assisted by some good mathematics. The sorption isotherm is readily measured though it takes time to do and is subject to variability of result. The sorption isotherm is by its definition the sorption characteristic of the coal at a fixed temperature. Also important is the moisture content of the coal wet coals absorb less than dry coals. Methane is generally generated as part of the process of coalification. It therefore displaces water and other gas in the microstructure. Sorption isotherms are measured by a reversible process which may not mirror reality in the field. This warrants detailed examination and the subsequent adoption of some standard but correct approach to measurement methods.

That leaves the issue of particle size to be considered. Particle size must be viewed as the size of the particles that can be formed during an outburst. Determining this is the subject of later recommendations. The potential size of an outburst may be determined generally by the volume of the geological structure that may be intersected.

The reader may note that I have generally steered away from issues of gas drainage. The reason for this is that in the energy model I have considered the state of gas pressure or content existing in a block of coal at a potential outburst site. Whether this pressure is virgin or has been arrived at after drainage has not been considered to be a matter for this report though techniques to measure permeability and low drainage have been discussed.

A huge amount of emphasis has been placed in this report on the theoretical mathematics of diffusion from solids. While it has been noted that this seems to hold for core desorption no rigorous testing of its applicability to coals has been undertaken. Such effects as that of moisture in the seam on diffusion need to be considered carefully.

In conclusion the relative figures of energy release for one cubic metre of coal with a gas pressure of 4.0 MPa, diffusion coefficient of 1x10-8 m^2/s, a free pore space of 1%, particle size of 1.0 mm, coal stress of 12 MPa, Youngs Modulus of 2.0 GPa and Poisson’s ratio of 0.3 are:

Strain Energy	0.16 MJ/m^3
Free Gas	0.07 MJ/m^3
Desorbing Gas	0.50 MJ/m^3
Total Energy	0.73 MJ/m^3

How this energy is expended in the failure is unknown. A significant amount of the strain energy is expected to be expended in breaking up the coal though a lack of knowledge on coal toughness prevents this from being estimated. Assuming all of this energy were able to be expended in projecting the coal then using the simple formula for kinetic energy of

$$E = \frac{1}{2} MV^2$$

Where M is the mass and V is the velocity

Then for a coal of density 1300 kg/m^3 we would calculate a velocity of ejection of 33 m/s. Even if only 1/10th of the potential energy could be converted into kinetic energy then the velocity would be 10.6 m/s which is still significant.

Particle sizing is clearly extremely important in the calculation of energy available from diffusion.

7.0 Recommendations

7.1 Basis for Establishing the Likelihood of an Outburst

That the industry changes from a one parameter measurement (gas content or DRI 900) to one that is based on potential energy release. Making this happen is not necessarily a major effort as many of the measurements required are already being undertaken. To be able to make this change will require some research though.

7.2 The Location of Structures Prone to Outbursting

The industry through ACARP has spent a small amount of money on the partial development of systems to detect outburst prone structures. If the industry really wants these to come to fruition it will need to spend a lot more both on their laboratory development but particularly on their testing in the field. These systems are principally associated with drilling and are described in Section 5.

7.3 Research Needs

From an energy release viewpoint the prime needs are to verify the diffusional characteristics of coal so that the model presented can be used with confidence or be amended to take account of different behaviour. This testing of the diffusional behaviour should specifically take into account the effects of moisture.

Determining the size of the particles that may be formed in an outburst is also an important research need. If small particles (less than 3 mm diameter) are not formed then the energy release is greatly diminished. Sampling of gouge material from potential outburst sites in various coal seams needs to take place. This material needs to be subjected to some form of mechanical breakage to see what size particles it will readily break down to. Such a test could be by simply gassing the coal up fully and then suddenly dropping the gas pressure. Any such test should be accompanied by the measurement of the gas release rate.

Determining whether outbursts will occur from solid coal also requires the assessment of what particle size may form. This requires some detailed thought but a test whereby the coal is fully sorbed with gas, stressed to near failure, and then the gas pressure is suddenly dropped is envisaged. This is shown schematically in Figure 27.

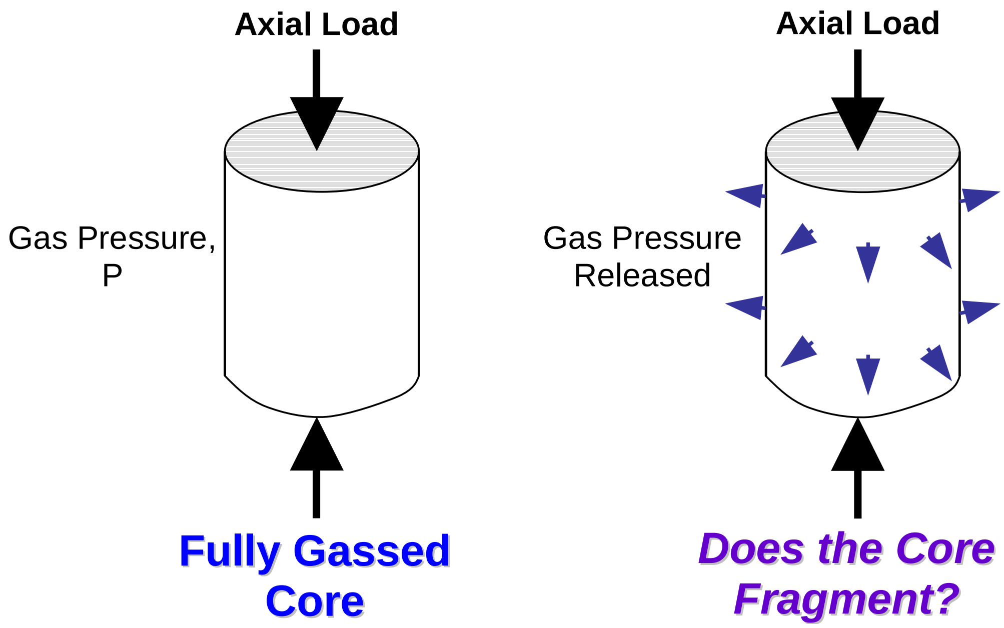


Figure 27: Proposed test for outburst proneness in solid coal

There is a need to arrive at a threshold energy release which should be considered to constitute an outburst. This should probably be arrived at theoretically, by comparison with energy release from explosives and by assessment of what has happened in real outbursts.

The topic of techniques to detect outburst prone structures has been previously discussed in section 5.

In the current booming economic climate persuading people to actually do the research will be difficult and will cost money. The work proposed requires good professionals and real effort. These people can earn a lot more using the knowledge that they have rather than in undertaking research.

8.0 Concerns

I am pleased to have presented here, something that may become the basis of a new approach to determining whether an outburst might occur. I can therefore see the prospect of (financially) interested parties fighting over the development of an Australian Standard embodying the findings of this report and any further research. Once an Australian Standard has been arrived at I have a worrying vision of it being applied rigidly so that someone ticks a box and does not think. Such an approach is wrong. It will lead to incorrect estimations of risk and may kill someone. There will always be a need for insight beyond any standard or procedure as geological conditions are infinitely variable.

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