INSIGHTS INTO THE ENERGY SOURCES OF BURSTS IN COAL MINES AND THE EFFECTIVE OF PREVENTION AND CONTROL MEASURES

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ABSTRACT: Coalburst is a general term, which is commonly used in the coal mining industry for the violent failures of coal in the ribs and face of roadways and panels in underground coalmines. Due to lack of interest in the industry to reveal the causing source of the event, or due to uncertainty about the source, they happily use this term. The term by its own does not reveal the source of the energy, which causes the event. There are three sources of energy that can cause a burst event in underground coalmines: 1) store elastic strain energy, 2) seismic events and 3) gas expansion energy. This paper presents the fundamentals about these sources of energies and discusses our known and unknown facts about the mechanisms. Additionally, it discusses the reliability and effectiveness of stress relief holes and gas exhaust holes as controlling measures to prevent burst events.

INTRODUCTION

Bump, pressure bump, pressure burst, coalburst and outburst are the terms which have been used to describe the events in underground coal mines which cause audible sound, ground vibration, violent failure of coal, and propulsion of coal and gas from the ribs or working face. The scale of burst events varies from very small coal spitting from a roadway's face to dislodgement and propulsion of hundreds tonnes of coal from the face or ribs. Apart from the source and mechanism of the burst events, they are considered a significant risk for workers and the financial success of an underground coal mine.

Recently ACARP provided several research funds through the coalburst task group to study the mechanism of the burst events in underground coalmines (Elvy 2015). SCT Operations Pty Ltd secured two funds as follows:

- Mechanics of gas related bursts in mining (C26060).
- Energy, burst mechanics required for coal bursts and energy release mechanics (C26066).

This paper presents an insight into the available energy sources to cause a burst event. It also discusses the effectiveness of the gas exhaust holes and stress release holes as two common preventing control measures.

AVAILABLE ENERGY SOURCES IN UNDERGROUND COAL MINES

When a burst event occurs, the broken coal from the ribs or face is propelled into the roadway or into the longwall working space. Figure 1 shows an example of a burst event in which the coal blocks were propelled from the roadway face and landed at distance around 4 m behind the face. If it is assumed that th blocks sat on the face as a free-body (with no resistance), the trajectory graphs of the block for various initial velocity can be calculated as Figure 2.

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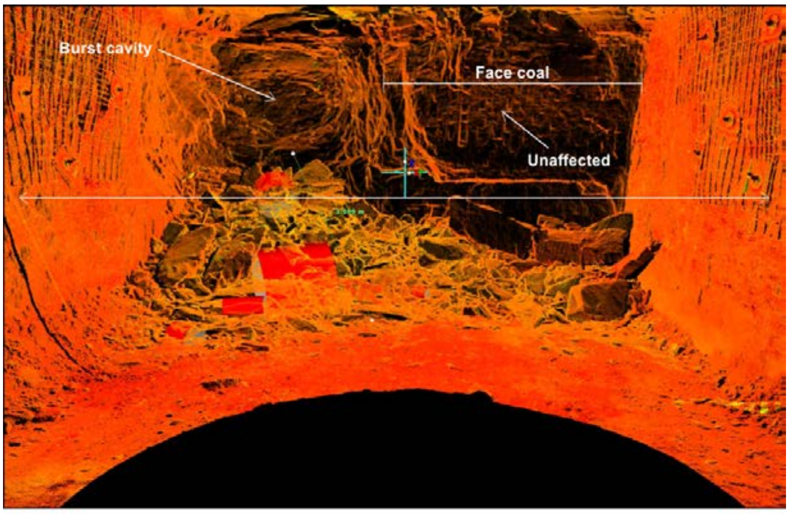


Figure 1: Laser scanner image of a burst zone

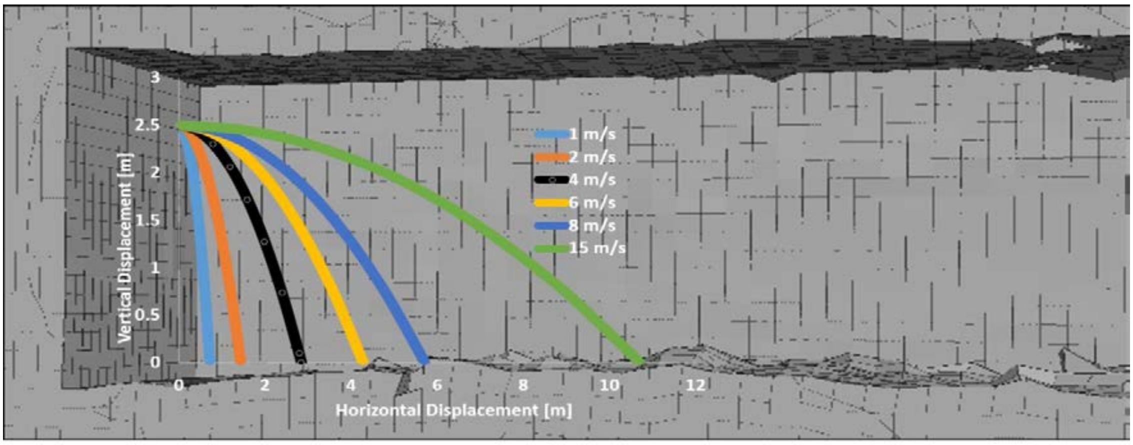


Figure 2: Displacement and trajectory of a free coal block sitting 2.5 m above floor on the roadway face with various initial velocities applied

The Figure 2 shows that it takes only 0.71 s for a free block of coal to hit the roadway floor if it is propelled from 2.5 m above the floor. The horizontal distance that the block travels depends on its initial velocity (velocity of burst). For a block to travel 4 m before hitting the floor, the initial velocity should be around 6 m/s. As previously mentioned, this velocity is required for a block of coal with no resistance. However the coal in the roadway face and ribsides carries the redistributed vertical stress which provides considerable confinement and resistance. Figure 3 shows 3D numerical model results of the vertical stress distribution at the face and right rib of a roadway excavated in Bulli seam geology whit the maximum horizontal stress is acting perpendicular to the roadway. Considering this fact, the available energy of the burst is required to firstly overcome the resistance provided by confinement (frictional resistance) then new fracturing needs to be generated to free the coal blocks. Finally, the remaining energy will turn into the kinematic energy, which controls the initial velocity of the blocks and propel them.

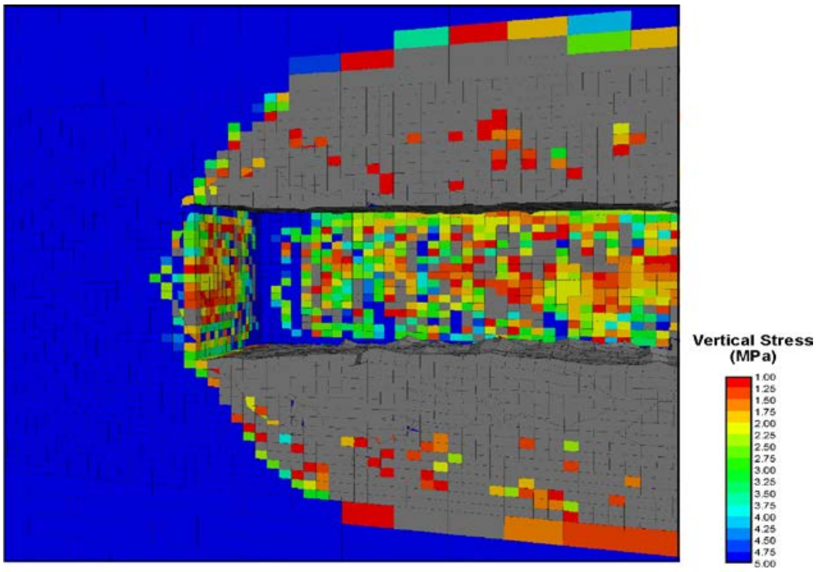


Figure 3: Vertical stress distribution ahead of the face

There are three sources of energy with the potential to cause burst event in underground coal mines; 1) stored elastic strain energy, 2) seismic energy, 3) gas expansion energy (Gray 1983; Gale 2002), which are explained in the following sections.

Stored elastic strain energy

When a sample of rock or coal is loaded under uniaxial stress condition, the external work done by the loading machine is stored in the sample while it deforms due to the applied stress. This energy is equal to the area under the stress-strain curve (Figure 4) and is calculated as follows and shown in Equation 1 (Jaeger and Cook, 1979):

$$\left.\begin{aligned} Elastic\ Energy\ (W_e) &= \tfrac{1}{2}\sigma\varepsilon \\ \sigma &= \varepsilon E \end{aligned}\right\} \rightarrow W_e = \frac{1}{2}\frac{\sigma^2}{E} \qquad \left(\frac{MJ}{m^3}\right) \qquad (1)$$

The energy stored in the rock unit and coal is related to the applied stress and the stiffness of the materials. The rock can be visualised as a spring, which is compressed by the *in situ* stresses. The stored energy is the amount required to have compressed the strata (spring) to the in situ state. For a unit volume of coal under *in situ* stress conditions, Equation 2 forms the following formula:

$$Elastic\ Energy = \frac{1}{2E}[(\sigma_1^2 + \sigma_2^2 + \sigma_3^2) - 2\nu(\sigma_1\sigma_2 + \sigma_2\sigma_3 + \sigma_3\sigma_1)] \qquad (2)$$

where E presents the elastic modulus of the rock or coal, ν is the poisons ratio, and $\sigma_1, \sigma_2, \sigma_3$ are the principal stress components. These equations show that for a constant stress fields, the magnitude of the energy stored in a softer sample is more than in a stiffer sample due to the increased deformation that occurs in the soft sample. On the other hand, the softer rocks such as coal, attracts lower tectonic stress compared to stiffer rocks, which can reduce the stored energy.

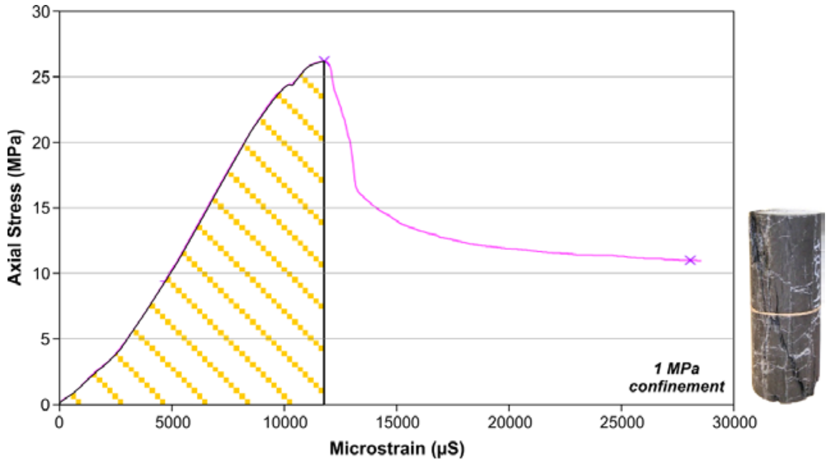


Figure 4: Elastic stored energy as area under stress-strain curve

Equation (2) was applied to a 3D elastic model of the roadways excavated in Bulli seam geology. The stress conditions were related to a depth of 500 m, maximum horizontal stress was related to a Tectonic Stress Factor (TSF) of 1.6 and the minimum horizontal stress was related to a TSF of 1. The stored elastic energy in coal obtained from this 3D modelling is presented in Figure 5. For the modelled stress conditions, the stored elastic energy in the coal ribsides is approximately 20 KJ/m^3 and it increases to a maximum of 45 KJ/m^3 at intersections where stress concentration occurs.

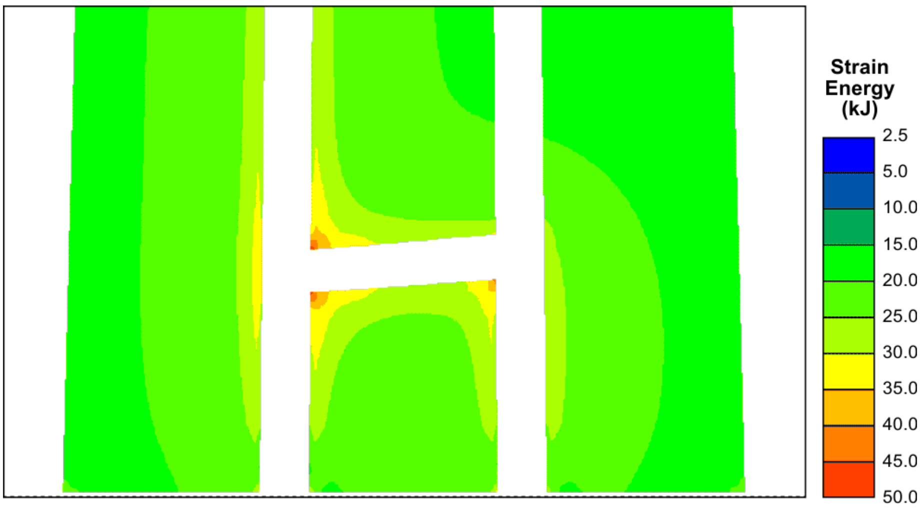
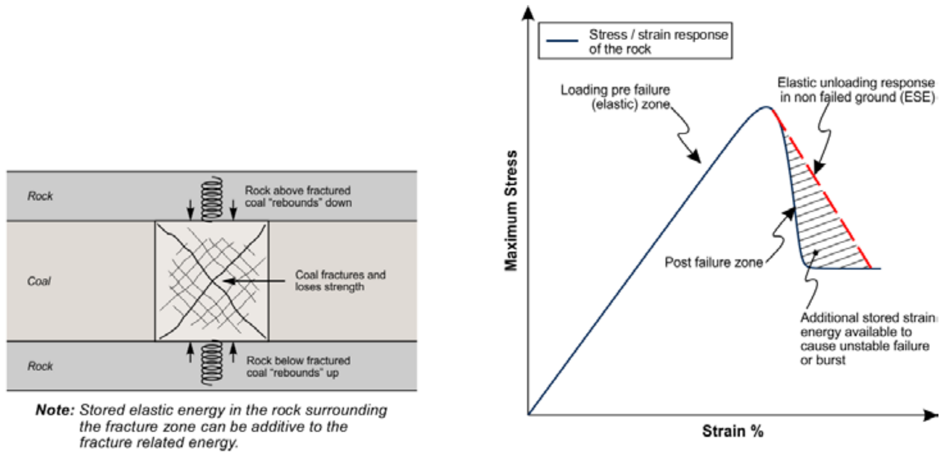


Figure 5: Stored elastic energy in coal calculated from 3D modelling

In addition to the elastic energy, which is stored in a unit volume of coal, additional energy is available from unloading of the roof and floor or surrounding coal in the failure process. The generalised concept is presented in Figure 6, which relates to the surrounding strata (or coal in a thick seam) as a spring which when unloaded by the failure of the coal, can rebound and provide additional energy to the system. This energy is commonly required to form macro fractures in the coal, however depending on conditions additional energy may be available which could contribute to the energy level required to propel the fractured coal.



a) Concept of stored elastic energy in the surrounding rock being released during fracturing of the coal.

b) Strain energy resultant.

Figure 6: Concept of additional stored elastic energy

An approach that takes into account the post failure stiffness of the coal can be used to calculate the additional stored elastic energy as follows:

$$Additional\ stored\ energy\ (W_{ae}) = \frac{stress\ drop^2}{2*(E_r - E_c)} \qquad (3)$$

where the stress drop is that at the time of failure of the coal, E_r is the unloading modulus of the surrounding rock and E_c is the post failure modulus of the coal (or rock units undergoing fracture). All units are in MPa and energy is in MJ/m^3.

This energy can contribute to the propulsion of the outside skin of a rib and face. The initiated burst process can grow further into the rib or face if the additional energy is sufficient. Since the magnitude of the stored elastic energy and available energy from the unloading of the surrounding strata are not significant, the scale of the burst initiated by them is small.

Seismic energy

The seismic events that affect mining activities can be grouped into mine induced seismic events and regional seismic events. The regional seismic events are generated due to regional tectonic activity and where mining production has no contribution to the occurrence of them. This paper discusses only the mine induced seismic events, which are directly initiated by the mining activities.

The initial stress in the ground is disturbed by excavation of roadways or retreating of longwall panels. When the magnitude of the new distributed stress exceeds the tensile or shear strength of the intact strata, the strata will fail and a portion of the energy stored in the volume of the failed strata is released in the form of seismic events. Additionally the redistributed stress can exceed the strength of the bedding planes, pre-existing fractures and faults. In these cases, the energy, which was stored across the rupture plan is also released and causes seismic events.

The energy released by a seismic event depends on the amount of stress drop, average displacement along the fracture, and the surface area of the stress induced fracture (Mendecki 2013). When rock fails, the difference between the initial stress to the residual stress presents the stress drop. As shown in Figure 7, the stress drop associated with tensile fracture is significantly less than shear failure. Also, for the shear failure formed in the tensile side of the strength curve (Zoorabadi et al. 2017) the stress drop can be significantly less if the minimum stress is compressive.

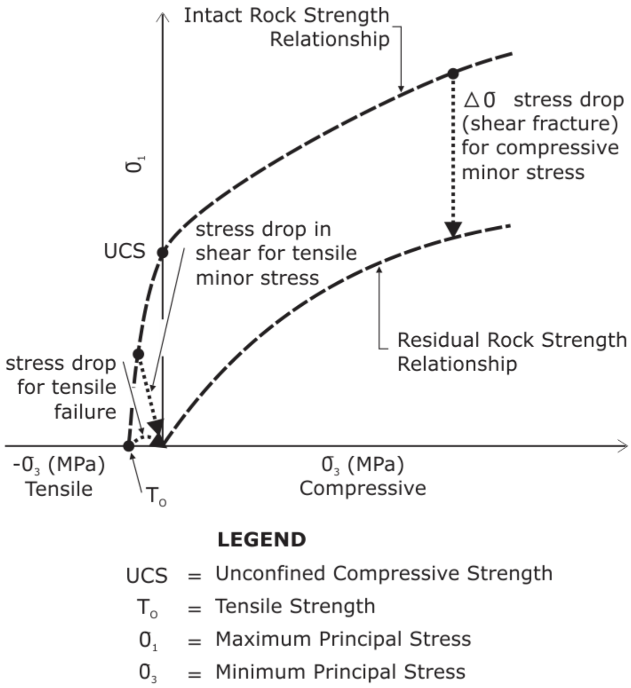


Figure 7: Stress drop associated with various failure modes (Gale 2002)

Considering the smaller shear and tensile strength for bedding planes and pre-existing structures such as faults compared to the intact rock, the stress drop associated with the shear or tensile failure of them will be smaller. Despite this, the surface area and displacement for pre-existing structures could be significantly higher.

The amount of energy released through forming a square shape shear fracture within an elastic medium is calculated by the following equation (Gale 2002):

$$Energy = \frac{\Delta\tau^2 \times L_f^3}{2G} \quad (4)$$

where, $\Delta\tau$ is shear stress drop, L_f is the fracture length, and G presents the shear modulus of the rock.

The energy associated with a tension fracture is estimated as:

$$Energy = \frac{\Delta\sigma^2 \times L_f^3}{E} \quad (5)$$

where $\Delta\sigma$ presents the magnitude of the stress drop due to tensile fracturing and E is the elastic modulus of the rock.

Several empirical formulations have been introduced by seismologist to calculate the energy of a seismic event on the basis of its magnitude. For this paper the following equation was selected (Denton P., 2014):

$$Log\ (Energy) = 4.8 + 1.5M \quad (6)$$

where M is the magnitude of the seismic event in Richter, and energy of seismic event is in joules.

Equations 4 to 6 were used to calculate the approximate fracture length associated with various magnitudes of the seismic events as presented in Figure 8. These results show that the magnitude of the stress drop, which is expected to be higher for shear failure under compressive confinement, has a significant impact on the energy and magnitude of the induced seismic event.

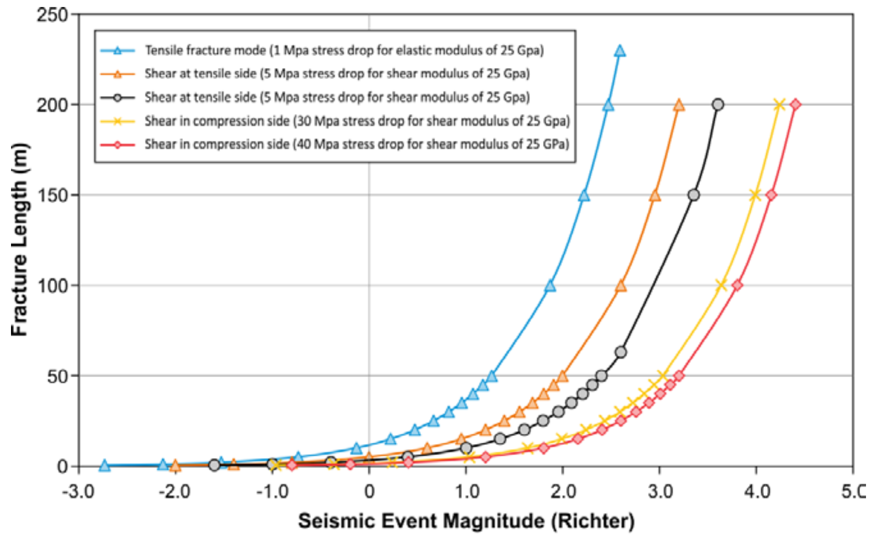


Figure 8: Fracture size versus magnitude of seismic event for various fracture modes

As previously mentioned, seismic events are induced by the sudden release of stored elastic energy throughout rock fracturing or reactivation of pre-existing structures which occur in surrounding strata. The mine induced seismic events are very common in both hard rock mining and coal mining. In coal mining, coal bump is a commonly used term which has been used for describe the events that generates audible sounds and ground vibration. In fact, the bumps are seismic events and they can dislodge fractured coals from the ribs or face into the working areas. Despite the limited number of publicly reported burst events induced by seismic events in coalmines, coal bumps have been regularly reported by deputies and operators in most of the underground coalmines. Therefore in contrast to the common view that coalburst induced by seismic events are very rare and isolated, they are occurring regularly in the coalmines.

A seismic monitoring network is commonly used in underground mines to locate the seismic events and measure the source parameters such as radiated energy, mechanism, corner frequency, magnitude, stress drop, etc. (Figure 9).

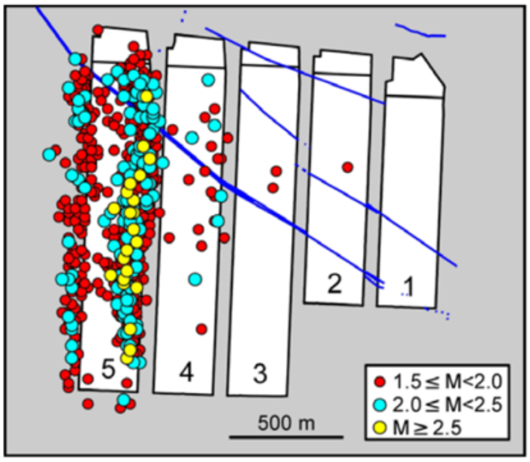


Figure 9: Microseismic events distribution recorded for a real case, M presents the event magnitude (Swanson *et al.,* 2016)

Peak Particle Velocity (PPV) is a commonly measured parameter, which is reported for each seismic event. The PPV is a function of seismic event magnitude (energy), distance between measuring point and seismic event location, and wave deterioration through wave travel from source to the measuring point. Kaiser et al. (1992) proposed the following equation to estimate the PPV from event magnitude on the basis of their experience from hard rock mining in Canada.

$$PPV = 1.4\frac{10^{M_r/2}}{r} \quad (7)$$

where PPV is in m/s, r is in m and M_r is the magnitude of seismic event. Kaiser et al. (1993) proposed PPV less than 50 mm/s as the threshold for no damage, falls of loose rock occur at 50 mm/s <PPV< 300 m/s, falls of ground for 300 mm/s <PPV< 600 mm/s and severe damages are expected to occur at PPV> 600 m/s. These thresholds are not applicable for all conditions and the induced damage depends on rock mass strength and characterisations of support system. The literature from hard rock mining cases (McGarr et al., 1981; Milev et al., 1999) show that when a seismic event reaches to the fractured rocks around the roadway, the PPV can be amplified up to 10 times compared with PPV measured in intact rock (Figure 10). The frequency of the seismic wave and thickness of the fractured zone control the amount of PPV amplification.

SCT Operations database for seismic event occurring in development show that the average event magnitude is in the range of -2.7 to -2. The general data appears to have a normal distribution relationship with a largest expected magnitude of approximately -1.25. It is also noted that there are a small number of events which were in the -1 to -0.25 range.

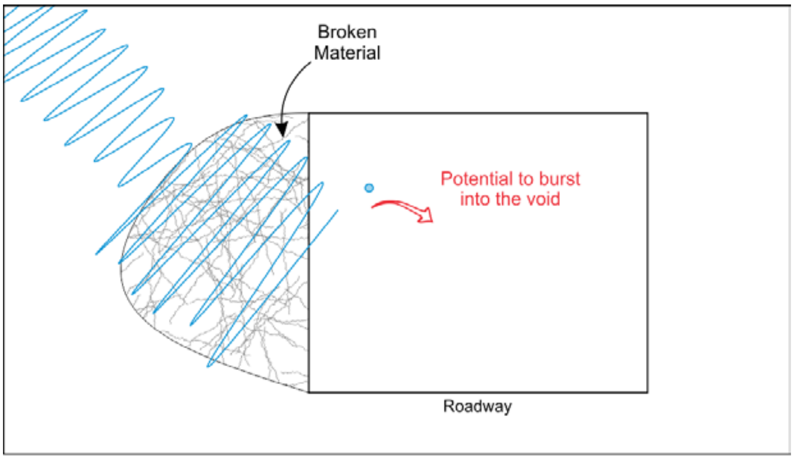


Figure 10: Concept of PPV amplification in broken ground

When a roadway is approaching a fault zone, the fractured and softened zone ahead of the face may push the stress concentration into the coal ahead of the face. Additionally the stiffness of the ground in a fault zones is usually lower than in surrounding ground. Therefore the redistributed stresses ahead of a roadway face may not transfer through fault zones as easily and may become more concentrated on one side of the fault zone (Figure 11). This stress concentration creates a shear stress on faults zones and can exceed the shear strength of the fault close to the roadway. If fault is located above or below a roadway, the stress release toward roadway reduces the normal stress acting on a fault which results in a drop in shear strength. Both mechanisms may induce a shear slip along the fault zone. In addition to this, the stress concentration ahead of the face caused by approaching a fault zone increases the rock fracturing and corresponding seismicity.

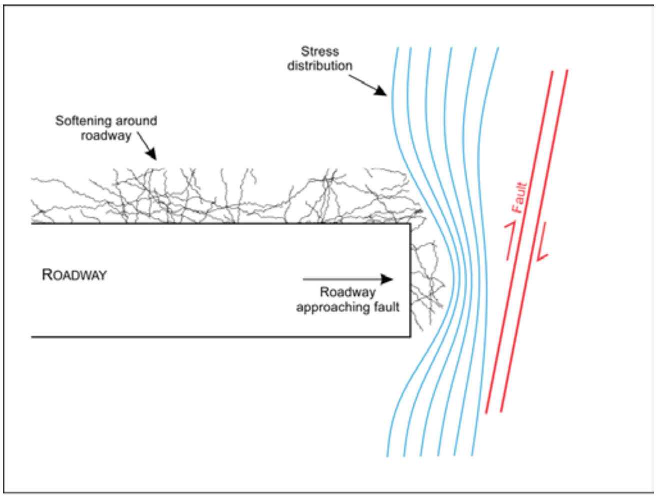


Figure 11: Stress concentration ahead of roadway face approaching a fault zone

Alternatively, considering the strain in the ground associated with the fault zone and its en-echelon nature, the stress in the fault zones may be elevated above the normal ground conditions. This is due to the start-stop nature of the fault and their varying throw along strike. Such a variation is likely to disturb the normal stress regime. Considering the possibilities to have elevated stress about faults, more seismic activity with relatively larger magnitude are possible for development approaching a fault zone such as the fault zones.

The mine induced seismic events through longwall retreat could have much higher magnitude compared with the roadway development. The average event magnitude for the longwall retreat varies between -1.5 to -1 with an upper expected of 0.5. However there is potential for inducing large events with magnitude 2-3.5 Richter for longwall retreat.

Figure 12 shows the PPV induced by seismic events with magnitude between -2 to 0.5 at various distance from the event location. Therefore a seismic event (or bump) with magnitude of 0.5 occurring at 5 m ahead of face is able to generate a PPV of 498 mm/s at the face. This PPV is has potential to cause a minor burst and fall of ground.

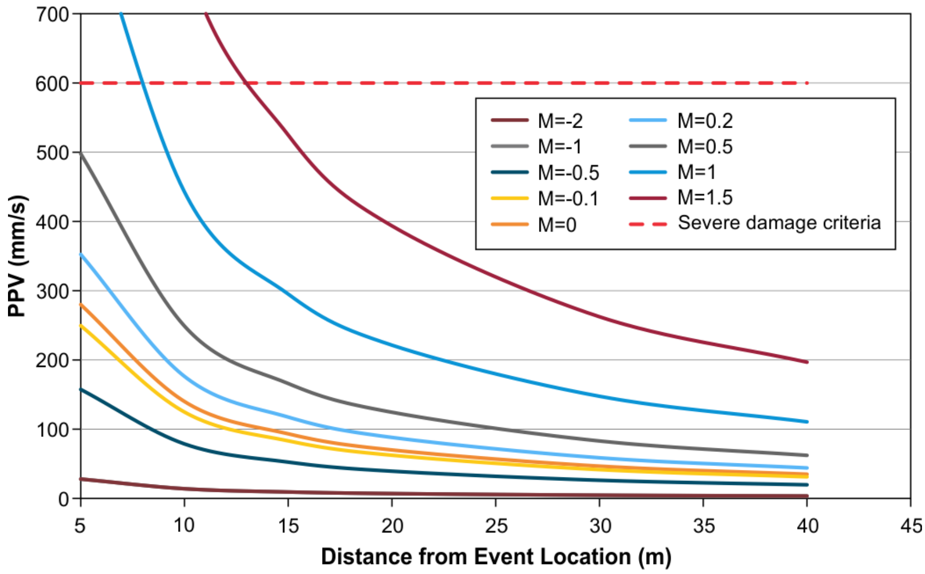


Figure 12: PPV induced by seismic events at various distance from event location

Gas expansion energy

Gases in coal are divided into two groups; 1) free gas, which exists in the pores and open cracks and forms only a small percentage (approximately 5-10%) of the total gas and 2) adsorbed gas. Most of the gas in coal is present in a sorbed phase on the internal surface of coal. The coal molecules are attached to these internal surfaces (pore space and cracks) as mono- or multi-layer. Since coal has very large internal surface area, it has high gas adsorption capacity.

The free gas is a function of coal porosity and gas pressure can be estimated by the following equation:

$$Q_{free\ gas} = \frac{273nP}{TP_a} \tag{8}$$

where, n is the porosity of coal, P is the gas pressure (KPa), T is strata temperature in degrees f Kelvin, and P_0 is the atmosphere pressure.

Coal with porosity of 10% and temperature of 20 degrees of Celsius, can have 1.38 m^3/m^3 of gas as free gas under a gas pressure of 1.5 MPa. The relation between pressure and adsorbed gas for a constant temperature is called isotherms and is presented by Langmuir's equation as follows:

$$Q = \frac{abp}{1+bp} \tag{9}$$

where, Q is the volume (m^3/t) of the gas adsorbed at a given pressure of p with dimension of bars, a is the Langmuir's constant representing the volume of the adsorbed gas (m^3/t) when pressure approaches to an infinite value, and b presents the Langmuir's constant with dimensions of 1/bars.

The amount of adsorbed gas depends on the rank of coal, gas type, pressure, moisture content, ash content, temperature. The gas absorption capacity of coal for carbon dioxide is almost 2.5 to 4 times that methane. Both stress and gas play a role in the process of the appearance of coal burst events involving gas expansion. The burst process is conceived as occurring in eight stages as follows:

1. In situ stage: Coal in front of the face is under triaxial stress condition and its strength is a function of the applied confined stress.
2. As the face approaches an area with high gas content (gas pocket) the water pressure reduces and the confining stress can reduce.
3. If the water pressure drops below the desorption threshold, then pore space and existing fractures (micro and macro) can fill with gas.
4. If the pore pressure is greater than the confining stress and strength of the coal then propagation of fractures and new fractures can form.
5. This allows more gas to be desorbed from the coal matrix and a greater volume of gas to be readily available to contribute to a burst.
6. If the volume of desorbed gas is higher that the volume of gas that can flow to the roadway then elevated pore pressures remain.
7. If the pressure developed is sufficient to push the coal from the ribside then a burst occurs. The strength of the rib is a control on the pressures required to initiate a burst.
8. Once a rib burst has been initiated the expanding gas acts as a means of maintaining energy to the coal being expelled.

In the final stage, all fractured coal is ejected from the gas pocket. The fracturing process can be propagated into surrounding coal but those areas remain in place and cannot be ejected. The fracturing of the surrounding coal increases gas desorption which is recorded as a spike in gas emission data. Gas expansion energy for adiabatic expansion for a pocket of gas with pressure of P1 to a new pressure of P2 (atmosphere pressure=101 KPa) is calculated as follows:

$$W_{gas} = \frac{P_1 V_1^{\gamma}}{\gamma - 1} V_1^{1-\gamma} \left(1 - \left(\frac{P_2}{P_1}\right)^{\frac{\gamma-1}{\gamma}}\right) \quad (10)$$

where, W_{gas} is the expansion energy of gas (J) P_1 is the initial absolute pressure (Pa), P_2 is the final absolute pressure (Pa), V_1 is the initial volume of gas, γ is approximately 1.3 for methane and carbon dioxide.

The above mentioned mechanism for the coal burst induced by gas expansion occurs in a short time which requires a considerable portion of the adsorbed gas being desorbed during that short time. The available gas expansion energy for 1 tonne of coal with gas content of 10 m^3/tonne and porosity of 10% for various desorption scenarios is presented in Table 1.

The available energy from gas expansion compared with other source of energy is higher if the desorption rate is high enough or if the volume of the free gas in pore space is significant. The probability of having these conditions is higher in the area close to geological structures. For example for mylonite in a fault zone, the volume of free gas is higher due to higher porosity and desorption rate is also higher because of smaller particle size. In addition to this, a barrier with low permeability is required to help gas pressure build up ahead of the face or in ribsides. Without this barrier the gas can drain through fractured coal with no pressure build up which is a main requirement for the gas expansion energy.

Table 1: Available gas expansion energy

Portion of desorbed gas to total gas [%]	Available gas expansion energy from free and desorbed gas (For gas pressure 1.5 MPa) [KJ]
5	388
15	806
30	1433
45	2060

EFFECTIVENESS OF BURST CONTROL MEASURES

Methods to control burst events during development includes stress relief boreholes, gas exhaust boreholes, cable bolting, physical barrier on the miner, and remote mining. The effectiveness of stress relief holes and gas exhaust boreholes are discussed in this paper.

Stress relief holes

Stress relief boreholes are used in a number of countries as a means of minimising the potential of coal bursts in high stress conditions. Boreholes act similar to excavations in rock and they disturb stresses and impose a new stress distribution. If the stress around the borehole exceeds the strength of coal, a fractured zone is formed. The broken coal will reduce stress based on its residual strength characteristics. Therefore, if fracturing occurs there will be some stress relief about the borehole. If fracturing does not occur, then there is no vertical stress relief adjacent to the borehole, rather an increase in vertical stress.

This process was investigated through numerical modelling. In the modelling, a borehole with diameter of 50 mm is drilled in coal with Bulli seam geotechnical properties at a depth 500 m and tectonic stress factor (TSF) of 0.5, 1, and 1.6. The principal stress if vertical and the horizontal stress is varied on the basis of tectonic stress factors. Figure 13 show the failure modes and extent of the failure zones for each TSF value. Shear failure and tensile failure and reactivation of these are the failure modes of the fractures that form around the borehole due to stress concentration.

Despite the different fractured zone extent for various TSF values, the fractured zones only extend 2R (R is the radius of the borehole). Therefore the stress released zone around the each borehole is limited to only two times of the borehole radius. These results show that a large number of closely spaced boreholes would need to be drilled in a roadway face to release stress concentration ahead of the face. These results are in accordance with drilling practice and experience in creating a stress relief slot about an excavation.

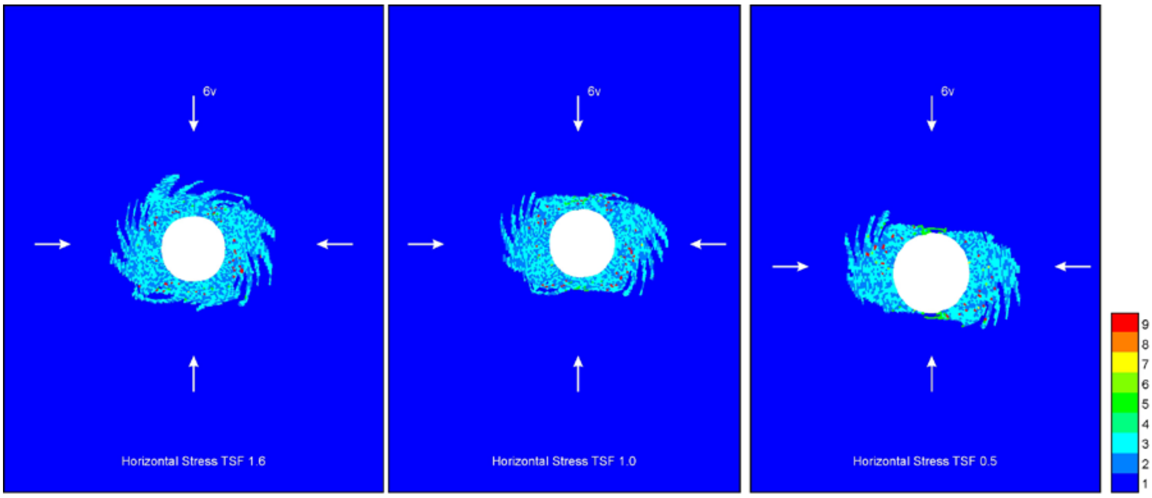


Figure 13: Damage and stress relief zone about a borehole under a variety of horizontal stress conditions

Gas exhaust boreholes

Gas exhaust boreholes are drilled into the roadway face to act as high conductive pathways for desorbed gas to be drained from the face without building pressure in pore spaces. This method has been used for several decades in Europe and Australia, but its efficiency is open to question. There is mixed experience with exhaust holes, which is most probably due to the variation in desorption rate and location of the gas source for the cases where it has been used.

The efficiency of gas exhaust borehole was investigated using 3D numerical modelling. For this model, a gas pocket was assumed to exist 1.5 m into the face of a roadway developed in Bulli Seam geotechnical conditions. A typical adsorption isotherm for Bulli coal was used for this modelling. For the purpose of this assessment the desorption isotherm is assumed to be the same as the adsorption isotherm. Gas content was assumed to be 10 m^3 which is equivalent to 1.5 MPa gas pressure according to this isotherm.

The gas desorption mechanism was implemented in a 3D numerical model of the roadway, where it was assumed that only 20% of adsorbed gas is able to desorb at early stages of desorption. Figure 14 shows the burst caused at the face due to gas pressure. This analysis shows that a gas pressure of 1.5 MPa is sufficient to burst the roadway face under modelled seam geotechnical conditions. This is not considered to be the pressure threshold for a burst but a value used for the assessment.

The efficiency of the gas exhaust boreholes to eliminate or downgrade the severity of a coal burst was investigated by modelling five boreholes drilled into the face with approximate spacing of 1 m. The boreholes locations are presented in Figure 15a. The results were not significantly different from the previous model, where the gas pocket is located 1.5m behind the face and gas content is 10m^3/ton. The gas flow is shown in Figure 15b and shows that although pore pressure along the exhaust borehole has dropped significantly, their efficiency to take all desorbed gas out of the face is compromised by the ability of the gas to transport between holes.

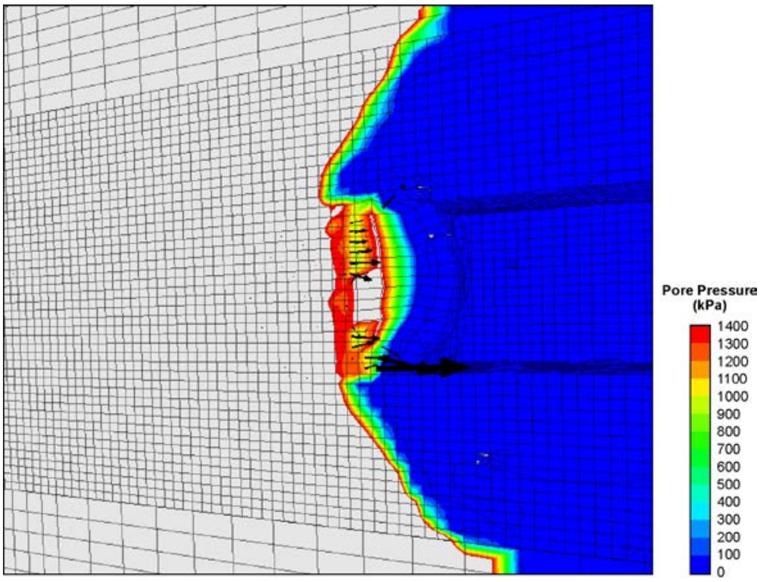


Figure 14: Slice of the model showing the pore pressure and burst zone

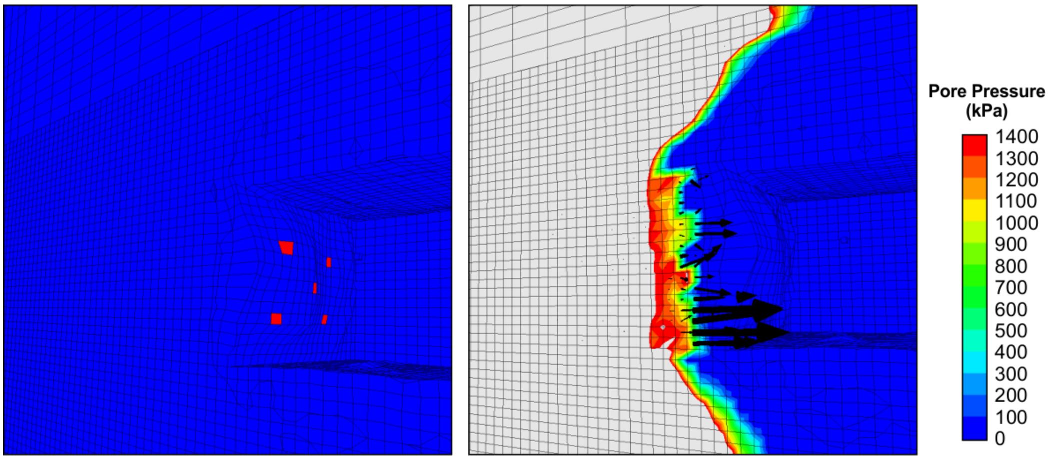


Figure 15: Location of boreholes in the model and gas flow through boreholes

The conductivity of the fractures intersecting boreholes is very important in allowing gas to migrate in a pre-burst time zone, but in the instance of a burst the gas migration is too slow to allow the boreholes to exhaust the gas. The modelling indicates that if gas desorption or diffusion takes place at a modest rate, then boreholes can drain gas and minimise pressure build-up. If the volume of free gas is the main driver of gas expansion energy (Mylonite zone), the gas exhaust hole can drain the free gas ahead of the face. However, if gas desorption occurs at a very fast rate (which is typical of bursts), then such holes may not move a sufficient volume of gas in time.

CONCLUSIONS

Bursts events in underground coalmines are risks for the safety of workers and the financial success of projects. Although the Australian coalmine industry previously used different terms such as bump, pressure bump, coalburst, and outburst to report the events, but the term coalburst is becoming the commonly used term in recent years. This term on its own does not reveal the mechanism of the burst and the driven source of the energy. SCT Operations managed two ACARP projects to investigate the energy and mechanism of bursts. This paper discussed the available sources of energy for burst events in underground coalmines. The stored elastic energy, seismic energy and gas expansion energy were discussed and the methods to quantify the available energies were presented. Research on this topic is not yet complete and so this paper presents the current understanding and knowledge obtained through these research projects.

In addition to this, the effectiveness of stress relief holes and gas exhaust borehole and control measures were discussed in this paper. The 3D modelling showed that the stress relief holes only release the stress in volume of surrounding coal within two times the hole radius. Therefore a large number of closely spaced boreholes would need to be drilled at the roadway face to release stress concentration ahead of the face.

The 3D modelling of gas exhaust boreholes also showed that the exhaust holes are only effective where the conductivity of coal is high enough to allow the gas to drain quickly through holes.

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