GEOTECHNICAL EXPLORATION FOR UNDERGROUND COAL MINES

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Abstract
This paper describes the steps and testing that should be adopted for geotechnical purposes during exploration for a new coal mine or an extension to an existing one. It covers the field operation involving drilling, core logging, geophysics, stress and fluids measurements. It also covers the testing of core. In the latter particular attention is paid to the effects of anisotropy of rock properties.
It takes this information and describes how it may be usefully used in mine design so that such problems as periodic weightings, gas outbursts, excessive gas make and pillar failure may be recognised and avoided.
Some cases of where things went wrong and where things were done correctly are presented, along with a discussion of what might be done to avoid them in the future. Finally, it concludes with observations on what research still needs to be done to ensure good geotechnical mine design.

Introduction
Exploration for a new mine is very much dependent on the size and nature of the mine. At one extreme, it may be a large multi-seam longwall. At another, it might be a mine which is intended to produce a boutique coal destined for some niche of the chemical market and which is only found in a limited area that has been affected by intrusions. A mine might also be to supply a local power demand. Each mine requires exploration to prove the resource and then a mine design to define the reserve. This mine design cannot be arrived at without a geotechnical assessment which will include gas and groundwater. Getting the design of this aspect of exploration correct is extremely important to the long term economics of the operation.

The depth of the coal seams to be mined is always critical as it affects virtually all aspects of mining. Just as important is whether a multi-seam development is planned. Following this, decisions must be made on the entry and development of the mine. The entry may be shaft or drift and is more commonly both. In some cases, it may be advantageous to develop much of the mine in rock, thus avoiding gas problems, and having what might be improved stability while utilising in-seam development purely for extraction panels. The seam thickness and its variation will have a large effect on mine design. So too will the location and nature of faults. Longwall operation is particularly sensitive to the latter two. Bord and pillar operations have different requirements depending on whether the pillars are extracted by some method or left in place.

With each mining option, the geotechnical factors need to be considered. The means for this exploration is via boreholes, which may or may not be cored, by geophysics from within these boreholes and by surface seismic surveys and other surface geophysical techniques. An examination of surface features and constraints is essential.

What Needs to be Determined?

Mine design is focused on removing the maximum amount of the resource that it is economic to achieve within the bounds of safe working practice, environmental constraints and legislative requirements. Unlike construction of civil works, the general population does not care about the aesthetics of an underground mine, though they may care very much about the subsidence or what the development does to their groundwater and surface water. This poses particular constraints but also provides a great deal of freedom as to how mines are designed. This design can be focused on meeting the purpose of getting the resource out of the ground as cost effectively as possible.

To be able to undertake a mine design, the following grouped questions need to be answered:

Rock mechanics questions
What will be required to prevent the roof falling in?
Will the floor remain intact?
What pillar sizes will be required?
How will the goaf form – not at all, sub critical or super critical?
What is the risk of rock bursting or coal bursting?

Gas related questions
What gases are present?
Is there a risk of outbursting?
What gas make will occur during mining?
Will gas pre-drainage be required?
Will gas post-drainage be required?
What gas is in overlying sediments?

Groundwater related questions
What is the regional groundwater system(s)?
What will happen to the regional groundwater?
What needs to be done to keep the mine dry?
Will the groundwater corrode rock supports?
Will bacterial action within groundwater occur?
How can the extracted groundwater be disposed of in an environmentally acceptable way?
How does groundwater relate to surface water flow and storage?

Spontaneous combustion
Will spontaneous combustion occur?

Rock Mechanics

The key question in the rock mechanics or strata control of mining is: “what ground disturbance is permitted to occur?” This, along with groundwater constraints, will largely determine what type of mine design is permitted. Within the underground coal mining context, this potentially means a bord and pillar mine which does not lead to any subsidence. It could also be a bord and pillar mine with pillar extraction where some subsidence will occur. If a longwall extraction process is used, it could be of sub-critical width so the broken super-incumbent strata does not break or bend to surface. If it is of super-critical width, subsidence will extend to surface bringing with it all the complications of changing surface levels on infrastructure and drainage (Yang et al, 2019).

In the case where subsidence is designed to not occur, the pillars that are left behind must be stable and take the increased stress brought about by removal of a fraction of the coal seam. In this case, the remaining coal in the pillars becomes a very significant fraction of the resource as loading increases with depth.

Where subsidence is permitted to occur, the resulting surface profile should be as even as possible. In the case of super-critical width longwall systems, this means the use of yielding pillars that do not leave ridges on the surface. These, by their definition, do not carry large loads and crush in a controlled manner. Whether they will do this depends on their geometry, the nature of the roof and floor rock, and the reinforcing of the pillars. The use of yielding pillars is particularly relevant to multi-seam mine design. Here, stress concentrations related to the pillars left in an adjacent mined seam may play havoc with mining an adjacent seam. (Tadolini & Zhang, 2007).

The alternative to yielding pillars is wide pillars. Conventional pillar design leads to these becoming very wide with increasing depth (Gilbride & Hardy, 2004). Non yielding pillars may carry significant internal stresses that are a function of the pre-existing stress state and the stresses brought about by mining. Cases exist where pillar bursts in very wide pillars occur (Kennedy, 2008). This is a function of the pre-existing horizontal stress in the pillar, the stress developed through mining and the material properties of the coal, and the coal to roof or coal to floor rock contact behaviour. Current pillar design practice ignores the pre-existing stresses within the coal. This is a serious error. Where pillars are stressed vertically to a high degree the risk of the pillar punching into the roof or floor exists as shown in Figure 1.

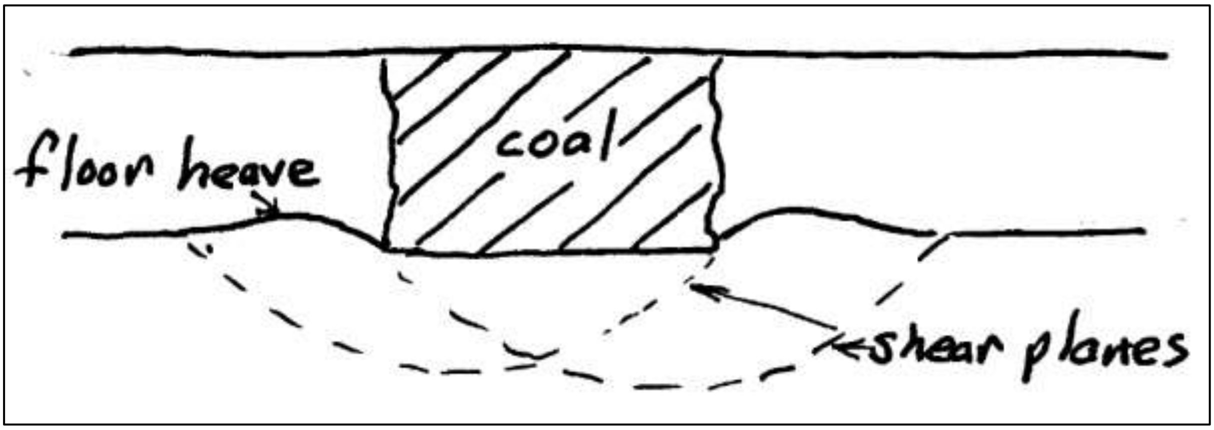

Figure 1. Floor heave associated with pillar punching into the floor

Where any rock is removed, the stress that it did carry is shifted to some other part of the rock mass and increases stresses there. It may lead to the rock shearing or failing in tension. This failure may lead to local de-stressing, in which case, further stress re-distribution occurs. Determining the loading combination at which the rock will fail is critical to any rock mechanics design.

Because coal mining is conducted in sedimentary rocks that are deposited as a series of laminations, their behaviour tends to be highly anisotropic. This anisotropy affects both the pre-failure behaviour of the rock and that at failure. A typical laminated siltstone may have a Young's modulus which is 1.5 to 2.5 times stiffer in the direction of the bedding plane compared to that perpendicular to bedding. Its tensile strength is likely to be five times greater parallel to bedding compared to that that perpendicular to it. Its resistance to shearing is likely to be much less along, compared to across, bedding. If we use a Mohr-Coulomb failure model, this may be divided into terms for cohesion and friction angle. The cohesion on a bedding plane may be one tenth that across bedding. Highly laminated rocks tend to de-laminate either in the roof or the floor leading to slabs falling out. This is exacerbated by lateral stresses which tend to cause buckling of the slabs once delamination commences. The typical anisotropies of a laminated sedimentary rock are shown in Figure 2.

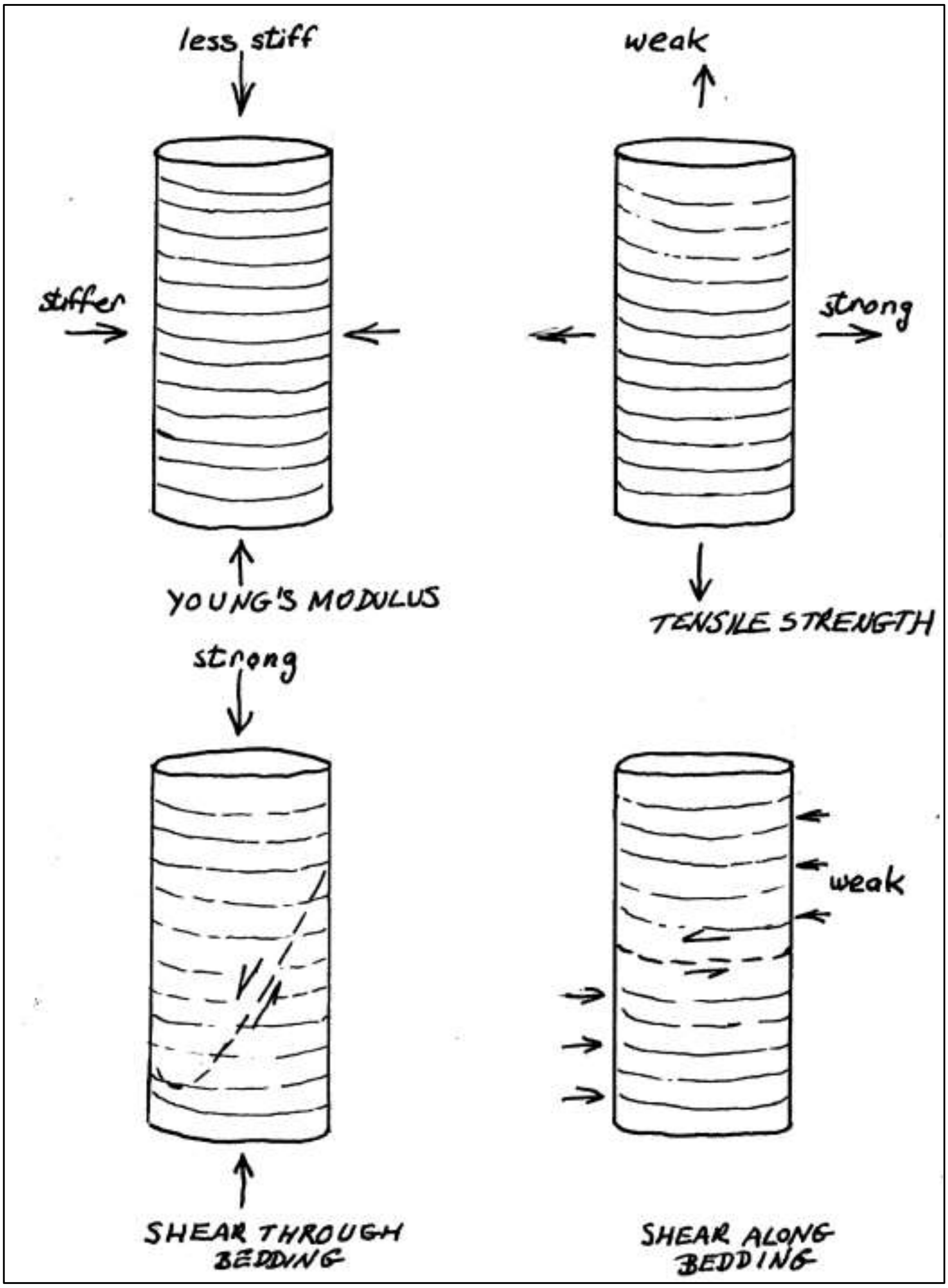

Figure 2. Typical anisotropy in a laminated sedimentary rock.

Some of the laminations in sedimentary strata are actually very thick. It is possible to have relatively homogeneous sandstones which may be several if not tens of metres thickness. This rock will behave in an entirely different way to highly laminated rocks. While having a good strong sandstone roof is highly desirable during mine development it can pose very significant problems in goaf formation. The problem is that it will tend to remain intact and then suddenly fail. The failure mode may be lateral compression in which case it can be explosive. It can be in tension at the top of the massive unit. In this case, a large block can collapse causing a wind blast and imposing huge loads on the powered supports of a longwall (Latilla et al, 2007; Sandford, 1998; Sandford et al,1999). Such sudden failure may cause a seismic event that leads to rock, or coal, bursting in the mine roadways. Failure in shear may also occur.

Where massive strata, with its associated risks exists, it is necessary to pre-condition the rock mass to make it break up in a more controlled manner. This may be accomplished by blasting or by hydrofracture (Lu et al, 2014). The desirable orientation of hydrofractures is parallel to the seam. It is very much a second choice to have them propagate vertically and parallel to the longwall face. Other fracture orientations do not assist caving. As hydrofractures propagate perpendicular to the direction of minimum stress, this needs to be determined before its use.

While the consequence of seam removal is generally thought about in terms of vertical displacement through the broken stratigraphic sequence above, the effects of horizontal displacement are just as important. Such displacement occurs at surface, but also through the strata below. This occurs because of the lack of constraint at the goaf edge which allows the adjacent strata to expand laterally. The effect may extend far beyond any subsidence (Ross et al, 2006; Mills et al, 2015, Gray & Gibbons, 2020). In the underground workings, the expansion of rocks laterally towards the goaf can lead to severe racking of the gate road pillars, particularly on the tailgate side (Tarrant, 2005).

Problems of roof and floor failure are similar except that roof failure has gravity to assist it while floor failure is constrained by gravity. Roof failure can be brought about by excessive stress, or by too little stress. In the case where the horizontal stress is low, the frictional restraint on blocks dropping out is low. Indeed, tensile stress may be developed at the bottom of the roof beam. Where stress is high, the risk is that it will induce shear in the roof rock or assist the development of delamination and buckling.

In the floor where gravity assists the floor rock to stay in place, the risks are more of floor heave or floor break up. The latter is made much worse if the rock is prone to slake.

Rocks may behave in several ways not yet discussed. One is plastic deformation, the second is nonlinearity and the third relates to how fluids act within the rock mass (Gray, Zhao & Liu, 2018a & 2018b). The more weakly-cemented rocks may show plastic deformation. This may be of significant magnitude and lead to stress redistribution. Many sedimentary rocks behave in a quite non-linear, but still elastic manner. Most show some minor plastic deformation, elastic non-linearity, elastic anisotropy and significant anisotropy associated with their tensile and shear strength.

Fluids at pressure provide a force which acts on any open surface within the rock mass. This may be the walls of pores or fractures of any scale. The action of fluid under pressure within an open joint may be readily calculated as the product of the open area of the joint and the pressure acting on it. The effect is directional. The effect of fluid in rock which is not obviously jointed is similar but a little more difficult to determine. The approach is to see how the rock deforms under fluid pressure compared to how it performs under external pressure. The ratio of the two is described as the poroelastic coefficient of the rock. This is also directional and in coals can be related to cleating. The poroelastic coefficient varies from zero to unity. This has very important effects on rock behaviour as fluid pressure will have a far greater influence on rock with a high poroelastic coefficient. It should also be borne in mind that poroelasticity changes with stress and stress history. As the effective stress decreases the poroelastic behaviour tends to increase, thus further driving down the effective stress. This can lead to a very rapid weakening of the rock mass as the strength component derived from friction is dependent on the normal effective stress to any potential failure plane (Gray, Zhao, Liu & Wood, 2018c).

This general discussion on coal mining rock mechanics highlights the need for:

a. Understanding the geology so a geotechnical model may be developed
b. Rock stress measurement
c. The measurement of rock elastic properties taking into account anisotropy and nonlinearity
d. The measurement of the rock failure behaviour and how it is influenced by anisotropy. Here the key measurements are:
    - shear strength – cohesion and phi (ϕ)
    - tensile strength - perpendicular to and parallel to the bedding
e. The measurement of rock plastic behaviour
f. The effect of fluid pressure within the rock mass
g. Slaking behaviour

Gas Matters

Gases pose many problems for underground coal miners. These include toxicity, flammability and the risk of outbursts.

From the viewpoint of both toxicity and flammability, the type of gas is important. If the gas contains hydrogen sulphide, the issue of toxicity will be paramount, followed by the effect it has on corroding the mining equipment. If the gas contains carbon dioxide, then this is also toxic in higher quantities, though not flammable. Methane is not toxic but will burn or explode. Sometime higher hydrocarbons may also exist in seam gas. Though it is frequently ignored, nitrogen may also be an important component of seam gas. Generally, there is a gas mixture which will vary over the mine lease, and care must be taken to determine its distribution.

The flammability of the gas mixture in air is a function of the gas types that are present. If the workings become filled with too much gas, an explosive mixture may result. Determining how much gas will enter the roadway and the development face is one of the important tasks in mine design. To do this requires the parameters determining gas release from the solid coal and that of cut coal to be determined. In cases of high gas content and high permeability, pre-mining gas drainage may have to be implemented because the flow from the solid coal exceeds the capability of the ventilation system. Drainage diverts gas from the workings and lowers the gas content so that mining may progress with suitable ventilation.

If a high gas content exists and the permeability is lower, the risk of outbursts becomes higher. Outbursts are caused by failure of the coal followed by the sudden expansion of free gas held in void space within it. This sudden expansion is almost instantaneous, though the total outburst process may itself not be (Gray et Al, 2021). The largest outbursts invariably come from faulted areas where broken coal exists and which are breached by the mining process. These faults may contain void space in which free gas exists or void space generated during the outburst. This may be by dilation or by the expulsion of coal and rock followed by the formation of a temporary plug. Gas flows into this void which becomes pressurised, following which there is a failure of the plug and an expulsion of gas and coal. This process may occur in several stages and has led to outbursts of more than 10 000 tonnes of material and a million cubic metres of gas (Wood & Gray, 2015).

Outbursts may be of much smaller size involving the erosion of a small quantity of gouge coal at quite low gas pressures or the failure of a jointed mass of coal which ejects blocks. The latter is likely to be far less serious as the void volume within the joints is limited and the potential for a void to form, plug and then fail is extremely unlikely.

During longwall retreat mining the key gas related concerns are caused by excessive gas production in the area of the face. This may come from the solid coal draining into the intake gate road or the face. It will be augmented by gas from the goaf and the surrounding strata, as well as the component of gas coming from cut coal, which may be large. Keeping the gas on the face down to safe levels is crucial to mining. Determining what gas will be released from the solid depends on the gas content and pressure, the permeability of the coal, the thickness of the seam and the diffusion of gas from the solid into the fractures that make up permeability. The volume of gas coming from the goaf is determined by the coal that is left in the goaf and that existing in other relaxed strata. In some cases, the free gas may also exist in pore space within sandstones or by adsorption in other carbonaceous rocks. The desorption rate from cut coal depends on the diffusional behaviour of the coal. This is a function of the gas content of the coal, the size of the cut coal and the diffusion coefficient of the coal lumps.

Once a goaf is formed, the questions are how quickly will it become inert both from the viewpoint of explosion and spontaneous combustion risk. This depends on the gas available to fill the goaf.

The key terms that have been used in this discussion on gas which need to be determined are:

- Diffusion
- Gas composition
- Gas content
- Gas pressure
- Faults
- Jointing (including cleating)
- Gouge
- Permeability
- Void space

The relationship between gas pressure, gas composition and gas content needs to be described further. Different gasses are stored in coal by what is generally considered to be a process of mono and multilayer adsorption within the coal's internal structure through the action of Van der Waal's forces. They compete with each other and with water molecules, for sites to bond to. This behaviour is complex. In addition to the adsorbed species, storage exists in void space within the coal. This void space may be in fine pores or in fractures. The pressure at which the gas is adsorbed may be less than the liquid (water) pressure within the coal. Under these circumstances, the gas is considered to be fully adsorbed in the coal with some also stored in gas saturated water held within the pore space. In some coal seams the pores are not filled with water but at least in part with gas. These seams behave somewhat differently to those which are initially water saturated.

The relationship between the sorbed gas and the gas pressure is called the sorption isotherm. The term isotherm here refers to gas storage at a specific (seam) temperature.
While isotherms may be re-created for a single gas by testing in the laboratory, the value of these is always dependent on the moisture content of the coal. What should it be, as received from core, or held at some nominal value? The answer is not clear. Neither is it clear as to how to combine isotherms from several different gases. Two main theories exist. These are the Ideal Absorbed Solution Theory and Extended Langmuir Theory. Both these can lead to odd distributions of sorption pressures which do not bear much resemblance to those actually measured. The only means to determine the sorption isotherm with confidence is by measuring the isotherm directly on desorption. This is called the Native Sorption Isotherm (Gray et al, 2015).

Coals are complex, comprising varying layers of dull and bright coals which may contain mineral matter (ash) and are frequently traversed to a greater or lesser degree by fractures called cleats. These may be open or filled with minerals or clays. The openness of these cleats is highly dependent on the effective stress that exists across them. The effective stress is a function of total stress, fluid pressure and the poroelastic coefficient of the coal. Permeability is highly dependent on effective stress. Indications are that permeability may vary by an order of magnitude with two MPa variation in what is thought to be effective stress. This variation is huge.

Coal has another behaviour that affects the total and effective stress; this is the tendency of the coal to shrink as it gives up gas. If a coal is being drained of fluid which is initially water the fluid pressure will drop and the effective stress will increase. Once the pressure in the seam drops below the sorption pressure, gas desorbs and the coal begins to shrink. This shrinkage leads to a reduction in total stress which is controlled by the degree of shrinkage and the stiffness of the coal. The higher the Young's modulus, the more shrinkage will reduce stress within the coal. The change in effective stress will depend on the change in total stress and the change in fluid pressure. If the effective stress increases with drainage then permeability can be expected to decline while if effective stress decreases (due to shrinkage effects) then permeability will increase. Determining which effect will take place and to what extent is extremely important to drainage design (Gray, 2011).

In between the cleats of the coal the flow of gas through coal is assumed to be by diffusive flow, often modelled as Fickian diffusion though it may also include Knudsen diffusion and is likely to include other complex components. If the cleats are widely spaced and the diffusion coefficient is low, then the gas will diffuse slowly into the cleat structure. This may be the rate-limiting step in drainage. Diffusion is also important as it affects the speed with which coal lumps on the armoured flexible conveyor or panel conveyor release gas. The question here is not only the diffusion coefficient, but the size of the lumps. Is the relevant size the lump or some smaller size related to the cleating? Coals with high diffusion rates appear to be more prone to outbursting as they can supply gas to cleats or voids at a greater speed (Wood & Gray, 2015). Diffusion alone cannot, however, supply sufficient gas to drive an outburst without some void space which will pressurise. This is covered in detail by Gray et al (2021) and along with it the reasons why the Apparent Diffusion coefficient from core and the IDR 30 index have value but the Desorption Rate Index (DRI) approach does not. The latter two indexes are used by the commercial laboratory, GeoGAS.

The presence of faults and the nature of these is of extreme importance to any mining in coal. Faults contain broken material and voids. The determination of the extent of any likely faulting and the nature of the gouge material associated with it is of considerable importance to mining and outbursting. The same applies to the distribution and nature of joints in the coal.

Groundwater

Any mine will change the groundwater situation. The question is to what extent? This depends primarily on the geology and whether there is water within the strata. The potential for surface water to enter the ground through mining related fractures to become groundwater also needs to also be considered. It will cause the loss of this surface water. Where groundwater pressure is lost due to mining, the potential exists for gassy strata to release gas through fractures. Figure 3 shows the water cycle which should always be considered in its entirety.

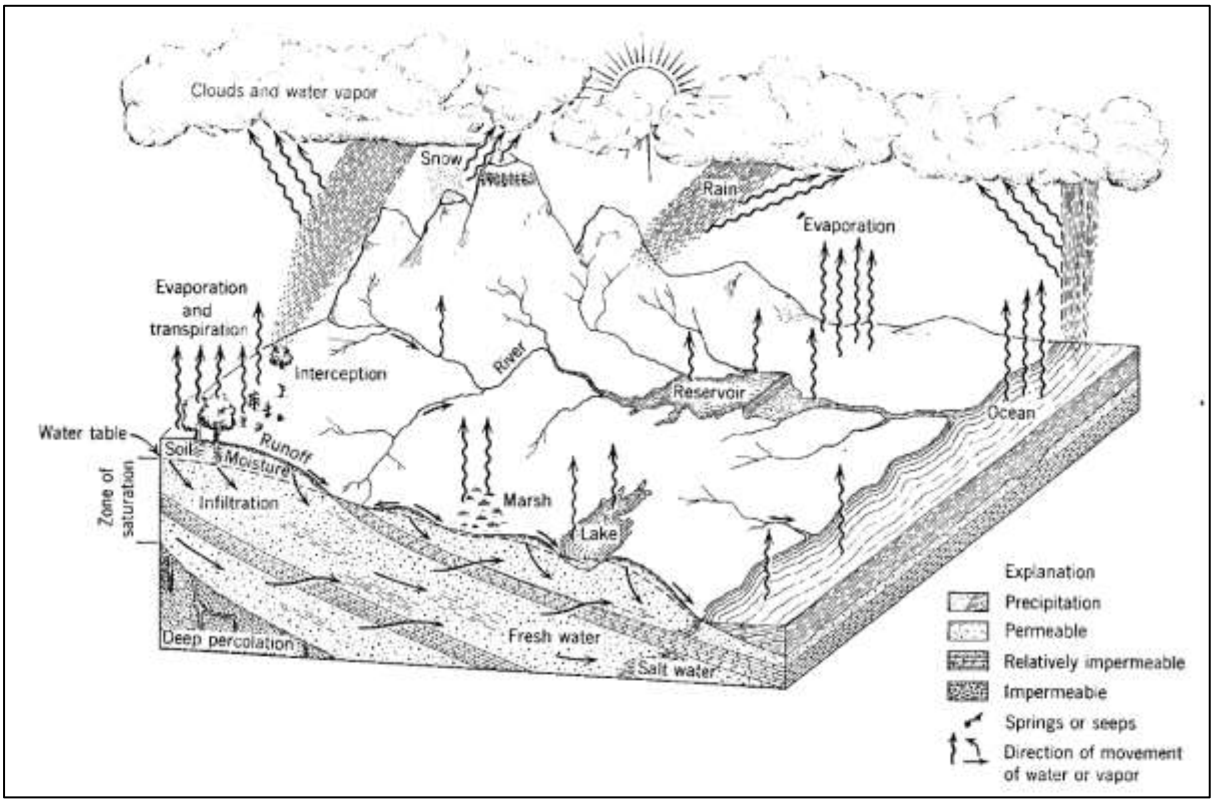

Figure 3. The movement of water in the air, surface and underground (De Weist, 1965)

The real problems with groundwater occur where there is a lot of it. This means that either a large volume of it is stored within the strata or that the strata has the ability to transport it quickly. Where there is a lot of stored water and there is an ability to transport it quickly, problems multiply in terms of water volume.

A clean gravel can store a lot of water in the voids and will have a high permeability (or in water terms hydraulic conductivity). This is why a thick gravel aquifer will quite probably be an important source of water through water supply wells. If mining causes this to drain into a goaf then it is likely to be a double source of woe for the miner as the mine may be flooded and there is the potential that the owner of the wells will be seeking damages for the loss of water.

The transport of water through strata is governed by its thickness, permeability and the hydraulic gradient that exists. There are two quite different storage behaviours. The first is called storativity and refers to the water that is given up by the formation on loss of head (i.e a pressure type term). The other is called specific yield and refers to the water lost by draining the void space. The latter is much greater than the former. Vuggy limestones or vesicular basalts may yield a large component of their volume by drainage. Fractured rock tends to have quite low specific yields. In some cases, coal seams are themselves aquifers.

The effect of water on stability of openings or excavations is not related to the potential of these to flow, but rather the pressure of the water and the influence this has on effective stress. This latter is dependent on the poroelastic behaviour of the rock or the size of open fractures within the rock mass.

The chemical composition of water is also important. Some water is highly corrosive and will corrode rock supports and equipment rapidly. In other cases, mining may lead to the contamination of comparatively clean groundwater with chemicals from other rock sources. Dealing with contaminated water is always a problem.

The key points raised in this section which require determination in exploration are:

- Ground water pressure (head)
- Permeability (or hydraulic conductivity)
- Storativity (or in oil and gas terms the compressibility porosity product)
- Specific yield
- Thickness
- Chemistry

Drilling Methods

Determining whether a suitable coal resource exists or not will always be an initial priority and exploration at the early stage will tend to focus on the location and volume of the coal. While some initial holes may be by open hole drilling with chip sampling (followed by borehole geophysics), this is likely to change fairly rapidly into a coring operation to determine coal quality. This has been dominated by the use of wireline coring. In Australia this has standardised on the Boart-Longyear HQ-3 triple tube system which provides 60.9 mm core from a 96 mm diameter hole. This is augmented by the use of PQ-3 coring where larger coal samples are required for special quality testing (e.g. for coking tests). In some cases, such as where seismic surveys cannot be conducted, the use of directional drilling from surface may be considered for seam delineation.

Vertical Open Hole Drilling

If exploration commences with open hole drilling, then sampling will be by chips and hopefully by subsequently running a suite of borehole geophysical logs. The chips will provide very limited information on the lithology and no information on structure other than by noting matters such as drilling fluid loss. However, the chips can provide information on gas within carbonaceous strata. If drilling is by air, the chips may be obtained very quickly and sealed within canisters for desorption monitoring. Drilling in potentially gassy formations with air does, however, bring with it an increased risk of blowout and at the least a good diverter needs to be incorporated into the well head, preferably with a valve at the top of cemented casing to shut in the well once the drill pipe is extracted.

If drilling is by mud, then there is a much less likelihood of blowouts occurring. The mud density may be controlled and a blowout preventer can be used to shut the well in if needs be. The use of drilling mud opens up the possibility of using oilfield type gas sensing equipment over a shaker to give measurements of gas type and indications of gas quantity.

Vertical Hole Core Drilling

Most exploration core drilling for underground coal is accomplished by the use of the Boart Longyear HQ-3 or PQ-3 wireline coring systems. It is, therefore, logical to base most in situ and core testing around these drilling systems and core sizes.

One of the complications of wireline coring in potentially gassy formations is that of a blowout. This is most commonly brought about by the inner tube being pulled out of the drill string at too high a rate, thus lowering the pressure within the hole. This is called swabbing and it may induce gas flow into the hole. The free gas expands on rising and lowers the pressure in the well bore and in the core rods, thus further inducing more flow into the hole. This is an unstable situation and most commonly leads to the ejection of the inner tube from the top of the core rods. This is a potentially lethal event that can be followed by another - the uncontrolled blowout of the hole.

These problems can be overcome by the use of a cemented casing, a blowout preventer and a wireline stripper. The latter is used to prevent swabbing of the hole. The use of this equipment is straightforward, but requires attention to detail in procedures. Rigorous adherence to oilfield blowout procedures is quite inappropriate for this type of drilling, as they are dealing with a different scale of problem and different drilling equipment.

Directional Drilling

Directional drilling using downhole mud motors to guide the drill string either from surface as surface to in seam (SIS) or underground in seam (UIS) has been increasingly used in exploration to determine the location of faulting within the seam. This is usually undertaken by drilling in seam with periodic branching to make the branch touch the roof and sometimes the floor. While this method will give the geometry of the hole, it does not give much information on the nature of the coal nor the type of fault. Problems may be encountered while drilling through some faulted areas, but this is the extent of geotechnical information that may be gleaned. The incorporation of good borehole geophysics is really desirable.

While part of the need for this type of drilling is simply to find whether the coal seam is present, the geotechnical need is to identify areas of changed parameters and particularly zones that may suffer from outbursts or rockbursting.

Borehole Geophysics

Borehole geophysics gives some indication of lithology types and more particular changes in lithology. From a geotechnical viewpoint sonic logs and acoustic televiewer images are particularly useful. Full waveform sonic logs can be interpreted in terms of the dynamic Young's modulus and Poisson's ratio. The interpretation is, however, invariably given in terms of isotropic rock behaviour and therefore information on anisotropy cannot be obtained. The use of correlations between the Young's modulus determined from the sonic log and laboratory determined moduli can be useful. Taking this further into trying to assess rock strength from the sonic log is tenuous.

The use of oilfield tools with rotating dipoles would permit the determination of fast and slow shear waves and with them an indication of anisotropy. This technology has not yet been adopted in coal mining.

The use of the acoustic televiewer is immensely valuable in determining structural features and bedding plane orientation from the borehole image. It does not remove the need to log core to see the structural features and examine their physical state. However, by reconciliation of the core log and that of the acoustic televiewer image, it is possible to orientate the core, and thus save on the complications of obtaining orientated core by other more complex means. The acoustic televiewer image also enables the borehole wall to be examined for breakout or tensile fracturing. These features are a function of stress at the borehole wall caused by concentration of the rock stress by the borehole. In dry holes where acoustic televiewers cannot be used, optical televiewers may be used.

Hatherly et al (2009) take the use of borehole geophysics even further with the development of the Geophysical Strength Rating. Borehole geophysics alone cannot, however, make up for having some core to look at and test. The properties of sedimentary rocks are too complex to determine from geophysics alone. The process should be one of using the geophysics to assist in the interpolation of properties from point measurements.

Seismic Surveys

The prime purpose of seismic surveys from surface is to delineate faulting within the coal seams of interest. The basis of this is a reflected pulse from each horizon. To achieve this it is necessary to have a reflection at each change of strata. This requires an increasing sonic velocity profile with depth. This assumption is frequently not met meaning there is loss of information.

The reflected signal is time based and is decoded by matching it with a P wave borehole sonic log to provide depth, rather than time, information. This can be corrected further by the use of other borehole data.

The clearest signal comes when the seam of interest is the first in the sequence. When more seams exist, problems occur with interbed-multiples (echoes). Where there are gradational changes of rock type, there is loss of definition.

The ability to determine the presence of a fault is very dependent on the strata. It may be possible to determine a seam thickness fault in a 2.5 m seam in good conditions or it may require 6 m throw in less favourable seismic conditions. Indications of seam thinning or the presence of roof channels may be hinted at by seismic surveys. However, the ability to positively identify either of these is not good. What seismic will reveal very well are major structures that may stop a mine. Where there is uncertainty in the seismic information it is an indication that more drilling should be undertaken. This means that drilling can be targeted on areas of uncertainty, rather than just working to a grid of drilling.

Most seismic surveys for coal are conducted in three dimensions (3D seismic).

Rock Stress Measurement

Most rock stress measurement is in some way or another reliant on assumptions of the rock being linearly elastic and isotropic in its behaviour. As most sedimentary rock does not behave in this manner, care must be taken in applying these assumptions.

Rock stress measurement falls into four categories depending on the rock type and its structure, Gray (2018). These are described in the following four sections.

Unfractured Rock Mass Exhibiting Borehole Breakout or Plasticity

Hydrofracturing may be used to determine the minimum rock stress. This is achieved by sealing a section of the hole with packers and injecting fluid into the test zone until the borehole wall fractures and a fracture propagates into the rock mass. This fracture will rotate to a plane perpendicular to that of the minimum principal stress or until the hydrofracture is captured by a natural fracture. If the plane on which minimum normal stress acts is horizontal, then that is the orientation which the fracture will take.

Once the fracture is generated, pumping is then halted and the fracture is allowed to close by leak off of the fracture fluid into the formation. The fracture closure may be determined by analysis of the pressure decline. It is however complicated if the fracture rotates or intersects a natural fracture as several fracture closure pressures may be analysed.

The concept of re-pressurising the hole to re-open the fracture and to determine a major stress is fraught with difficulty especially within sedimentary rock. The process of determining the major stress requires the assumption that the fracture does not rotate, that the rock is linearly elastic and that the hole closes perfectly after the initial fracture process. None of these criteria is likely to be fully met and certainly not simultaneously. Another practical complication with the use of hydrofracture is that the packers have to be inflated to a pressure that is greater than the injection pressure to prevent leakage around them. This means that the packers may initiate any fracture. Avoiding this requires keeping the packer sealing pressure just above that of the injecting fluid.

If the minimum stress perpendicular to the borehole can be determined by hydrofracture then borehole breakout offers an option to gain some insight into the major stress perpendicular to the hole. Breakout occurs where the compressive stress at the borehole wall exceeds the rock strength tangential to the hole wall. It is in itself a useful indicator of whether problems will occur in subsequent mining, especially for developments such as shaft which are in the same direction as the borehole. Breakout occurs at ninety degrees to the direction of principal stress perpendicular to the borehole. It therefore an indicator of stress direction.

The determination of the major stress magnitude perpendicular to the hole cannot be determined from breakout alone. If the minor stress is available from hydrofracture then it is possible to use this, and the angular width of the breakout zone, to estimate the major stress. Whether this calculation uses linear elasticity or some more complex material properties it is still going to be approximate and very dependent on knowing the compressive strength of the rock tangential to the borehole wall. This is most infrequently known and is more likely to be derived from the tenuous relation of rock strength with a sonic log. These are a lot of assumptions but they can still be useful in providing bounds on likely stresses.

In some cases the borehole wall may develop a tensile fracture. This is brought about by the combined effects of a large difference in major to minor stress perpendicular to the hole and by internal fluid pressure.

The determination of borehole breakout and the orientation of hydrofractures at the borehole wall are dependent on the use of an acoustic televiewer image.

Hydrofracture is about the only method that can be used to determine stress in coal. Only the very strongest coal may be overcored using glue-in overcore systems.

Fractured Rock Mass

In this case the only option may be to use hydrojacking to open pre-existing fractures. Ideally one fracture at a time is jacked so that the stress across that fracture may be determined. Practically this may be impossible because once one joint is opened the fluid used for hydrojacking will flow into another joint or joint set. In many cases it is simply impossible to isolate one joint between packers to achieve this. In either case something approximating the lowest stress across the intersected joint set is likely to be all that can be determined.

Un-fractured Rock Mass with Elastic Rock Behaviour

Rock that falls in this category is most usefully and efficiently tested by overcoring (Gray, 2000, 2018). This involves drilling to depth and pulling the last core run. The end of the hole is then prepared by either drilling a smaller hole or cone beyond it. This is flushed and a tool is installed within the hole or cone. This measures the hole diameter or adheres strain gauges to the wall of the hole or the cone. The orientation of this tool is then determined, usually by the use of magnetometers and accelerometers. The end hole or cone is then cored over thus relieving it of stress. The change in dimension or strain is recorded and the core and tool are retrieved by the wireline core retrieval system. Once on surface the sample may be tested for its elastic properties and the stress calculated. Because of the non-linear elastic behaviour of many of the sedimentary rocks this analysis can be quite complicated.

Methods of Stress Estimation based on Core

Various method to determine rock stress from core abound. These include the use of the Kaiser Effect and Deformation Rate Analysis techniques in which the core is reloaded and reportedly changes its behaviour when it reaches the previous stress state (Hseih, 2013). These involve taking sub cores at different orientations and then re-loading them to endeavour to determine the stress tensor. Another technique is that of measuring the post elastic deformation of core by adhering strain gauges to it and monitoring the strain change over time. For this to make any sense the core must be held at constant temperature and moisture level. It then becomes difficult in the field.

An alternative promising method to measure the difference in stress perpendicular to the core is to measure its ovality. This can then be linked to the difference in stress through the core's elastic properties. This requires the core to cut, but not re-ground, inside the core bit. Once recovered, the core ovality can be measured as frequently as required. In this form, it serves as a field indicator of stress change differences and, therefore, may be used as guide to when stresses have changed and when to make a more precise stress measurement.

Measuring Gas in Rock and Coal

Gas is stored in coal or rock in three modes. These are within open pore space, by solution in fluids and by sorption. The normal process adopted to determine the gas content of a coal is to take core, withdraw it from the hole and to put it into cannisters as quickly as is possible. The rate of gas production is measured and used to predict the gas lost on retrieval from the hole. Desorption may continue for long or short periods, followed by crushing of some of the coal to speed the desorption process and release so-called residual (Qm) gas. This is the process described by Kissell, McCulloch & Elder (1973) and more recently in the Australian context in the standard AS 3980:2016.

The core desorption process has several assumptions, one potentially major error and an omission. The major error is that any gas held in pore space is lost and is not, therefore, included. The assumptions around the lost gas on core retrieval are always open to question and the omission is in the determination of the total gas content to atmospheric partial pressure and below (Qm vs Qt). The latter is discussed by Saghafi (2010) and Wood and Gray (2015). The gas lost in desorption at the early part of core recovery is frequently considered to be small. This may not be the case. Unpublished reports exist that mention 50% gas loss from core recovered from lower rank coals in Sumatra.

An alternative method to measure gas content was thoroughly tested by Gray, Singh and O'Brien (2013). It involves drilling an open hole in which the drilling is in an overbalanced mode. This means that the fluid pressure in the hole is kept above reservoir pressure. In addition a drilling mud is used that does not invade the formation at a significant rate. In this mode, the only things that rise up the borehole are what is pumped down and what is cut. This is constituted of drilling mud, cuttings and whatever gasses or liquids they contain rising up the hole annulus. At the surface is a rotary seal and the mud, cuttings and gas pass through a cyclonic separator which splits the gas so that it can be measured. The mud and cuttings pass onto a shaker and the cuttings can be collected for further desorption. This method has a very low gas loss during the period the cuttings are on the shaker. The complications of the system are that the exact location of the gassy strata tends to be a little uncertain but can generally be ascertained by examination of geophysical borehole logs. Simpler versions of cuttings sampling may be used with associated loss of precision.

The alternatives are to core with a core barrel either that captures the core and seals it at pressure within the core barrel or alternatively which captures released gas as the core is brought to surface. The former is in the form of a large diameter oilfield core barrel while the latter system may be incorporated into either HQ or PQ wireline coring.

Where highly porous strata exist, the process becomes one of finding out the porosity, saturation (water and gas) and the gas pressure so that the gas volume may be calculated from pressure-temperature-volume information. This approach uses core and geophysical logging to determine in situ parameters and it becomes a more general petroleum approach.

The rate of gas release from core is an extremely important one as it can be used directly in determining the rate of gas release from cut coal, as part of the gas drainage process or in outburst risk determination. The complication in analysis is the effect of the core size and the fractures within the core on desorption. Desorption is controlled by the diffusive characteristic of the coal and the particle size and shape that is releasing gas. Wood and Gray (2015) have found it useful to derive an apparent diffusion coefficient which is derived from the initial diffusion rate of a core assuming it to be a cylinder undergoing Fickian diffusion. As some fracturing exists in any core the value of the diffusion coefficient it returns is always on the high side. This is conservative for most applications.

Permeability and Reservoir Parameters

The key reservoir parameters are the fluid type, its pressure-volume-temperature behaviour, reservoir pressure and permeability. Following in importance come the complications of directional permeability and inhomogeneity of the reservoir, be it an aquifer or a gas bearing coal seam. Coal seams also exhibit large changes in permeability during drainage.

The measurement of permeability and reservoir pressure is made infinitely easier if there is only a single fluid phase present. If the reservoir is an aquifer, then the fluid is water. This is also generally the case in most coal seams before any drainage has taken place. Once some drainage has been allowed to occur, two phases – gas and water, exist. Each of these impedes the movement of the other through the reservoir in a manner which is not proportionate to the fraction of the phase being present.

If a simple gas bearing stratum exists, such as a coal seam between a tight roof and floor, then its testing is also relatively straightforward, albeit with a need for caution because of the potential for blowout. A blowout occurs where free gas enters the well bore from the formation and rises up forming ever enlarging bubbles. This lowers the density of the fluid in the well thus lowering the pressure therein. This lowered pressure enables more gas flow to enter the well bore. This process can accelerate dangerously into a blowout. Well control practices are required to deal with such an event.

Where gas and water exist and they flow together, things become more difficult to measure and test as there are two fluids involved. Answers can be obtained for permeability, but there is always an approximation associated with the measurement as the permeability around the well bore will change with the fraction of whichever phase is present. For example testing a coal seam which is fully water saturated will produce water alone. Thus the permeability measured and the near well bore loss terms (skin) measured are based on water and will probably be to some degree pressure dependent. When the coal seam has both gas and water present the saturation (ratio) of water will affect the permeability to both gas and water. In this case the permeability will be based on an analysis which is complicated by both changing saturation away from the well bore and on changing pressures.

Being able to measure a large enough zone of influence around any test well is important, as is separating the near-well bore effects from the general measurement. The lower the permeability of the rock mass being tested, the smaller the distance that is affected by the test process. The zone around the tested section of a well is likely to be affected by either drilling mud invasion, or by stress concentrations associated with the presence of the borehole in the rock mass. These generally reduce the permeability locally around the hole. All test processes for permeability involve the measurement of both fluid flow and pressure. If this is undertaken simultaneously within the same well, then the separation of near-well bore pressure drop from that occurring in the formation becomes complex and relies on some assumption about the behaviour of near-well bore losses. This is usually that they are proportional to the flow rate. However, the loss terms frequently change during any test, particularly if that test involves injection.

There are two ways to get over the problem of separating flowing well bore loss terms from changes in pressure due to flow to, or from, the test zone. One is to measure the pressure changes on pressure recovery after the flow has ceased, while the second is to place a pressure transducer (piezometer) in the formation outside of the zone of well bore influence and to monitor it during flow and no-flow periods. The latter has the advantage that it enables the calculation of the storage behaviour of the reservoir.

However, it is frequently not convenient to drill two holes to measure reservoir parameters. In this case the most useful form of test has been found to be an adaptation of the oilfield drill stem test (DST). In its traditional form, this involves lowering an empty drill string into the well and isolating the test zone with packers. The pressure is then permitted to stabilise and then a valve is opened that permits flow from the test zone into the drill string. After a period of inflow, the valve is shut and the pressure is allowed to build up. The inflow, inflow time and the pressure build up are monitored and may be readily analysed to provide information on the reservoir pressure and a non-directional permeability. If some reasonable assumptions on the reservoir storage behaviour are made, it is possible to derive the well bore loss characteristics and the mean effective radius of investigation of the test.

The well bore loss may be described in terms of an effective well bore radius. If the formation is of low permeability and the effective radius of investigation is not significantly greater than the effective well bore radius, then the test may be regarded as of little value. A lot of information can be obtained from a DST single test. The test may also be performed in reverse by injecting into the test zone in the form of an injection fall off test (IFOT) which may be a falling head test with shut in (FHSI). In this case the risk is of clogging cleats in the coal or pores near the well bore and of injecting a fluid which is not the reservoir fluid. The latter may cause uncertainty in analysis due to the unknown fluid viscosity and due to any reactions between this fluid and clays within the formation being tested.

The authors have developed and used over the last 20 years equipment to enable the DST test to be conducted with HQ wireline coring equipment. Two forms of this exist, one that uses a single or a straddle inflatable packer on the end of the HQ or HRQ drill string and one where the packers can be lowered through the drill string on wireline and locate on the landing ring normally used to support the inner core barrel assembly. In both cases, the fluid level in the drill string is depressed by the use of compressed air or nitrogen. This equipment may also be used for injection testing. Both variants of the equipment produce a surface readout of the pressures in the test zone, drill string and packers.

Another variant of this equipment exists. This is the closed chamber DST test (CCDST). In this the tool is fitted with an evacuated chamber of fixed volume. Once the test zone is isolated and pressures stabilised the valve is opened to permit flow into the chamber. This fills with the well fluids. The valve is then closed and the pressure build up is completed. This tool suits lower permeability situations and it permits the capture of gas and water. The variant used by the authors is a wireline tool for use through the HQ or HRQ drill string.

The use of packer testing to determine a value of inflow measured in lugeons is not a valid means to determine permeability and should not be used. The test method does not provide any information on reservoir pressure and fails to separate pressure loss at the well bore from pressure change in the reservoir.

A number of DST, IFOT or FHSI tests may be used to gain information on the variability of permeability and pressure vertically and laterally. Conducting multiple tests in low permeability formations can be a very slow procedure and only those zones that are likely to be critical to the mine design should be tested.

A variant of the DST can be effectively used to determine both inhomogeneity and anisotropy of a formation. This is described by Gray (2015) and involves conducting a normal DST test in a section of a borehole to yield information on pressure and average permeability. A pressure monitoring system is then installed in the test zone. This may be by cement displacement installation (Gray and Neels, 2015) which overcomes the problems of connection to the formation and intra-connection within the borehole of cement-bentonite installations. It may also be by the use of packers, including swell packers. Following this, another hole is drilled and a DST conducted while monitoring the pressure in the first hole. This test yields the average permeability and the directional permeability between holes. This process may be repeated again with a pressure transducer installation and another hole to yield another average permeability and two directional permeabilities. This approach enables the segregation of inhomogeneity from anisotropy, something that a normal interference (pumping or injection) test with a central well and multiple transducers (piezometers) will not do as well.

Where highly permeable formations exist such as sands or gravels, DST testing is quite inappropriate. What is needed is the pumping of a large volume of fluid so that an adequate pressure drawdown occurs to cause the fluid level to drop within the test zone so that the specific yield of the aquifer may be determined. This needs to be an interference test with surrounding piezometers. The latter are best installed by traditional gravel pack methods.

Determining the nature of minerals dissolved in groundwater is important. This is from a number of viewpoints, including corrosion of equipment and rock support, from the view that the groundwater may be used in the processing plant and that it may need to be disposed of. Getting samples of groundwater can be difficult, particularly if drilling fluid has invaded sufficiently around the hole that it displaces the existing fluid. Taking groundwater samples requires either pumping for a long period to get the drilling fluid out of the formation first, or making sure it does not go into it by using a suitable mud that forms a filter cake on the hole wall. In the latter case, tests such as a closed chamber DST test may be useful.

It should not be forgotten that bacteria can exist in some of the most unexpected places, so any groundwater testing should also include testing for some of the microbes that may be encountered in the ground. Care should be taken to prevent oxidation during sampling and transport.

Rock Properties

Coal measure rocks are frequently anisotropic and, therefore, the test processes used to assess their properties need to reflect this. Where jointing exists, it is likely to have a major effect on rock behaviour and such jointing needs to be measured. The best way to do this is as part of the core logging process. Core logging enables the lithology to be determined and the joints to be measured in terms of orientation and the nature of the joint. The joints can then be aligned with information from the acoustic televiewer scan of the hole. Proper logging of core also enables visual observation for inhomogeneity and the choice of samples, and how they should be geotechnically tested.

All too frequently, testing has become a rote process of taking representative samples of each lithology and sending them off for uniaxial compressive testing, interspersed with the occasional triaxial test, to determine failure properties. The problem with this approach is that the failure mode in these tests is by shearing at an acute angle to the sample axis which, in vertical holes, is generally across the bedding planes. Furthermore, the only pre-failure parameters measured are the axial Young’s modulus and associated Poisson’s ratio. As most laminated rocks have quite different mechanical properties depending on the direction, this is very limiting. Not only are most coal measure rocks anisotropic, but they frequently display quite significant non-linearity prior to failure. The test process, therefore, needs to be thought about carefully.

Gray (2020) reviews existing test methods and presents some new ones which have particular relevance to coal mining. These tests can be subdivided into tests for elastic and plastic behaviour prior to peak strength being reached, and for the strength at failure. Index tests for strength are valuable, though they do not give any fundamental parameter. Point load testing is a particularly useful index test because it can be performed on fresh core on site before any expansive clays have time to react. Care must be taken to perform point load tests as soon as possible after core extraction. Core must be kept moist to prevent changes in strength due to desiccation.

In the point load test is the core is loaded to failure both axially and across the diameter. The resulting value is normalised to a 50 mm diameter core. This test gives a difference in strength of the rock to loading in different directions. The index is just that though, because the test imposes a complex loading situation on the core comprising local compression and the generation of tensile stress through the shape of the point and the core. The test may also be biased by the selection of unrepresentative stronger material for testing.

Another useful index test is the Protodyakanov Index in which lumps of rock or coal are struck with a drop hammer in a cylinder to make them break up. The test gives a useful first pass at rock toughness and correlates reasonably well with tensile strength. It is used as an indicator of outburst proneness in Russia and China. The Brazilian test in which a core is loaded by a line load across opposite diameters is also really an Index test as it generates tensile strength by the shape of the core and the line load imposed on opposing sides. The analysis of this test is dependent on the core being perfectly linearly elastic, something that is unlikely in coal measure rocks. Brazilian tests cause a complex failure involving tensile failure and shear.

Because of the importance of the bedding planes on the failure of laminated rocks, test processes that measure these need to be adopted. A modified form of the GOST (ГОСТ, 1988, государственный стандарт – Russian governmental standard) shear test is particularly suitable for this. It is a simple shear test in which the ratio of normal to shear stress can be varied by changing the angle of loading in a universal test machine with respect to the failure plane. In the modified form used by the authors, the shear plane is perpendicular to the core axis enabling shear parameters (cohesion and friction angle) to be determined along the bedding plane. This measurement may be used in roadway design or in determining whether a goaf will fail in shear on bedding planes.

Tensile strength is also important. The tensile strength across bedding planes may determine how a roadway roof fails. It may also determine how the immediate goaf falls behind a longwall. The tensile strength in the direction of bedding planes may also control goaf failure. This is especially the case in massive strata where the tensile strength at the top of the block controls the goaf overhang prior to failure.

Figure 4 shows the Mohr's circles of failure for the various strength tests along with the Mohr –Coulomb failure envelope. The simple shear test in this refers to a test purely to measure cohesion of the rock perpendicular to the core. This is measured with respect to a different failure envelope than that shown in the figure. The GOST test shear is shown as a point though it could equally well be shown with a Mohr's circle passing through it.

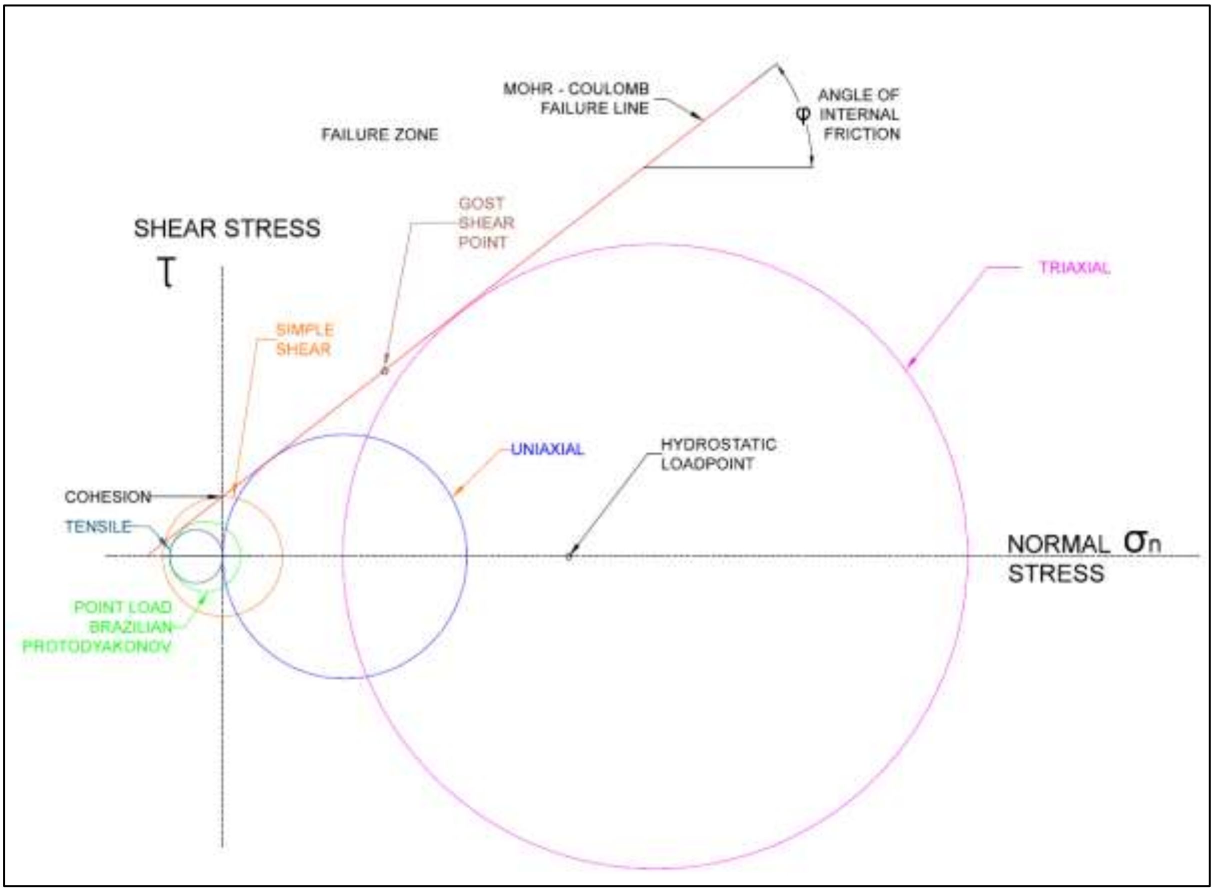

Figure 4.Stress loadings imposed by various rock tests, Gray, 2020.

The rock mass will strain prior to failure at some combination of stress level. Some of this strain is elastic and some of it may be plastic. What is elastic may not be either isotropic or linear. In this respect, the usual use of uniaxial compressive testing is totally inadequate, though cyclic uniaxial testing can be used to reveal plastic offset. The development of triaxial testing to determine the full pre-failure behaviour of the rock is extremely important. In this, the rock's Young's moduli and Poisson's ratios are determined at multiple stress states.

There is little point in numerical modelling of rock behaviour using linear elastic models with unrealistic failure behaviour. The pre-failure behaviour of the rock mass is just as important as the strength at failure. Whether the rock mass reaches a state of stress where it will fail depends on its initial behaviour and stress.

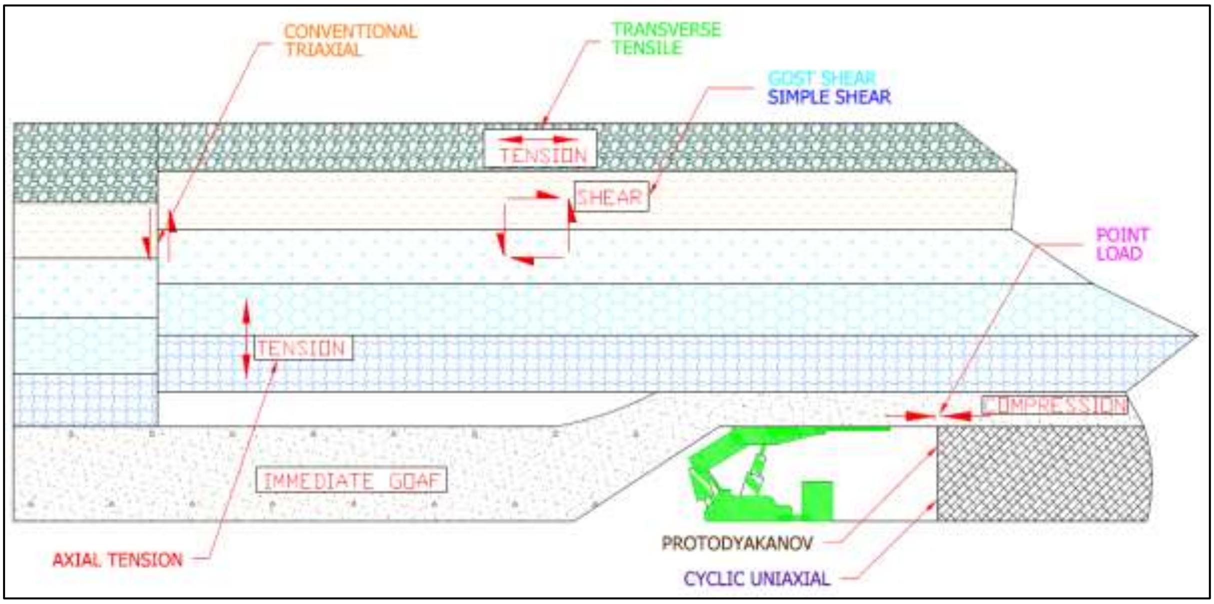

Figure 5. Schematic view of a longwall with tests that might be used to determine rock properties, Gray 2020.

Figure 5 schematically shows a section of a longwall with a weak roof and massive strata above. The figure shows where different tests to determine rock strengths might be used based upon core from vertical exploration core holes.

The weak roof forms an immediate goaf and it may have caused problems in the gate roads beforehand. Its strength measurement is practically done by the point load test which can be used to arrive at comparative strength in the rock in the direction of the core and transversely to it. Ideally, we would have core that is parallel to bedding to test its strength under the action of the cantilever. Sub coring HQ core is not, however, a good option. Just below the immediate roof we have the coal face.

In the coal, the Protodyakanov test provides an index which may be used to determine its outburst potential (Wood and Gray, 2015). The cyclic uniaxial test could also be used to advantage on the coal to help determine how much strength it has and whether it will deform as indicated by permanent offset (Gray, 2020).

Moving to the top of the massive block, we have a zone that may be in tension as it is at the top of the cantilever. Here the transverse tensile test is relevant. If we have a tensile failure stress that is high, then the goaf may hang up, and then fail suddenly, with the associated problems of wind blast and weightings on the face and powered supports. Failure may also be accompanied by a significant seismic event that causes rock bursting in the development roads.

The cantilever may also fail in shear. While we think of the shear stress as that being required to support the cantilever vertically, it is accompanied by a conjugate shear along the bedding planes. If shear failure takes place before tensile failure, then the entire goaf formation takes a different form as the strata folds downwards. Here, the simple shear and the modified GOST shear are particularly useful as they enable the determination of the shear strength along the bedding planes. One only provides cohesion, while the other can yield both cohesion and friction angle.

If we move to the left of Figure 5 we can see that it presents a potential shear failure through the bedding at the end of the cantilever. This shear failure is probably more likely above the face but there is limited room on the figure to show this. Nonetheless in determining shear through the bedding a conventional triaxial test on the exploration core is the most relevant. If we move down to the strata layer just above the immediately fallen goaf we can find that the strata there may pull away in tension. The axial tensile test to cause failure on the bedding plane is useful in determining strength of the rock in tension.

The behaviour of rock under the influence of water may be tested by examining its slaking behaviour. In its simple form, this test involves putting crumbs of rock in water and observing whether they disassociate. In cases of doubt, the mineralogy should be determined using scanning electron microscopy. The presence of reactive clays, such as smectite, can lead to problems of rock deterioration on exposure. This may affect roadways or roofs.

There is still another rock property that may be very important and should be measured. This is its poroelasticity. The effective stress within any rock or coal mass that contains fluid at pressure will depend on this. It is particularly important in porous rock or cleated coals and can be measured as part of the authors' triaxial test process.

Finally, where gassy coals exist there is a need to determine how their dimension will change as they desorb gas. This changing dimension will affect the stress in the coal and, subsequently, its permeability. It can also have an important effect on the lateral stresses that exists in pillars (Gray, 2014). Measuring dimensional change under gas pressure usually takes months to accomplish. However, it is a fundamental part of looking at the stress path that coal undergoes during drainage. Gray, 1983, 1987 and 2011 discusses these effects. In practice, this has been updated by the incorporation of real measurements of poroelasticity and non-linear coal properties.

Spontaneous Combustion

The risk of spontaneous combustion is ever present within a coal mine. Determining the propensity of the coal to self-ignite is one of the measurements that should be made as part of exploration and, also, routinely during mining. The determination of a coal's propensity to spontaneously combust has become one of sealing samples into an adiabatic vessel and allowing them to self-heat under conditions closely replicating the mining environment (Beamish and Beamish, 2011; Beamish and Theiler, 2017; Beamish and Theiler 2019).

Some Examples

All mining operations are dependent on the understanding of potential risks based on the measurements conducted through the exploration and development phases of the project. Although numerous examples are available, a few are highlighted here.

Leichhardt Colliery

This mine characterises all that might be done wrong in exploration. The man-riding and winding shafts were sunk in 1974 on the basis of four boreholes. These were located between four full seam thickness faults. Development in the coal led to multiple outbursts, most of which were thought to be relatively benign as they involved the expulsion of a few tonnes of coal from cone shaped failures in the face, roof or ribside as shown in Figure 6. Two of the outbursts were not benign. One was of the more normal form and ejected 350 tonnes of coal from a fairly solid face.

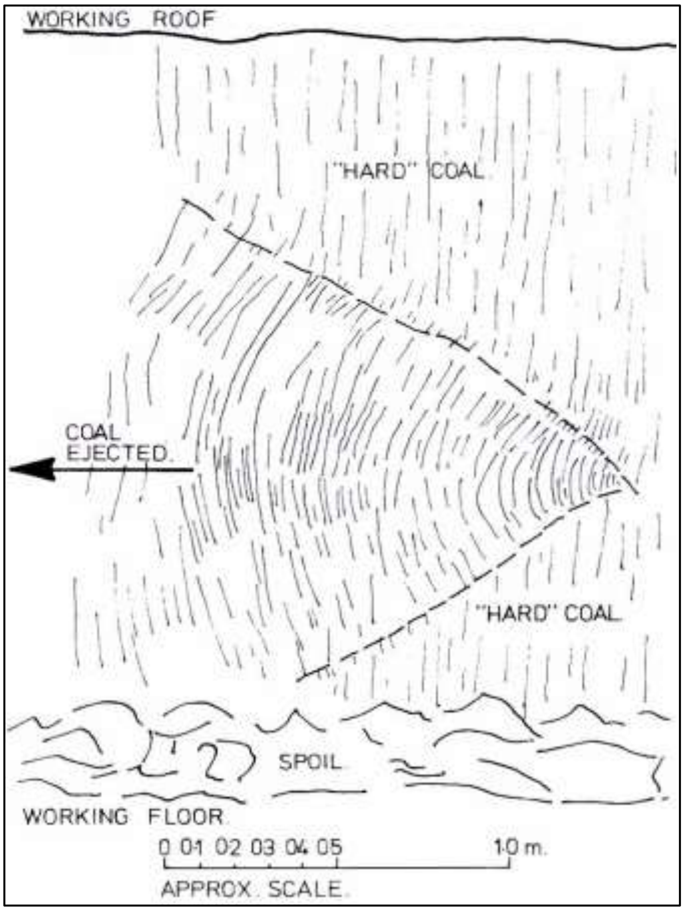

Figure 6. Elevational view of normal Leichhardt Colliery outburst cavity, Moore and Hanes (1980).

The next large outburst was from a reverse fault that was intersected in the direction of its strike. It was far more violent and led to a dual fatality. This outburst occurred in several stages probably corresponding to multiple events of erosion, blocking, pressurisation of a cavity by desorption and then violent expulsion. Diagrams of the outburst cavity are shown in Figure 7~~Figure 7~~.

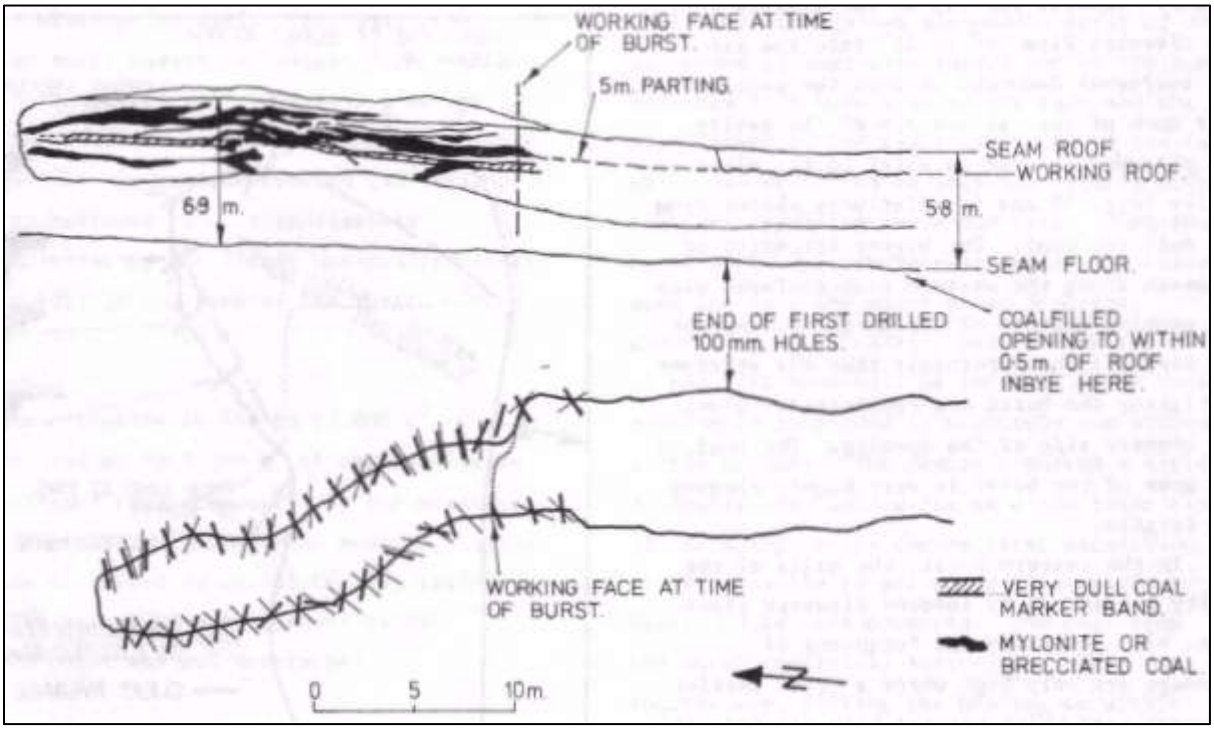

Figure 7. Outburst cavity from 1 December 1978 Outburst at Leichhardt Colliery. (Hanes, 1979). `

Efforts at gas drainage failed because gas did not flow and de-stress drilling did not work. However, when mining returned to a part of the mine that had drained into older workings, all the problems disappeared, proving that gas was the main culprit. Subsequent long term gas drainage trials showed that gas drainage did indeed occur, but only in the direction of the predominant cleating. Further initial flow from gas drainage was very slow, but increased significantly with time. This was due to a two to three orders of magnitude change in permeability that was associated with destressing due to coal shrinkage and the stress path that the coal underwent.

The mine did not produce much coal before it closed in 1982 but it did serve as an extremely good training ground where much research and development was done. The problem of being able to locate the reverse fault that led to the fatal outburst has, however, never really been resolved.

South Bulga

In 1984, South Bulga in the Hunter Valley Coalfield commenced operations as a punch mine from an open cut highwall at 40-160 m depth. The immediate roof was 22-28 m of massive sandstone with uniaxial compressive strength of 40-80 MPa and modulus of 13 GPa. The powered roof supports were a combination of 940 and 1150 tonnes, the highest capacity available at the time. The mine regularly experienced rapid convergence during cyclic loading. By 1998, the mine had become ironbound three times, along with an additional eight rapid convergence events. South Bulga had an extremely low initial stress regime with a horizontal to vertical stress ratio of 0.5 (Sandford, 1998; Sandford et al, 1999). The cause of this cyclic weighting was the massive rock in the roof which failed periodically, probably in tension. If this problem had been recognised at the time by proper measurement of the rock behaviour, especially that of the tensile strength of the rock rather than compressive strength, then measures to pre-condition the rock mass may have been considered. Figure 8 shows the frequency of weighting events at South Bulga mine.

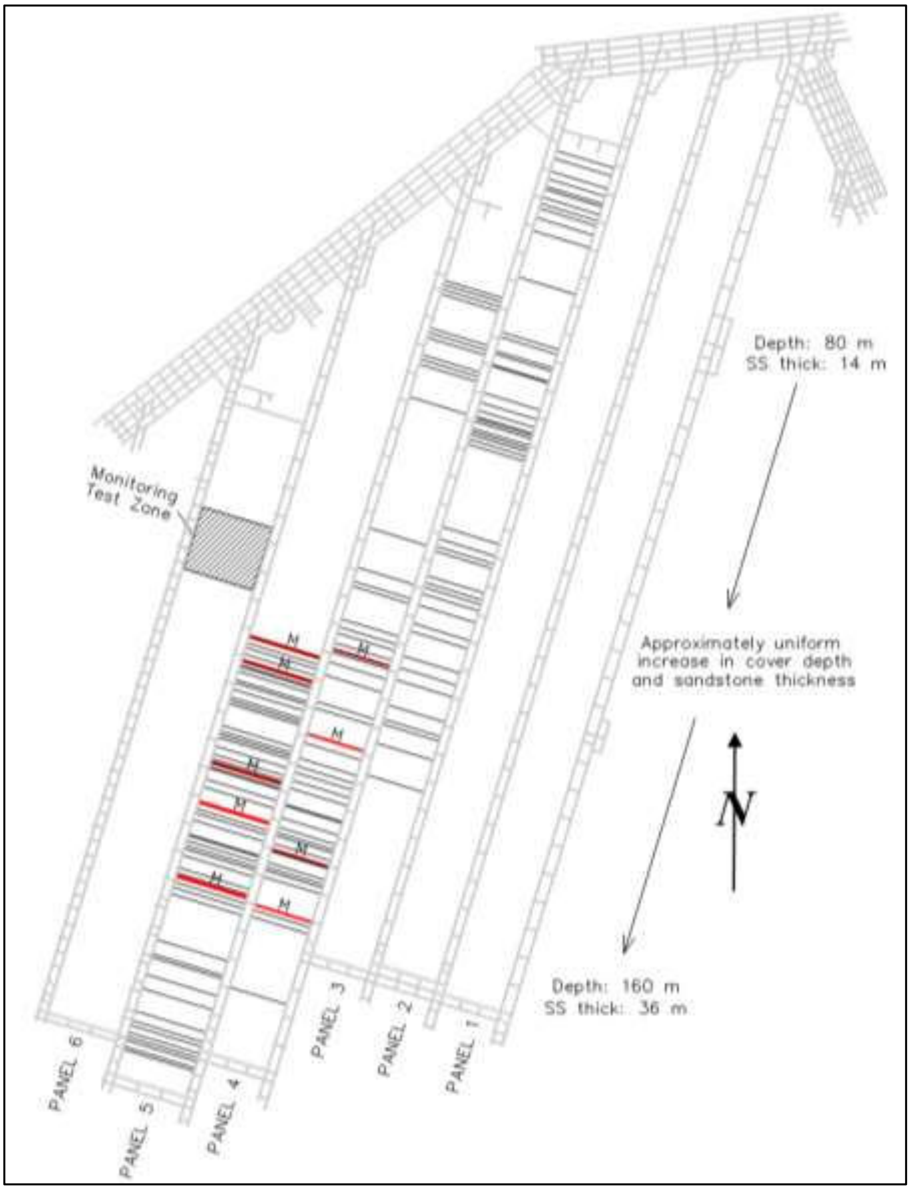

Figure 8. South Bulga face locations of reported weighting occurrences. Major weightings labelled M. From Sandford et al, 1999.

Similar weighting effects have been reported by Zu et al (2018) for Chinese mines.

Tower Colliery

Tower was a longwall mine in the Illawarra Coalfield, New South Wales, now part of the Appin complex. It mined a 2.4 m thick Bulli seam at a depth of 450 m. The overlying strata include units of sandstone, etc. including the thick Hawkesbury sandstone at the surface. This unit is massive and thickly bedded with well-developed vertical jointing. This unit exhibits concentration of high lateral stress and associated strain, particularly in valley floors.

The surface topography consists of steep-sided river gorges, up to 68 m deep covered mostly with natural bushland and traversed by a major road. Of particular concern for the extraction of LW16 and LW17, was the potential subsidence impact on a major highway bridge 600 m away and on twin-six-span box girder bridges with piers up to 55 m in height. Historically, lateral displacements of up to 350 mm had occurred with longwall mining and, because of the consequences of such movement, a great deal of preparation and monitoring work was undertaken to prepare the area for mining. This included replacement of the span mounts to take lateral movement.

The bridge moved as an intact unit by 100 mm towards the longwall panels with no effects on serviceability (Hebblewhite, Waddington & Wood, 2000; Hebblewhite, 2001). Horizontal movements and the bridge are shown in Figure 9.

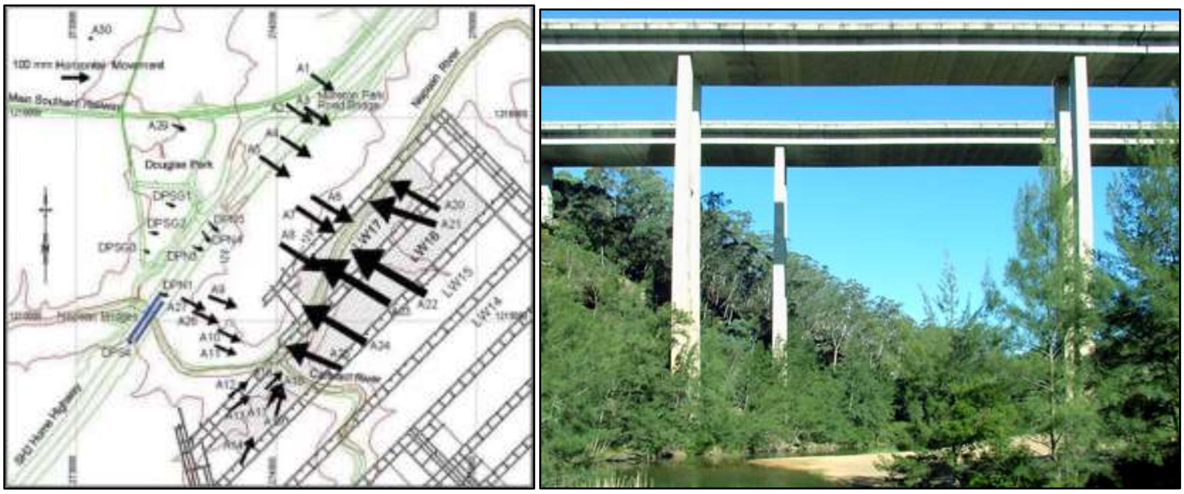

Figure 9. Tower horizontal movement from LW16 and LW17 and Nepean River bridge (Hebblewhite 2001)

Research Needs

The outstanding research need is the determination of changing conditions within the coal seam, especially those in the vicinity of faulting, intrusions etc. This information cannot be obtained by surface seismic surveys, nor by vertical boreholes which simply cannot be drilled frequently enough to determine these changes. What is needed is enhanced in-seam drilling. If this is to be conducted from underground, it is limited by the lack of geophysics (due to intrinsically safe requirements). While core drilling can provide a great deal of information, it is slow and, in most implementations, lacks directional control. Continuous conventional coring becomes impractical with hole lengths more than about 200 m. Wireline coring could be used and can be pushed to more than a kilometre in length, but issues associated with directional control need development. Another complication with wireline coring is that, in a gassy environment, well control practices become more complex to implement than with an open hole.

What is needed is an acoustic televiewer for use with in-seam holes. This needs to be augmented by the capability to go back into a directional hole and to be able to core off a branch where there are indications of changing structural features.

Conclusions

The reality of the interaction of geology, rock mechanics, gases and liquids in an underground coal mining situation can be extremely complex. There have been major advances in the understanding of mine geomechanics over the last twenty years but, in many cases, the implementation of these findings is slow to occur. There is also a tendency to want to simplify what is done for reasons of budget constraint. A failure to do so may lead to costs which are far in excess of the cost of undertaking geotechnical exploration properly.

Geotechnical and fluid measurements can be incorporated selectively into exploration programs in a cost effective manner. An understanding of the potential hazards and their mechanisms is an essential part of the design of these investigations and provides critical information for quantifying mine planning constraints. Effective design of these programs is dependent on a thorough understanding of potential mechanisms of failure, fluid storage, local structural control and stress regime. Anisotropy and directional changes in these factors are also critical to the development of a mine plan.

The cost effectiveness of understanding the geotechnical behaviour of a mine should be thought of in terms of the cost of the loss of a mine or the stoppage of a longwall due to weighting issues. The latter is likely to cost one million dollars a day. These costs make expenditure on drilling some holes and testing within them and on core become totally insignificant.

Observation for the Future

An observation gained from some 95 years of combined practice in the area of geology, mining and mine geomechnics by the authors is that there are currently too many specialisations and what is needed are people with a broad and deep knowledge of the entire process so that it can be brought together. The current splits within each profession and between professions fostered by the professional societies, and particularly those controlling professional registration, are extremely harmful to the industry. It is virtually impossible for someone to gain enough cross disciplinary experience within the current professional structure to ever gain an adequate breadth of experience to fully understand what is required to do the job.

References

Australian Standard, AS 3980 (2016). *Determination of gas content of coal and carbonaceous material—Direct desorption method*. SIA Global, November, 2016.

Beamish, B and Beamish, R (2011). *Testing and sampling requirements for input to spontaneous combustion risk assessment*, in Proceedings of the Australian Mine Ventilation Conference, pp 15-21 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Beamish, B B and Theiler, J (2017). *Recognising the deficiencies of current spontaneous combustion propensity index parameters*, in Proceedings of the Australian Mine Ventilation Conference, pp 113-117 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Beamish, B B and Theiler, J (2019). *Coal spontaneous combustion: Examples of the self-heating incubation process*, International Journal of Coal Geology, 215:103297.

De Weist, R.M (1965). Geohydrology. John Wiley and Sons Inc. ISBN 0 471 20410 2.

Gilbride L.T, & Hardy M.P. (2004). *Interpanel Barriers for Deep Western U.S. Longwall Mining*, 23rd International Conference on Ground Control in Mining, 3-5 August 2004, Morgantown. West Virginia, USA

ГОСТ 21153.5-88 (1988). *Method for the determination of rock shear strength under compression*, USSR State Publisher.

Gray Ian (1983, II). *Factors Influencing Outbursting and Gas Drainage in Underground Coal Mines*. Doctoral Thesis submitted to the University of New South Wales, School of Applied Geology, 348p.

Gray Ian (1987, I). Reservoir Engineering in Coal Seams, Part 1: *The Physical Process of Gas Storage and Movement in Coal Seams*. Society of Petroleum Engineers Reservoir Engineering, February 1987. SPE Paper No.12514. www.onepetro.org

Gray I. (2000). *The Measurement and Interpretation of Stress*, Bowen Basin Symposium 2000: The New Millennium - Geology Proceedings: Rockhampton, Queensland 22-24 October 2000 / J.W. Beeston, editor; organised by the Bowen Basin Geologists Group and the Geological Society of Australia Incorporated Coal Geology Group.

Gray, I. (2011). *Stresses in Sedimentary Strata, including Coals, and the Effects of Fluid Withdrawal on Effective Stress and Permeability*, 11th Underground Coal Operators' Conference, University of Wollongong & the Australasian Institute of Mining and Metallurgy, 2011, 297-306.
- http://ro.uow.edu.au/cgi/viewcontent.cgi?article=2032&context=coal

Gray I., Singh H. & O'Brien A. (2013). *Direct Measurement of Gas Content Without Coring*. SPE 167611. 2013 SPE Unconventional Resources Conference & Exhibition - Asia Pacific in Brisbane, Australia.
- https://www.onepetro.org/conference-paper/SPE-167611-MS

Gray I. (2014*). The Consequences for Pillar Stability of the Stress Path that Coals Undergo During Drainage*. AusRock 2014: 3rd Australasian Ground Control in Mining Conference, Sydney, NSW, Australia. 5-6 November.
- https://www.ausimm.com.au/publications/epublication.aspx?ID=16213

Gray I., Wood J., Shelukhina I. & Zhao X.. (2015), *Isotherms, Gas Contents and Diffusion Coefficients of Coals and Shales* – SPE177006 - MS Copyright. SPE Asia Pacific Unconventional Resources Conference and Exhibition, Brisbane, Australia, 9–11 November 2015.
- https://www.onepetro.org/conference-paper/SPE-177006-MS

Gray, Ian, and Neels, Bruce (2015). *The Measurement of Gas and Liquid Pressure in Rock and Soil.* Ninth International Symposium on Field Measurements in Geomechanics. 9-11 September 2015, Sydney.

Gray I. (2015). *The Anisotropy and Inhomogeneity of Coal Permeability and Interconnection of Adjacent Seams*. SPE-177007-MS. SPE Asia Pacific Unconventional Resources Conference and Exhibition, Brisbane, Australia, 9-11 November 2015.

Gray I. (2018), *The Practicalities of Effective Stress Measurement in Rock*, 4th Australasian Ground Control in Mining Conference, Australasian Institute of Mining and Metallurgy, 28 -30 November, Sydney, New South Wales, Australia

Gray I., Zhao X. & Liu L. (2018a). *Mechanical Properties of Coal Measure Rocks Containing Fluids at Pressure.* Proceedings of the 18th Coal Operators' Conference, Mining Engineering, University of Wollongong, 195-204. https://ro.uow.edu.au/coal/699/

Gray I, Zhao X & Liu L (2018b). *Anisotropic and Nonlinear Properties of Rock Including Fluid Under Pressure*. Proceedings of the European Rock Mechanics Symposium (Eurock 2018), Saint Petersburg. Russia 22-26 May 2018. Geomechanics and Geodynamics of Rock Masses, Volume 1. ISBN 978-1-138-61735-3.

Gray I., Zhao X., Liu L., & Wood J.H. (2018c). *The Young's Moduli, Poisson's Ratios and Poroelastic Coefficients of Coals*. ACARP Project C26061: Sigra Job 457 Final Report 20180829.

Gray I. & Gibbons T.R. (2020). *Longwall Behaviour in Massive Strata*. Proceedings of the 20th Coal Operators' Conference, Mining Engineering, University of Wollongong. https://ro.uow.edu.au/coal/757/

Gray I (2020). *Rock property determination*. Underground Mining Technology 2020 - J Wesseloo (ed.)© Australian Centre for Geomechanics, Perth, ISBN 978-0-9876389-9-1.Underground Mining Technology 2020 379. doi:10.36487/ACG_repo/2035_20.

Gray I., Wood J., Zhao X., Gibbons T.R. (2021). *New Outburst Risk Determination Measures Along With Data Gathering and Analysis for Coal Burst Assessment*. ACARP Project C26062. Sigra Job 456. Final report 20210524.

Gupta R N & Ghose A K (1992). *Strata Support Interaction on a Powered Support Longwall Face under a Massive Dolerite Sill – A Study*, Indian School of Mines, Dhanbad 826004, India, 11th International Conference on Ground Control in Mining, Wollongong, New South Wales, Australia, July 1992.

Hanes J (1979). Leichhardt Colliery, a geological assessment of the major coal burst in A North Intakes, 1st December 1978. Dampier Mining Company, internal report.

Hatherly P, Medhurst T, Ye G, and Payne D (2009). *Geotechnical Evaluation of Roof Conditions at Crinum Mine Based on Geophysical Log Interpretation*, in Naj Aziz and Bob Kininmonth (eds.), Proceedings of the 2009 Coal Operators' Conference, Mining Engineering, University of Wollongong, 18-20 February 2009 https://ro.uow.edu.au/coal/67

Hebblewhite B.K., Waddington A. & Wood J. (2000), *Regional Horizontal Surface Displacements Due to Mining Beneath Severe Surface Topography*, 19th Conference on Ground Control in Mining, 8-10 August 2000, Morgantown, West Virginia, USA https://www.onemine.org/document/abstract.cfm?docid=149869&title=Regional-Horizontal-Surface-Displacements-Due-To-Mining-Beneath-Severe-Surface-Topography

Hebblewhite B. (2001), *Regional Horizontal Stress Movements associated with Longwall Mining*, 5th Triennial Conference of Mine Subsidence, Maitland, New South Wales, Australia.

Hseih C-m (2013), *In situ stress reconstruction using rock memory.* Thesis presented for the degree of Doctor of Philosophy of the University of Western Australia, School of Civil and Resource Engineering.

Kennedy E.M. (Chairman) (2008), *Report on the August 6, 2007 Disaster at Crandall Canyon Mine*, Health, Education, Labor and Pensions Committee, United States Senate.

Kissell, F.N., McCulloch, C.M., Elder, C.H., 1973. The direct method of determining methane content of coalbeds for ventilation design. US Bureau of Mines, Report Investigation 7767, 17 pp.

Latilla J., van Wijk J., Booysen H. Silver C. & Fourie P. (2007), *Multi Seam Interactions of Relatively Shallow, High Seam Longwalls in South Africa*, 26th International Conference in Ground Control in Mining, 31 July – 2 August 2007, Morgantown, West Virginia, USA

Lu J., Su D.W.H., Hasenfus G.J., Van Dyke M. & Stull L. (2014), *Mitigation of a Massive Sandstone Channel's Impact on Longwall Face Evaluation and Enhancement of Hydraulic Fracturing Technique*, 33rd International Conference on Ground Control in Mining, 29-31 July 2014, Morgantown, West Virginia, USA

Mills K.W., Puller J. & Salisbury O. (2015), *Measurements of Horizontal Shear Movements Ahead of Longwall Mining and Implications for Overburden Behaviour*, 34th International Conference on Ground Control in Mining, 28-30 July 2015, Morgantown, West Virginia, USA.

Moore, R.D. and Hanes, J. (1980). *Bursts in coal at Leichardt Colliery, Central Queensland and the apparent benefits of mining by shotfiring*, in Proceedings of the Symposium on the Occurrence, Prediction and Control of Outbursts in Coal Mines, Australasian Institute of Mining and Metallurgy – Southern Queensland Branch, pp 71-83.

Ross T.A., Yu B. & Nyikos C. (2006), *Analytical Investigation of Shaft Damages at West Elk Mine*, 25th International Conference on Ground Control in Mining, 1-3 August 2006, Morgantown, West Virginia, USA.

Saghafi A (2010), *Novel Methods of Coal Seam Gas Content Determination for Estimation of Greenhouse Gas Emissions from Mining*. 27th Annual International Pittsburgh Coal Conference. 11-14 October 2010.

Sandford J. (1998), *Review of Longwall Mining Experience at South Bulga Colliery*, International Conference on Geomechanics/Ground Control in Mining and Underground Construction, 14-17 July 1998, Wollongong, NSW, Australia

Sandford J., Mahoney S., Conover D.P. & DeMarco M.J. (1999), *Shield Monitoring to Forecast Severe Face Weightings at the South Bulga Colliery*, NSW, Australia, 18th Conference on Ground Control in Mining, 3-5 August 1999, Morgantown, West Virginia, USA

Tadolini S.C. & Zhang Y. (2007), *The Unpredictable Life of a Coal Pillar*, 26th International Conference on Ground Control in Mining, 31 July-2 August 2007, Morgantown, West Virginia, USA

Tarrant, G. (2005), *Skew Roof Deformation Mechanism in Longwall Gateroads - Concepts and Consequences,* Coal 2005 Conference, 26-28 April 2005, Brisbane, Queensland, Australia

Wood, J. & Gray, I. (2015). *Outburst Risk Determination and Associated Factors.* Australian Coal Association Research Ltd. Report C23014.
- https://www.acarp.com.au/abstracts.aspx?repId=C23014

Yang X., Wen G., Dai L., Sun H. & Li X. (2019), *Ground Subsidence and Surface Cracks Evolution from Shallow-Buried Close-Distance Multi-Seam Mining: A Case Study in Bulianta Coal Mine*, published online 10 January 2019, Rock Mechanics and Rock Engineering (2019) 52:2835–2852 https://doi.org/10.1007/s00603-018-1726-4

Zhu W., Xu J., Ju J. & Qu Q. (2018), *Strong Weighting Events in Shallow Multi-Seam Longwall Mining*, China University of Mining and Technology, Jiangsu, 18th Coal Operators' Conference, 7-9 February 2018,Wollongong, New South Wales, Australia