LONGWALL BEHAVIOUR IN MASSIVE STRATA

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ABSTRACT: Longwall mining of coal below massive strata now has a 50 year history, and much of it has been problematic. For the first 30 years, there was much experimentation, with some successes, and many failures due to lack of understanding about how such massive strata behaves during caving, and how this differs from the more conventional and successful longwall mining beneath softer strata, which caves readily. This has been compounded over the last 20 years as mines extract wider faces and thicker seams in a single pass, often at quite shallow depths. Today, China is leading the way with thick seam longwalls under massive strata in shallow conditions, though not without problems.

Determination of the required capacity for longwall roof supports in such conditions is still not adequately understood, and overloading of supports remains a common problem in modern mines. The authors' view is that mathematical modelling to determine support requirements can only succeed if the model reflects real observed behaviour in such conditions.

There is an extensive body of technical literature documenting observations of what actually happens. This paper draws on past experience in many different countries to categorise common themes, and then proposes in simple forms the basis for real behaviour. Future modelling should be advanced on this basis. Such models need to incorporate a combination of multiple failure modes, including tensile, bending and shear, which may produce large rock blocks. These may move laterally, vertically, or in rotation depending on the loads which act upon them. Large blocks may impose substantial loading on the face, the powered supports or on the gateroad pillars.

The problems associated with large block formation may be mitigated by preconditioning to break up the massive strata before problems occur, so that manageable blocks are formed. In the United States of America and Australia, the focus has been on hydraulic fracturing. In Europe, China and South Africa, explosives have more often been used.

INTRODUCTION AND BACKGROUND

Longwall mining originated at depth in fairly weak roof conditions where the stress to strength ratio of the rocks was high. This meant that the rock failed, and readily formed a goaf behind the roof supports. Under these conditions, the major challenge was keeping the gateroads serviceable. The common way to do this was to permit deformation of the roadways by the use of wood lagged arched supports with yielding slip joints. Much of earlier longwall mining was conducted in thinner seams.

Modern longwall mining often occurs in completely different geological environments. In these, we find that the rock is frequently strong in comparison to stress, and therefore does not break readily. Longwall mining is now also frequently conducted at shallow depths where the stresses are lower. The combination of strong rock and low stress means that the rock mechanics focus changes from supporting broken rock, to dealing with unbroken rock, and in particular large blocks of rock which may not readily break, but when they do break, can move substantially.

The consequences of large rock blocks include:

- unpredictable weightings on the powered supports;
- rock levers developing over the face causing major face break problems;
- large voids at the face as blocks break off and slowly rotate and advance in the direction of the goaf;
- varying face stresses as the blocks apply loadings irregularly;
- rock (coal) bursts associated with the uneven block loadings;
- rock (coal) bursts that are the consequence of seismic events brought about by sudden massive rock failure;

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- sudden falls that generate wind-blasts;
- severe vertical loadings on gateroad pillars;
- shear movements on gateroad pillars; and
- uneven subsidence.

The importance of the size of rock blocks is inversely proportional to their distance from the longwall mining roof. For example, if a large cantilever of rock extends out over the longwall and then suddenly fails, its load will be transferred suddenly to the powered roof supports below. If this load is being transferred through already broken rock, then some of the loading will be distributed through this broken material and not directly on to the powered roof supports. How this transference takes place, and what design criteria should be used for powered roof supports, is really not resolved satisfactorily in any published literature.

Strong rock means rock strong in its mass behaviour. This means that it is massive as well as intrinsically strong. It is quite possible to have a strong rock type which is weak in its mass behaviour because of jointing. The real question when dealing with massive rock is "how big will the rock blocks that are formed be?" Rock breakage is primarily determined by jointing and planes of weakness. These still require some stress to break them apart. This can come from gravitational load or from initial stress redistribution. It should be borne in mind that any mining that removes stressed rock will have the consequence that the load (stress x area) it carried will be redistributed somewhere else.

There is an increasing trend to deal with massive rock units through pre-conditioning. This means forming planes of weakness within the rock mass so that goaf failure takes place readily. Pre-conditioning in coal mining may be brought about by hydraulic fracturing in multiple forms, blasting, or a variant thereof - high energy gas fracturing. These developments have progressed both in coal mining and more particularly in hard rock mining, where pre-conditioning is used in both block-caving, sub-level caving, and panel-caving environments. A great deal of the technology of hydraulic fracturing has come from the petroleum industry, where obtaining fluids from low permeability reservoirs became a major focus of interest.

This paper endeavours to explain some measured, and often very odd behaviour, of longwalls in massive strata. These problems affect the face, the powered supports, the gateroads, as well as the installation and recovery rooms. The problem appears to be due to large blocks of rock that move not only vertically, but also laterally. This results in block translation and rotation as the mass migrates in the direction of minimum stress. Such blocks can impose very large loads on the powered roof supports. They can also lead to face spall and pillar failure, including bursts. The cause of the problem is fundamentally the failure of the massive strata to break up. This seems to be worse under shallow conditions, presumably because of the lack of stress to break up the rock blocks.

The problem may be thought of as being at two ends of a spectrum. One has small broken blocks in the roof which behave more like a soil and which will bridge off a short distance above powered roof supports. This leaves the support carrying little load. The other being that of massive blocks which do not fail until a substantial face advance has taken place. These then fail suddenly and impose huge loads on the mechanical supports. The blocks also slew and may move into the adjacent panel's goaf, severely damaging the tailgate and pillar as they do so.

Longwall mining can be divided into cases of sub-critical and super-critical goaf formations. Where the longwall panel is comparatively deep and narrow, the rock bridges, or arches, across the top of the panel and transfers loads down to the gateroad pillars beneath. In this sub-critical case, the subsidence is minimised. Rock beneath the bridge needs to collapse to form the goaf. If it does not collapse readily then the problem of wind-blast occurrence is of immediate concern. Wind-blasts brought about by roof falls are one of the most dangerous events that can occur in mining. They can induce wind speeds that are a substantial fraction of the speed of sound, with the results that personnel are blown violently around the mine, equipment is dislodged and gas and dust are expelled out of the goaf, with the associated risk of an explosion initiated by friction or electrical damage.

The super-critical goaf is one where the rock within the goaf collapses and transfers most of the load directly through the broken material to the floor. Only where pillars exist is there some disruption to the loading. The formation of the super-critical goaf requires a face width to depth ratio that is large. This ratio has been described as being at least 1.4 but is highly dependent on the rock type.

It is quite possible to have an intermediate situation take place between critical and super-critical goaf formation, where the goaf has collapsed and where substantial lateral stress is re-established in the stratigraphic layers in the upper part of the goaf. The rock bridge dimension is too great to support the goaf, as in a sub-critical situation, but high lateral stresses do exist. This has been noted to occur along longwall panels, as opposed to transversely across them, where the effects of gateroad pillars tend to break up the re-establishment of stress (Gray, Wood & Shelukhina, 2013).

The increasing trend to mine thick seams using longwall techniques leads to more vertical disruption of the strata and a higher probability that lateral stress will not be transferred within the goaf. In cases where the lateral stress is not maintained during longwall mining, the strata can move substantially laterally. This lateral movement is dependent on where the rock may start to move from and the strain it contains prior to mining taking place. For example, if the rock is stressed so that it contains a lateral strain of 500 microstrain across a longwall panel and the panel has a width of 400 m, then the potential movement released by mining at the tailgate is 0.2 m over one panel width. However, this movement is not necessarily limited to a single panel but may be associated with a shear along the roof of the seam that extends to a far greater distance. This is a shear movement between roof and floor that will have to be withstood by the tailgate roadway and its pillar.

Ideally, a longwall mines coal with as little disruption as possible to production and the ground surface. In the case of a shallow mine, there is no sensible choice but to aim for super-critical panel widths. This can be argued on simple economic grounds, as the number of gateroads that would be required for sub-critical mining would mean that it would be pointless having a longwall at all. Wide super-critical panels reduce the disruption to the ground surface associated with gateroad pillars. This disruption may be reduced further if yield pillars are used, as they will crush. The question is whether a yield pillar can be made to protect the tailgate during mining and crush when it becomes part of the goaf?

Because of the inevitable consequences of large-scale rock movements that accompany longwall mining, it is important to design any form of support to behave in a ductile manner. This is achieved by having adequate hydraulic set and yield pressures on the powered roof supports of the longwall face, and most importantly maintaining a reserve displacement capability in the powered roof support hydraulic cylinders once yielding takes place.

Ductile design is no less important for the gateroads. The support system must be able to withstand deformation and survive. This deformation that the gateroads should withstand may include:

- floor heave,
- rib failure of the solid block or or pillars,
- shear of the roof with respect to the floor (taken within the pillar),
- shear within the roof itself,
- compressive failure of the roof,
- tensile failure of the roof,
- flexure of the roof, and inevitably
- slabbing where laminated materials exist.

Any support also needs to be able to deal with jointing or faulting that may be encountered, though support is frequently modified where varying conditions are encountered. It is dangerous to design support for one expected mode of failure when in fact multiple types of loadings may take place.

The use of secondary “standing” roadway support in the form of concrete filled “cans”, stone filled wood cribs, or other props is regarded as a last resort because it will severely restrict the serviceability of the roadway, not least for ventilation. The exception is in the use of normal or, self-advancing, powered roof supports at the end of the tailgate and for the extraction of the longwall at the end of a panel in a recovery room.

The art of rock mechanics in longwall mining may be summarised by a simple statement - keep the roof up safely while you are working below it, and get it down as quickly as possible once it is behind the face.

EARLY LONGWALLS

In South Africa, early longwall experiments were undertaken at Durban Navigation in the Klip River Coalfield. Two 1.0-1.2 m seams were mined sequentially under a dolerite sill. The mine experienced sudden unpredictable weightings, erratic surface subsidence, bad falls, methane, and floor heave, along with serious injuries and fatalities (Deats, 1971). From 1976-1980, Coalbrook in the Vaal Basin Coalfield mined the 2.2-2.8 m No.1 seam with roof lithology being competent sandstones and a 40 m thick dolerite sill. The weight of dolerite in cantilever caused breaks in the roof and coal ahead of the face, often associated with flooding (Henderson, 1980). Sigma, also in the Vaal Basin Coalfield, mined the No.3 seam at 115 m depth under a 40 m dolerite sill 50-80 m above the coal. The face experienced excessive weightings, severe coal face spall and flooding (Cloete, 1980). More recently, Matla in the Witbank Coalfield operated shortwalls in the No.2 and No.4 seams. In 2002, the No.4 seam experienced extreme loading of the tailgate pillars. A large dolerite sill block, having detached itself over the maingate, was hanging up some 300 m behind the tailgate over the goaf. Surface pre-split blasting was successfully used to relieve the pressure, resulting in mitigation of facebreak and wind-blast events (Latilla, 2007).

In India in 1990, a state-of-the-art longwall was commissioned at Churcha West in Chhattisgarh to extract a 3.0-3.4 m seam at a depth of 223 m (Deb 2004). It was equipped with a data logger to measure support behaviour. The immediate roof was 80-133 m of sandstone overlain by 112-137 m of dolerite. When the face had advanced 198 m, a weighting destroyed half the 101 Gullick-Jessop 4/680 t chock-shields in 3.6 seconds. Coal and sandstone filled the face to canopy level. Coal between chocks #52-60 was crushed to powder. Similar catastrophic failures involving massive sandstone have occurred at Jhanjri in West Bengal and Godavarikhani ("GDK") 11A in Telangana after 204 m of retreat (Deb 2004).

In Canada, Phalen, in the Sydney Basin, Nova Scotia, began extraction of the 7 East Panel in December 1994. The panel mined the 1.5-3.0 m Phalen coal seam with a 260 m wide face and 3400 m length under the sea 200-700 m below sea level. There were previously flooded workings above the Phalen seam. Over a 30 month period until May 1997, 48 weighting events occurred with five being very serious. The planned extraction period was 18 months. The weightings were caused by the Lower Sandstone Unit (LSU), a massive paleo-sandstone river channel which formed the main roof overlying the weak immediate roof. No weightings occurred when the immediate roof was > 7.5 m thick and LSU < 9.5 m thick. However, when LSU > 9.5 m thickness severe weightings occurred accompanied by violent face spalling (0.5-1.0 m) and loud noise. The first weighting occurred 35 m past the start of the previous and adjacent 6 East goaf. The first serious weighting occurred 30 m past the start of 5 East goaf. The thicker the LSU became, the greater the weighting interval length (MacDonald, 1997).

Australia has also had its share of problems. Ellalong, in the Cessnock Coalfield, commenced operations in 1983 after extensive geotechnical investigations. A 3.0-4.7 m seam at a depth of 320-640 m was overlain by a main roof of thick, strong, massive sandstone, and immediate roof of 6-16 m of conglomerates, sandstones and shales. The first two longwalls experienced severe, rapid weightings and heavy coal face spall (Wold, 1986).

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 UCS of 40-80 MPa and modulus of 13 GPa. The powered roof supports were a combination of 940 t and 1150 t, 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 (Sanford, 1998 & 1999).

In the 1990s, wider and more productive longwall faces were introduced in the Newcastle Coalfield, where strata incorporates massive sandstones and conglomerates, some of which formed the immediate roof of the coal seams. The unexpected implications of introducing wider panels included more variable and increased subsidence, severe periodic weightings with associated mid-face falls, as well as wind-blast. In the late-1980s, Teralba in the Newcastle Coalfield experienced surface subsidence four times that previously experienced as a result of increasing face width to 150 m. Similarly, Newstan, near Teralba, experienced seven severe weightings and falls in LW5 on a 225 m face resulting from mining under up to 50 m of conglomerate. A decision to split LW6 into two 90 m faces solved the weighting problem, but induced 80-140 kph wind-blasts (Creech, 2014).

LARGE DETACHED BLOCKS

In 2002, Matla, in the Witbank Coalfield, Mpumalanga, South Africa, operated shortwalls on the No.4 and No.2 seams, and the No.5 seam had been shortwalled in the past. On 13 April 2002, the No.4 seam shortwall suffered a facebreak restricted mainly to the tailgate side due to a thick, near-surface dolerite not breaking. The resultant overloading led to severe pillar damage and subsequent roadway collapse. This was partly due to the premature suspension of longwalling in the overlying No.5 seam, resulting in No.4 seam overloading with severe pillar damage and collapse. The roof collapsed to a height of about 2 m ahead of the face in the tailgate of shortwall panel 8 (Latilla, 2007).

A dolerite sill close to surface had been observed in the past to break in large blocks within the No.5 seam and also overhang as much as 40 m from the goaf edge. The No.5 seam had been longwalled to the west of No.4 seam panel 8. The No.4 seam panel 8 was the first to shortwall under unmined No.5 seam and an intact sill. Higher than usual pillar stress occurred when tailgate 8 was the maingate for panel 7. Field observations soon after the face break revealed that the dolerite was breaking around 40 m in from the maingate pillar edges but that there was no sign of it breaking along the tailgate side. Extreme loading on the tailgate pillars indicated that the dolerite, having detached itself from the maingate side, was overloading the tailgate pillars. It appeared that the sill was hanging up for about 300 m behind the face on the tailgate side.

Underground, on the tailgate side, it appeared that the goaf of panel 8 was moving towards panel 7 across the two tailgate chain pillars. Cracks on the surface also indicated the same behaviour. It was decided to assist this tendency by blasting a pre-split in the dolerite on surface. Two lines of blast holes were used, the first diagonal to the face to assist the dolerite to break along the surface crack and another parallel to the tailgate to reduce dolerite loading on the tailgate chain pillars. Pre-split blasting used closely-spaced (3 - 4 m) holes.

Line 1 was blasted on 8 May 2002, causing noticeable reduction in load on both the face and tailgate pillars. This reduction in load was evident by the virtual absence of noise from the pillars and roof where previously there was constant noise and pillar spalling. Average powered roof support leg pressures for the tailgate side showed an increase after the pre-spilt from 19.5 MPa to 26.6 MPa. This appeared contradictory, and a close inspection of pressure readings revealed that the pressures were more uniform after the blast. The maximum and minimum pressures before the blast were 38.3 MPa and 14.9 MPa respectively. Within 30 minutes of the blast, movement was observed on surface with appreciable steps 5-10 cm high forming along cracks. No damage was observed along the shortwall face. An area with cracked roof ahead of this fall remained unchanged. Lines 2 and 3 were blasted on 28 May 2002.

Surface damage from the second blast was severe. The second blast was successful, with goaf falling within 30 minutes. No roof damage was noticed on the tailgate or the face. Shield leg pressures on the maingate side dropped from 32.1 MPa to 28.9 MPa. Blasting relieved the leg pressures and preconditioned the next two panels, with reduction in face break and wind-blast events.

a) Dolerite thickness contours (b) Presplit lines (c) 2-3 m surface subsidence

Figure 1. Matla dolerite presplitting (Latilla 2007) and 2-3 m surface subsidence (Hebblewhite, 2013)

GATEROAD ROOF DIVERGENCE (UPLIFT)

In February 2005, Enlow Fork, in the Northern Appalachian Basin, Pennsylvania experienced a massive sandstone caving event in the LW6-R tailgate. The mine had previously experienced large caving events associated with initial weighting in thick sandstone. However, this event was not related to an initial weighting. Prior to the incident, the tailgate was relatively quiet with no audible fracturing ahead of the face. However, there was significant floor heave which was normal. Overburden depth was 670 m. The event began with a loud bang, followed by a wind-blast.

Different theories emerged, but a consensus was that a high-strength massive sandstone channel was the primary instigator. Similar conditions were found in the LW7-R gateroad which might affect LW8-R. It was decided to instrument LW8-R tailgate. Convergence stations were installed from cut-throughs 26 to 7. Typical immediate roof was dark, silty shale overlaid by two distinct sandstone strata (lower and upper sandstone). In the vicinity of LW6-R, the roof geology was typified by 6 m thick first sandstone and 9 m thick second sandstone. Top of coal to bottom of sandstone was less than 0.6 m. The interval from the top of the coal to sandstone is important because of its potential to impart wind-blast, and also any bulking of weaker rock below cushions and mitigates wind-blast (Akinkugbe 2007).

During the monitoring programme, an unusual tailgate roof divergence (uplift) was recorded.

This roof behaviour contradicts expected norms of roof convergence when the longwall is influenced by periodic weighting. The recorded roof movement data indicated that the roof divergence started well ahead of the longwall face. The most inbye position transducer (cut-through 26) started recording progressive roof divergence when the longwall face was 196 m away. Maximum roof uplift recorded was 22 mm when the longwall face was 91 m from the transducer; thereafter it began to gradually converge. Accelerated convergence only started after a massive caving event with the face within 85 m of cut-through 26. The event resulted in mild wind-blast, with no damage or injury on the face or gateroads. Prior to the event, the roof of the tailgate was observed to be hanging 45 m behind the 15 powered roof supports nearest the tailgate.

After the event, mine roof displacement recorded by transducers at cut-throughs 22-26 were examined. While the transducers at cut-throughs 22-25 recorded minimal to negligible convergence, that at cut-through 26 recorded an unusual roof divergence (roof uplift).

After much consideration by the mine personnel, it was agreed that the most likely explanation appeared to be classic beam bending resulting from a roof cantilever effect. It was surmised that the weight of the rock in the hanging sandstone was sufficient to cause a flex in the sandstone resulting in an upward movement of the immediate strata. Although the exact location of the caving event is not known, anecdotal evidence and logic suggest that it most likely happened on the adjacent LW7-R goaf. Face conditions at the tailgate were relatively normal, not exhibiting excessive loading.

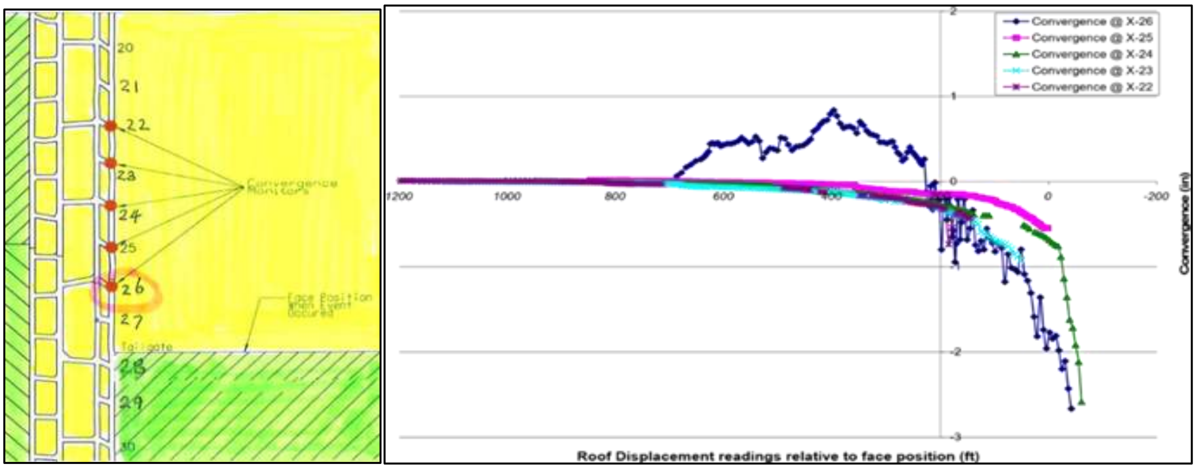


Figure 2. Enlow Fork roof lithology (Akinkugbe 2007) and roof displacement in LW8-R tailgate cut-throughs 22-26 and 26 (Akinkugbe 2007)

FARFIELD HORIZONTAL MOVEMENT

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 surface topography consists of steep-sided river gorges, up to 68 m deep. The surface is mainly natural bushland traversed by a major road which crosses one of the gorges on twin-six-span, box girder bridges with piers up to 55 m in height. Although the road and bridge are outside the conventional concept of angle of draw subsidence influence criteria, and have experienced negligible vertical deformation as a result of mining, there is widespread evidence of regional horizontal deformation of the land surface large distances away from the mining area. Gorge closure and evidence of large headlands moving en masse have been observed. Horizontal movements at Tower up to 350 mm have been recorded.

Of particular concern for the extraction of LW16 and LW17 was the potential subsidence impact on the bridge 600 m away. Overlying strata consists of sandstones, shales, claystones and mudstones. Some of these strata are quite massive and thickly bedded but with dominant vertical jointing persisting through most horizons. Tower has a high ratio of horizontal to vertical pre-mining stress up to 3:1. Most horizontal movement took place towards the gorge and the active goaf, with some movement towards the old goaf. The bridge moved approximately 100 mm en masse towards the longwall panels, fortunately with no impact on serviceability. (Hebblewhite, 2001).

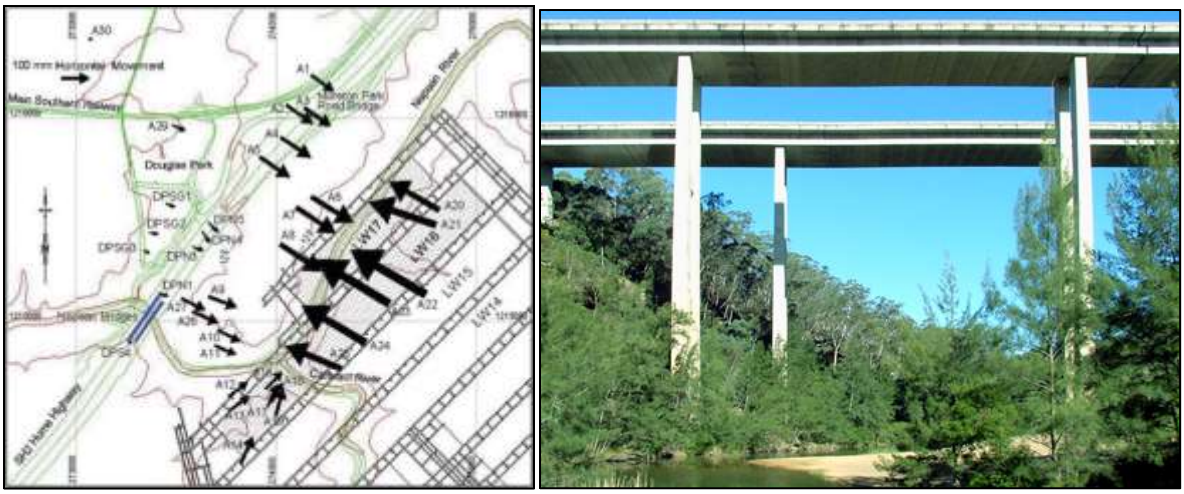


Figure 3. Tower horizontal movement from LW16 and LW17 and Nepean River bridge (Hebblewhite 2001)

Narrabri is a mine operating in the Gunnedah Basin, New South Wales. The mine targets a 4 m section of a 4-8 m thick seam 160-180 m deep directly below a 16-20 m thick conglomerate with a history of significant periodic weighting events. The face width is 306 m. Adjacent longwalls are separated by 30 m wide gateroad pillars. As part of an investigation to better understand the weighting events, inclinometers capable of measuring horizontal shear movements through the full section of the overburden strata were installed ahead of mining at two locations 1 km apart, above the centre of two adjacent longwall panels. Horizontal shear movements were observed to develop on shear horizons that correlate closely across the two sites. The horizon on top of the conglomerate mobilised almost immediately after initial deformation of the longwall goaf 425 m ahead of the face. The direction of horizontal movement was consistent with the relief of the major principal horizontal stress. As the face approached each inclinometer, other horizons within the upper overburden began to shear with the upper strata moving further towards the goaf than the immediately lower strata. Within the last 30 m, tilting of the strata associated with the onset of vertical subsidence caused reverse shear offsets (Mills 2015).

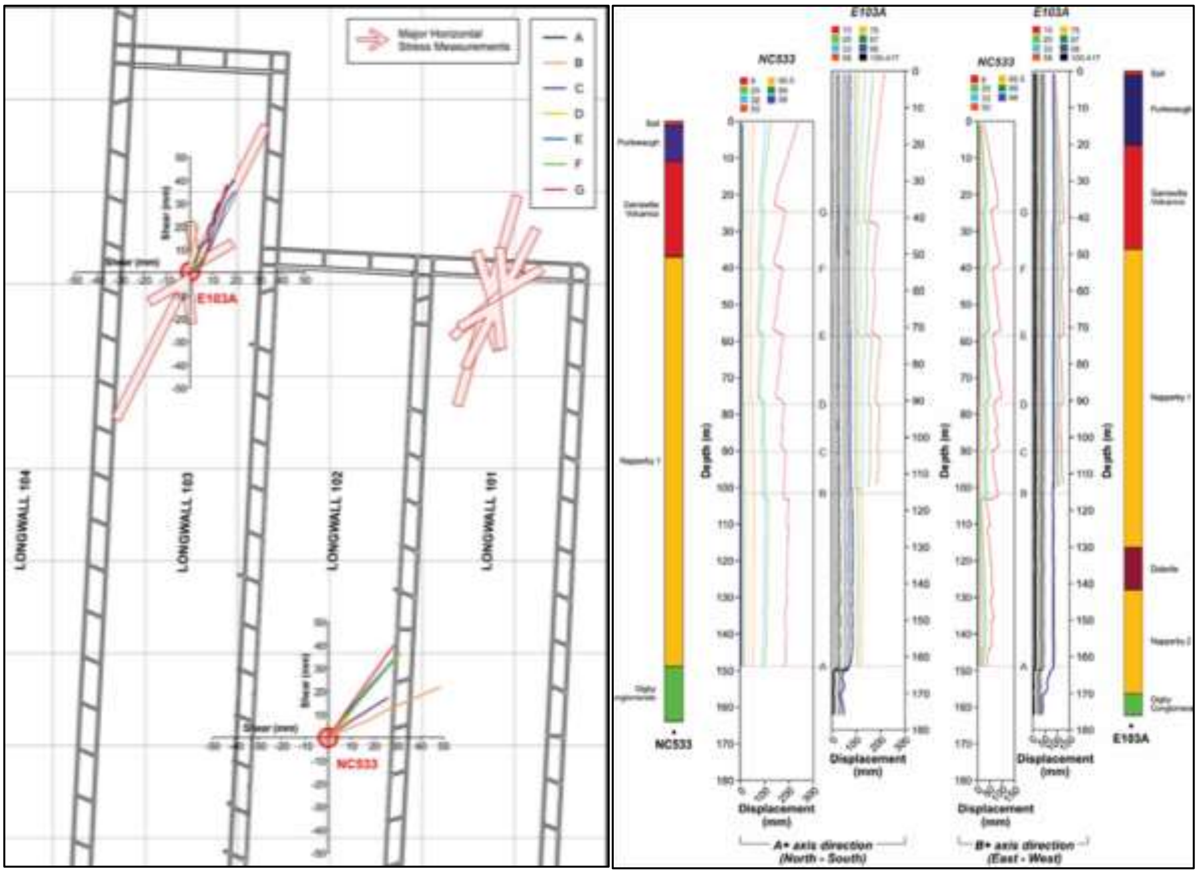
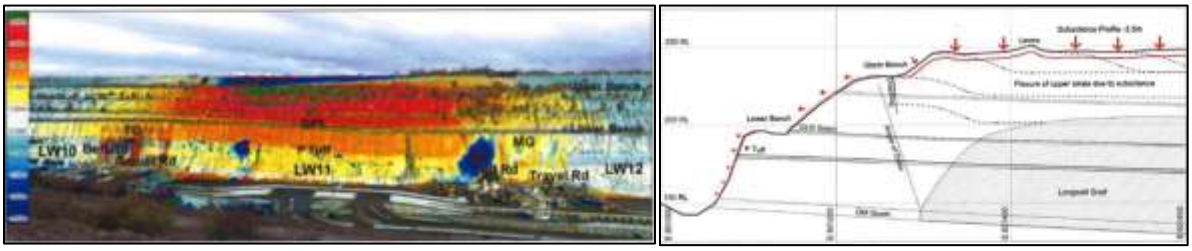


Figure 4. Narrabri major horizontal stress and horizontal movements (Mills, 2015)

Broadmeadow, located in the Bowen Basin Coalfield, Queensland, is a punch longwall mine. It has experienced significant highwall movement associated with the effect of longwall subsidence when the longwalls approach their final position close to the open cut highwall. In response to this movement, the mine deploys two types of broad scale highwall monitoring, using both radar and laser scanners. Results from the monitoring found the highwall is displaced to magnitudes unlike those typically measured in open cut mining, and in direct contrast to typical longwall subsidence behaviour.

During the mining of LW11, borehole inclinometer monitoring confirmed that initial horizontal shear movement was towards the centre of the longwall goaf. However, as the longwall approached the final position near the opencut highwall, the ground deformation did not conform to either typical longwall subsidence profiles, or typical highwall movement, with values far exceeding any stability limits in adjacent open cut mines, indicating the onset of slope failure. This outward movement in excess of

1000 mm, shown in green, at the top of the highwall destabilised local areas. Falls are shown in blue as negative horizontal movement. White indicates no movement (Payne, 2019).



(green >1000 mm, red > 800 mm, orange > 400 mm, white zero mm, blue <0 mm)

Figure 5. Broadmeadow LW11 horizontal and vertical displacement into opencut (Payne, 2019)

GATEROAD SHEAR (SKEW ROOF)

A deformation mechanism termed "skew roof" relates the regional influence of differential horizontal strata movement (shear) about longwall extraction to gateroads. Under geological and mining conditions where the skew roof mechanism operates, strata units move progressively further towards the goaf with increasing height into the roof. Skew roof has implications for chain pillar design and indeed all roadways within the vicinity of longwall extraction, including the face itself. At Metropolitan, in the Illawarra Coalfield, New South Wales, horizontal movement was so severe that the immediate roof material was essentially pulverised and flowed out of the roof space between the installed standing supports. This roof behaviour is evident in many Australian coal mines that experience poor roof behaviour, either adjacent to longwall extraction (travel roads), or during approach of the next longwall (tailgates). The propensity to skew is a consequence of strain relief and horizontal strata movement towards the goaf. These movements frequently progressively increase from seam to surface.

The effect is regional in that horizontal movements on the surface can extend in the order of kilometres from longwall mining, and at seam level the influence can extend over many tens of metres and potentially hundreds of metres. The direction of skew is the net influence of the direction from the roadway to the goaf, and the direction of the maximum horizontal stress (Tarrant 2005).

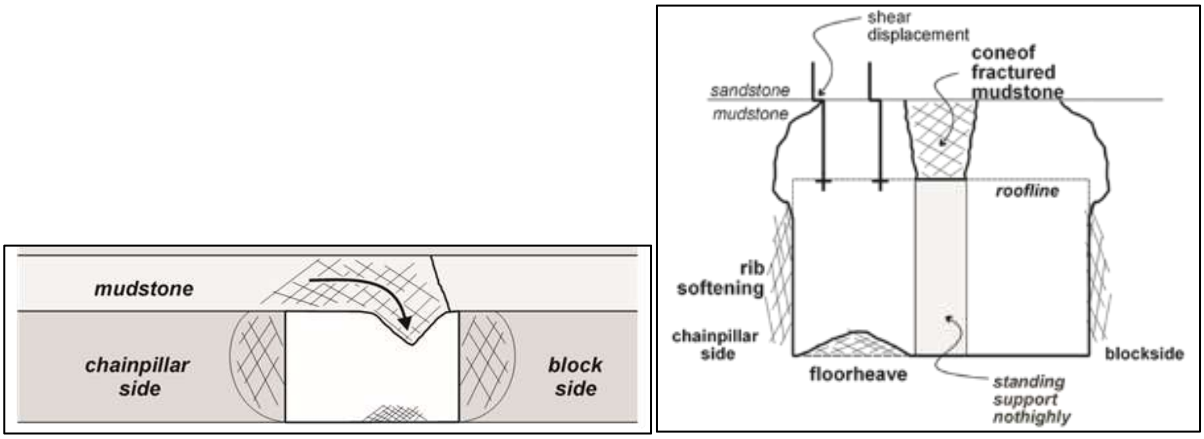


Figure 6. Metropolitan Observed Tailgate Behaviour (Tarrant, 2005)

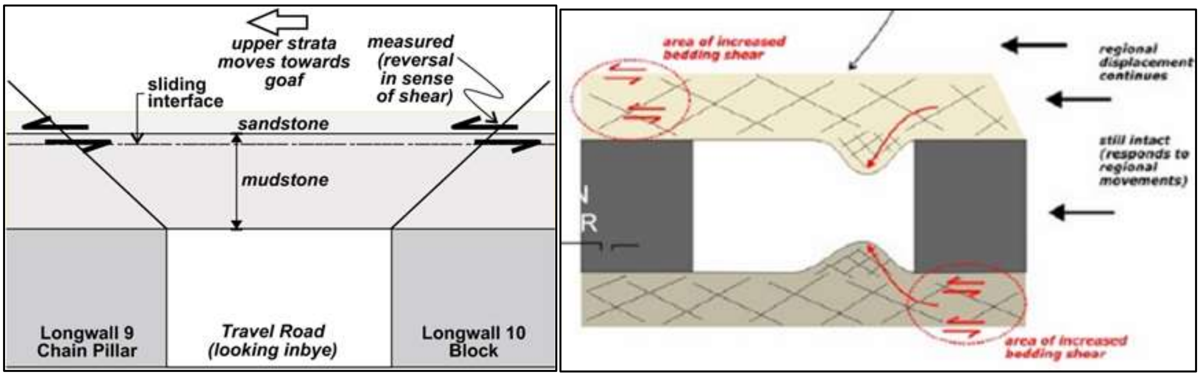


Figure 7. Skew Roof Concept (Tarrant, 2005)

Austar, previously Ellalong, in the Cessnock Coalfield, New South Wales, has a long association with difficult strata conditions due to mining depth, and a highly jointed / cleated coal. Matters of concern include poor longwall face conditions, cyclic loading, heavy tailgate conditions, difficulty in maintaining roadways on development, and in the installation and recovery of longwall faces. The depth of cover is 530 m. The mine experiences significant pressure bumps during development, typically in association with stiffer rock units above and below the seam. The most severe weightings occur when, on cycles between 120-150 m when a Branxton Formation (conglomerate-sandstone-shale) weighting event coincides with a weighting due to the immediate sandstone channel (Moodie 2011).

Figure 8. Austar gateroad centreline bagging and chain pillar side guttering near bent tendons (Moodie, 2011)

WIND-BLASTS

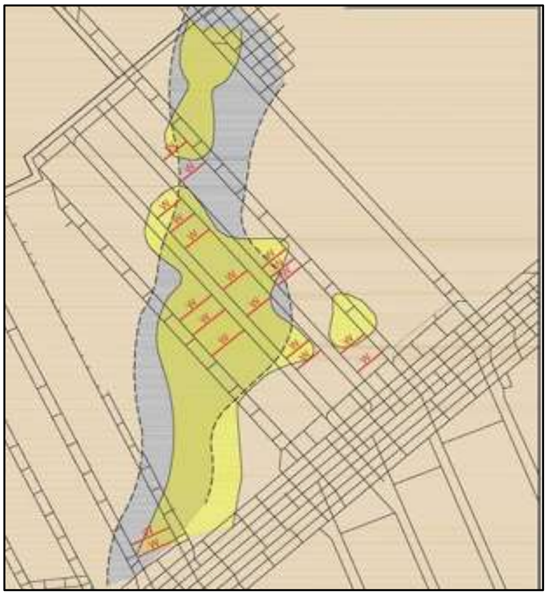
A wind-blast comprises a rapid rise in absolute pressure to a maximum (positive compression phase), usually followed by a similarly rapid fall to below ambient atmospheric pressure ("suck back"). After decreasing to a minimum value, the absolute pressure gradually increases until it becomes equal to the ambient atmospheric pressure. At around the same time, although not necessarily in phase with the overpressure, the wind velocity also rises rapidly to a maximum and then frequently exhibits a sudden reversal into the "suck back" phase. Each event usually lasts for a few seconds. From wind-blast monitoring and the recorded overpressure and wind velocity histories, it has been observed that there is no acoustic precursor to the event in the roadway other than that from "roof talk". Consequently, people in the working place will receive no warning of the wind-blast before it strikes them (Sharma, 2004).

An analogy often used to describe wind-blast is that of a "leaky piston" where a large intact rock mass initially drops, creating compression below it and a corresponding vacuum above it. When the rock fall ceases, air then "sucks back" into the vacuum above the fall.

Newstan and Moonee mines are located in the Newcastle Coalfield, New South Wales. The wind-blast events at Newstan were localised and confined to where the massive strata was within twice the extraction thickness and bridged the panel. In contrast, at Moonee, incidence of large goaf falls and associated wind-blasts continued for virtually the whole length of the longwall panels other than a few localised faulted zones where regular caving took place. The lower magnitude of the wind-blast parameters at Newstan are due to the fall of the roof element being cushioned by the caved immediate roof, lower volume of air being displaced from the void, and the higher resistance to flow due to partial packing of the goaf by the prior caving of the immediate roof.

Table 1: Newstan and Moonee wind-blast parameters (Sharma, 2004)

Parameter	Maximum Value	
	Newstan	Moonee
Peak air velocity	40 m/s	123 m/s
Rate of rise of velocity	50 m/s^2	138 m/s^2
Peak over pressure	10 KPa	34.5 KPa
Rate of rise of pressure	25 KPa/s	36.4 KPa/s
Impulse	20 KPa.s	89 KPa.s
Maximum excursion (air flow distance)	67.2 m	184 m



W – wind-blast location, grey – channel > 40 m thick, yellow – septum < 7 m thick

Figure 9. Newstan wind-blast events and associated massive strata (Creech, 2014)

SHALLOW (NEAR SURFACE) LONGWALLS

A large number of shallow coal seams of Shendong Coalfield, Shaanxi, China have been mined in recent years. These have posed serious strata control problems and have been studied (Zhao, 2018). Jinjie coal mine is one of these. Longwall LW31109 mines a 3 m thick seam at a depth of 120 m, with 3.3 MPa vertical stress. Face width is 280 m. There are 162, 1.73 m wide powered roof supports with a maximum working resistance of 12000 KN (1225 t) at 51.6 MPa hydraulic leg pressure. This corresponds to a load per unit width of 6940 kN/m (707 t/m). The setting pressure was 25.2 MPa, but during caving of the main roof the load of some powered roof supports exceeded the maximum support working resistance. Roof collapse and powered roof support failure occurred frequently. The first weighting distance and the periodic weighting distance of main roof have been analysed from November 2013 to June 2014. Immediate roof caving occurred at 63 m. Initial weighting of the main roof occurred when the face had advanced 80 m. The pressure in most of the powered roof supports exceeded 35 MPa with and average pressure of 40 MPa. More than half of the powered roof supports were overloaded during the first weighting.

Support pressure (MPa)

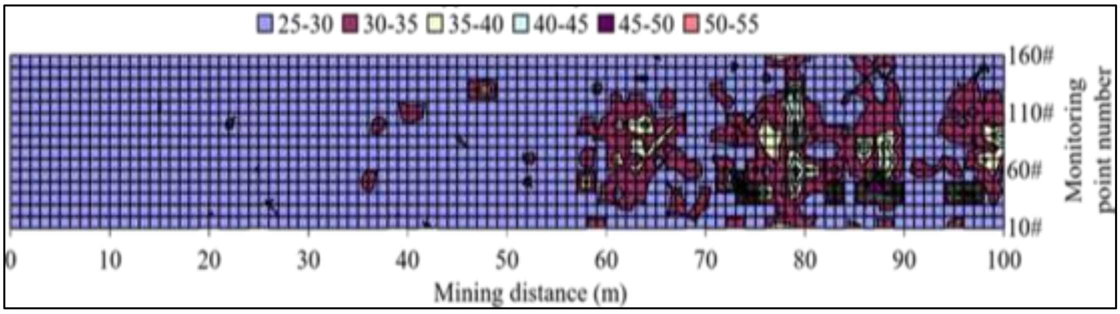


Figure 10. Hydraulic leg pressure at start of Jinjie longwall LW31109

In June 2014, when mining advanced from 3045 m to 3160 m, powered roof support pressure distribution was as shown in Figure 11. As can be seen the loadings became problematic with support No 80 reaching a pressure of 52.2 MPa which was above the design capacity of the supports.

Support pressure (MPa)

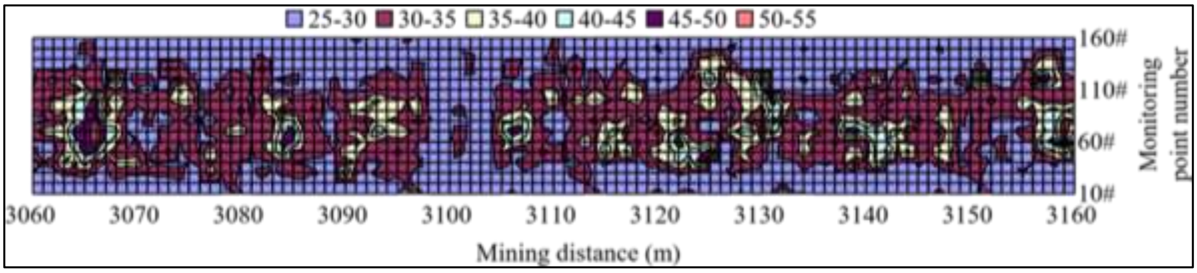


Figure 11. Hydraulic leg pressure from 3060 to 3160 m of Jinjie longwall LW31109

Table 2: Jinjie material properties of the coal strata (Zhao, 2018)

Position	Thickness (m)	Density (Kg/m^3)	Shear modulus (GPa)	Bulk modulus (GPa)	Cohesion (MPa)	Friction angle (^{o})
Main roof	6.0	2,550	4.7	6.0	1.2	30
	3.0	2,700	1.6	3.4	1.6	30
	1.5	2,650	1.7	3.5	1.7	32
Intermediate roof	4.5	2,460	2.0	3.2	1.1	18
Coal seam	3.0	1,400	1.5	2.8	0.6	20
Floor	2.0	2,650	1.7	3.5	1.7	32

Shangwan mine is located in the Shendong Coalfield, Inner Mongolia. Longwall LW51104 mines a 6.5 m thick seam at a depth of 115 m, with face width of 301 m and dip angle of 0-5^o.

Table 3: Shangwan gross material properties of the coal strata (Wang, 2018a)

Name	Unit weight (KN/m^3)	Elastic modulus (GPa)	Poisson's ratio	Cohesive strength (MPa)	Tensile strength (MPa)	Friction angle (^{o})
Sandstone	25.0	36.5	0.22	2.6	1.5	30
Coal	13.1	12.7	0.29	1.2	0.6	27
Siltstone	24.6	37.9	0.20	4.5	3.0	40

There are 11 different lithologies in the Jurassic, Cretaceous and Quaternary rock strata.

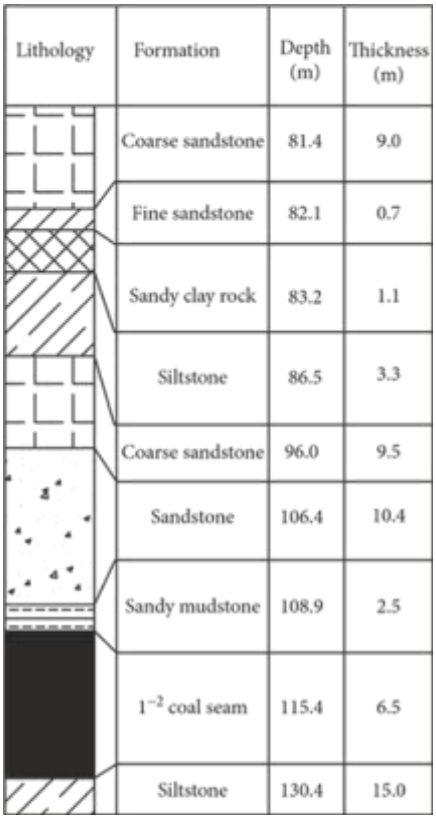


Figure 12. Shangwan lithology (Wang 2018a)

Wang, (2018a) reports an observation period during which 20 weightings occurred with intervals of 9.4-32.3 minutes with an average 16.7 minutes. Support resistance varied from 5571-8975 KN (559-901 t) averaging 7107 KN (713 t). The support's rated working resistance was 8638 KN (867 t). During coal mining beneath the thick bedrock and overlying loess layer, the weightings regularly alternated between strong and weak. As part of the periodic weightings, short time intervals corresponded with strong weightings, but normal time intervals had weights which could be either strong or weak.

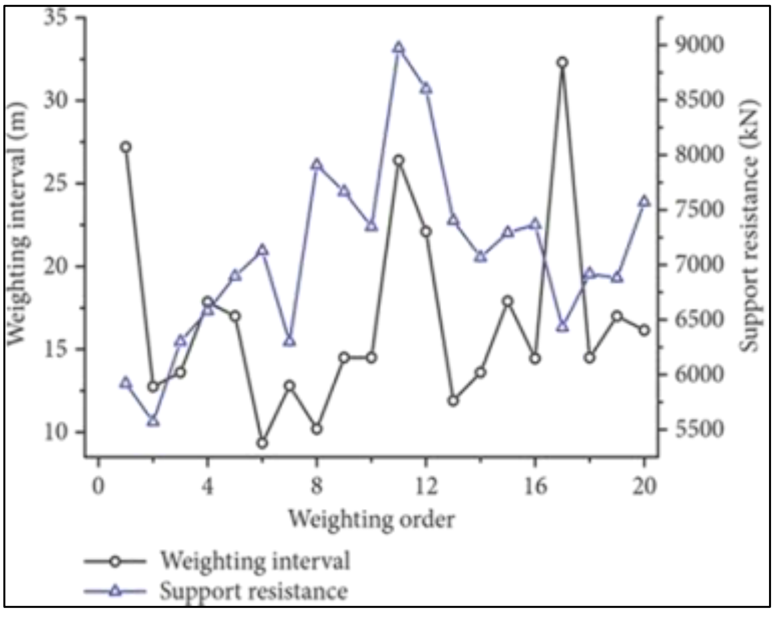


Figure 13. Shangwan weighting order (time sequence) vs weighting interval (m) and support resistance in kN (Wang 2018a)

SUBSIDENCE IN SHALLOW LONGWALLS

Bulianta, in the Shendong Coalfield of Inner Mongolia, annually produces the most coal for any underground coal mine in the world. It is characterised by multiple, thick coal seams, shallow depth, and thick loess soil layers. Currently, the mine is extracting 30 Mtpa from the 2^{-2} seam by longwall methods from under the previously mined bord-and-pillar workings in the 1^{-2} seam. LW22307 was mined in the 6.8-7.7 m thick 2^{-2} seam at a depth of 100 m with dip ranging from 1-3^{o}, a mining height of 6.8 m, a face width of 300 m, and a panel length of 4954 m. Soil 8 - 23 m thick covers the whole panel. During the mining of LW22307, a detailed study was undertaken because of the severe surface damage at the mine resulting from subsidence and associated cracks.

Subsidence after mining from under a previously mined coal seam is enhanced, which results from the consolidation of the ground, closure of existing cracks, and the stress release in the previously mined area. Ground cracks, as the other product of ground deformation, not only have a great impact on the stability of surface structures, but also threaten the productivity of land and affect the safety of residents in a mining area (Yang, 2019). Figure 14 shows the subsidence along the centre line of the longwall. The zone labelled 0 to 170 m was under a previous goaf while that from 170 to 400 m was under previous pillar workings. The subsidence was reasonably symmetrical between chain pillars. The maximum subsidence was 5.85 m associated with extracting the 6.8 m seam. This subsidence was not an even flexure of the surface but occurred with large surface cracks and block movement as can be seen in Figure 15.

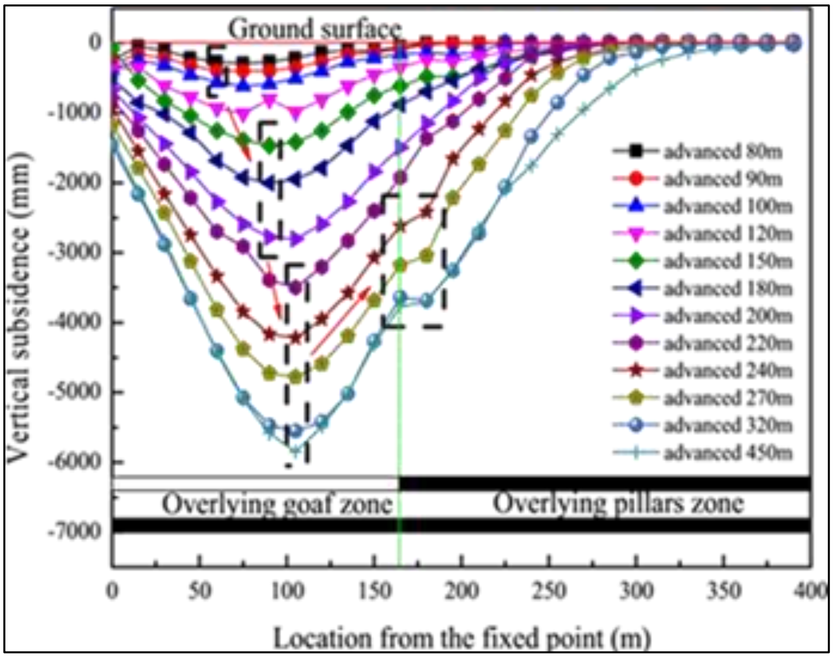


Figure 14. Bulianta LW22307 centre line vertical subsidence in mm. (Yang, 2019)

Figure 15. Bulianta LW22307 surface subsidence and cracking (Yang, 2019)

ROCKBURST & COALBURST (BURSTS)

Coalbursts involve the sudden, violent ejection of coal or rock into the mine workings. Coalbursts are a particular hazard because they typically occur without warning. Coalbursts are almost always accompanied by a loud noise, like an explosion, and ground vibration. The nature of coalbursts is quite variable. There are a number of forms of coalbursts that include:

- Events that may be partially initiated by gas operating as a component of effective stress in fractures but are principally strain energy events;
- Strain burst type events;
- Events that are associated with the release of seismic energy from the breakage of massive strata in the goaf;
- Events associated with sudden loss of strength on any plane – such as the roof and floor of the seam or of a specific joint.

Despite decades of research, the sources and mechanics of bursts are imperfectly understood, and the means to predict and control them remain elusive. High stress is a universal feature of burst-prone conditions. The overburden depth is responsible for the overall level of stress, but pillar design, multiple seam interactions, and/or mining activity can concentrate stresses in distinct locations. Geological factors also contribute. The presence of strong, massive sandstone near the seam has often been noted where bursts have occurred. In the Utah coalfields of the western United States of America, for example, miners refer to “bump sandwich” geology where the coal seam is slotted between massive sandstone roof and floor (Mark, 2016).

Significant rockburst and coalburst events have been recorded in longwall coal operations in massive strata in the United States of America. These include:

- Sunnyside, Book Cliffs / Wasatch Plateau Coalfield, Utah (Mark, 2016);
- Willow Creek, Book Cliffs / Wasatch Plateau Coalfield, Utah (Richardson 1996, Mark 2016);
- Lynch No.37, Harlan County, Kentucky (Hoelle, 2009);
- Elk Creek, North Fork Valley, Colorado (Mark, 2016).

Most American experience has been associated with deep longwall mining below 500 m of depth. However, this experience has resulted in the development of tailgate yield pillars which have the potential to significantly improve the conditions in shallow, thick seam longwalls in massive strata.

The previously described event that destroyed the longwall face at Churcha West in India was undoubtedly a coalburst.

Figure 16. Elk Creek longwall tailgate coalburst (Mark 2016)

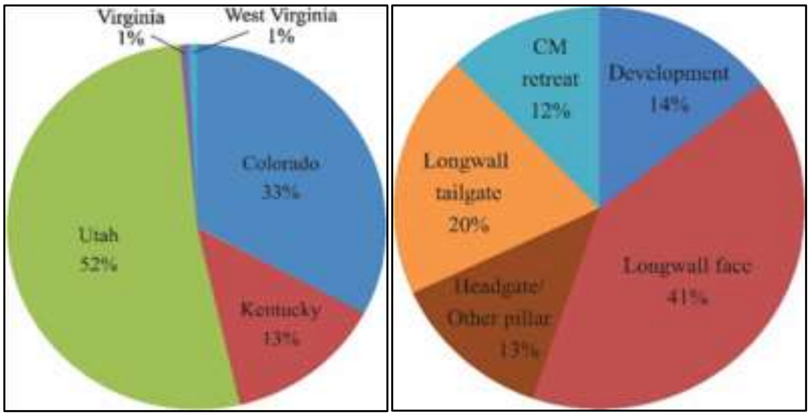


Figure 17. US 1994-2013 coalbursts (140 events) by location (Mark 2016)

FACE SPALL

Shangwan coal mine is located in the Shendong Coalfield, Inner Mongolia, China. Rib spalling in longwall operations is becoming an increasingly serious safety issue as thicker seams are mined in a single pass. Spalling also contributes to an increase in unsupported roof span at the face and a corresponding increase in roof falls (Zhang, 2016).

Rib spalling has been studied in LW12301 at Shangwan, where the seam was 6.2 m thick at an average depth of 240 m. Here the mining height was 6.0 m, the panel was 249 m wide and 4948 m long. Immediate roof was sandy mudstone 0.63 - 3.87 m thick. The main roof was sandstone 1.3 - 4.2 m thick and the immediate floor was mudstone 0.56 - 2.11 m thick and softened by water. Large areas of face spall exploded into the working area several times during periodic weightings, resulting in fish-scale-like face spall. The maximum depth of spall was 1.7 m.

A study showed that rib instability was generally located below the roof at 0.58 times the mining height. Using a "thin plate" mechanical model, a theoretical study showed that in LW12301, the maximum depth of rib spalling was 0. 98 - 1.61 m and the initial rib spalling started 2.53 m below the roof, which is basically consistent with field data.

Coal rib fracturing growth is influenced not only by the vertical stress, but also by the horizontal constraining effect of the intact coal seam beyond the fractured coal at the longwall face. The deformation of the fractured rib coal is inelastic and fractures resulting from compressive stress expand rapidly until rib failure occurs.

Table 4: Shandong mining height and depth of rib spalling on five faces (Zhang, 2016)

Longwall #	Mine	Mining height (m)	Form of rib spalling	Depth of rib spalling (m)
LW12301	Shangwan	3.3	Tension crack	0.83
LW12302	Shangwan	4.5	Tension crack	0.95
LW22307	Bulianta	6.8	Tension crack	1.45
LW52302	Duliuta	6.6	Tension crack	1.35
LW42104	Buertai	3.9	Tension crack	0.45

In general terms, depth of rib spall in the Shendong Coalfield is approximately 25% of the coal face mining height.

Figure 18. Shangwan rib spall on a longwall face (Zhang, 2016) and hard coal thick seam longwall face (Wang, 2018b)

SANDSTONE PALEOCHANNELS

Su (2001 and 2012) describe problems with mining at 180 m depth under a paleochannel of thick and massive sandstone at Enlow Fork mine in the Northern Appalachian Basin, Pennsylvania. This sandstone channel of 300 m width was known to cause serious longwall face roof control problems in the mine's B-panel area in 1998 and 1999. Some of the problems were large face cavities which caused production delay in the preceding panel. These problems were mitigated by the use of hydraulic fracturing. Early fracture operations improved face conditions but the pancake fractures were of inadequate size to cover the panel. Lu (2014) describes the beneficial effect of good hydraulic fracture design. In this operation fractures were developed of approximately 180 x 120 m horizontal dimension by pumping 16 m^3 of water and lubricant at 5.6 m^3/min. The purpose of the fracture was to relieve high shear stresses in the roof and was apparently successful.

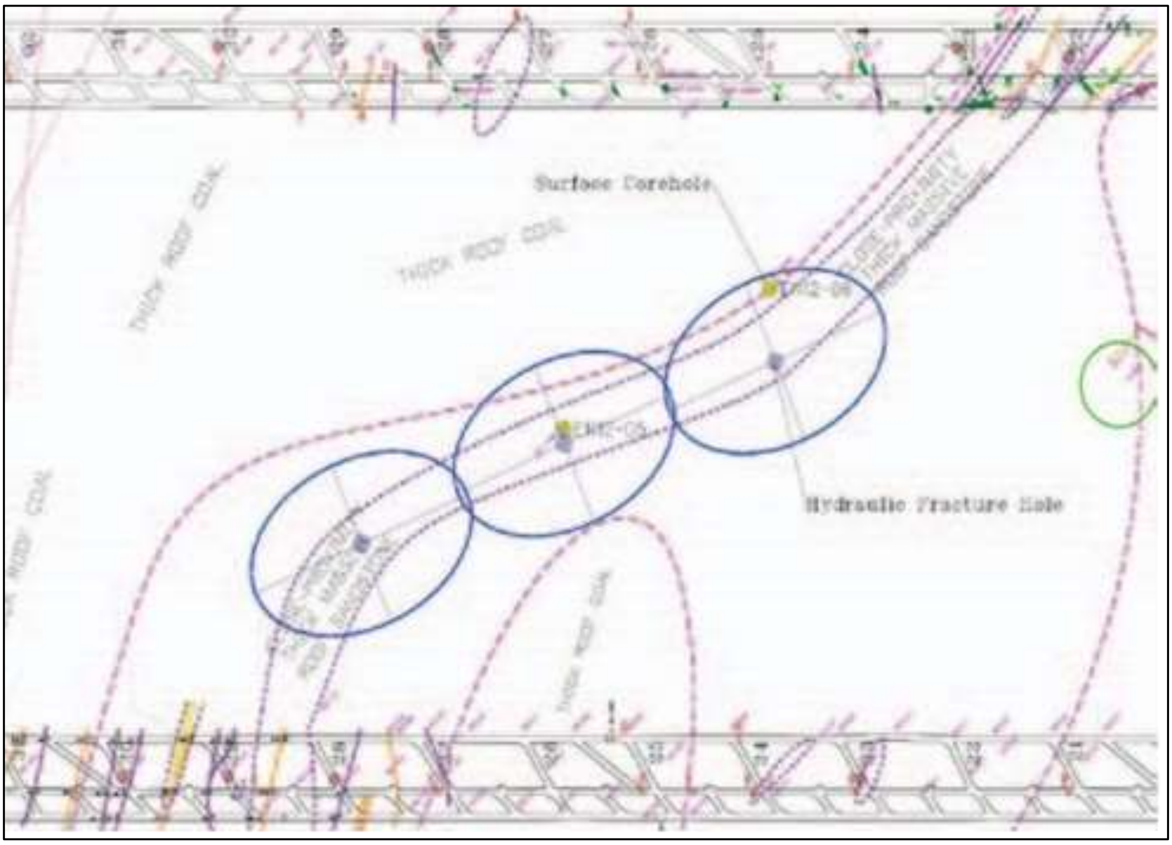


Figure 19. Enlow Fork sandstone paleochannel traversing longwall panels and hydrofractures (Lu et al, 2014)

Broadmeadow, in the Bowen Basin, Queensland, introduced a top coal caving (TCC) longwall face in 2013, but since then has experienced severe convergence events at the start of each panel after 60-70 m of retreat, resulting in equipment damage, and the longwall almost becoming iron-bound.

The longwall is Caterpillar-supplied with 158 x 2 m-wide powered roof supports. The run of the face powered roof supports are two-leg, 1460 t capacity, the gate end special powered roof supports are three-leg supports with a 1580 t capacity, the two powered roof supports covering the front, and rear maingate drives are four-leg with 1800 t capacity. The shearer typically extracts the basal 3.8 m of the seam, with the remainder recovered using the TCC method.

Figure 20 shows that periodic weighting has occurred throughout the mine life, due the presence of the moderately strong MP41 and MP42 sandstone units. The MP42 sandstone channel increased in thickness and was located over the start positions of LW7-11, with the thickest portion over LW10. A thick sandstone unit close to the extraction roof was likely the cause of the convergence events at the start of each longwall panel. (Coutts, 2018).

After a severe convergence event on LW10, the powered roof supports were modified to provide 450mm of additional convergence capability. The shearer height was lowered by 150 mm. These changes combined with a planned increase in extraction height to 4.1 m allowed a total of 1300 mm of convergence to be sustained without impacting operability. This was equivalent to 4 days of the worst case convergence experienced in LW10.

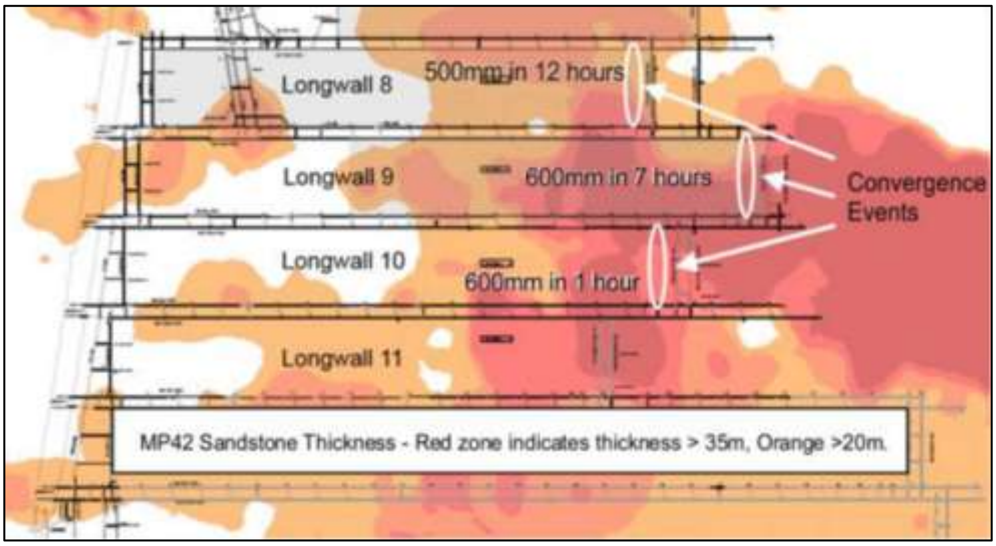


Figure 20. Broadmeadow MP42 sandstone thickness contours and convergence rates (Coutts 2018)

PRECONDITIONING

Preconditioning is a technique used to increase the fracturing of the whole or part of a mineral deposit so that it will cave or fragment more easily. Preconditioning has gradually been introduced into underground coal mining using techniques pioneered in the petroleum industry and underground hard rock mines. The latter have used it to assist block caving, sub-level caving, and panel-caving methods in rock types not traditionally suited to caving (Catalan 2012, Cuello 2018 and Florez-Gonzalez 2019). There are two preconditioning techniques which have met with success in longwall mining: explosives and hydraulic fracturing (often referred to as "hydrofracturing", "hydrofracking" or simply "fracking"). The Cadia East operation is shown in Figure 21.

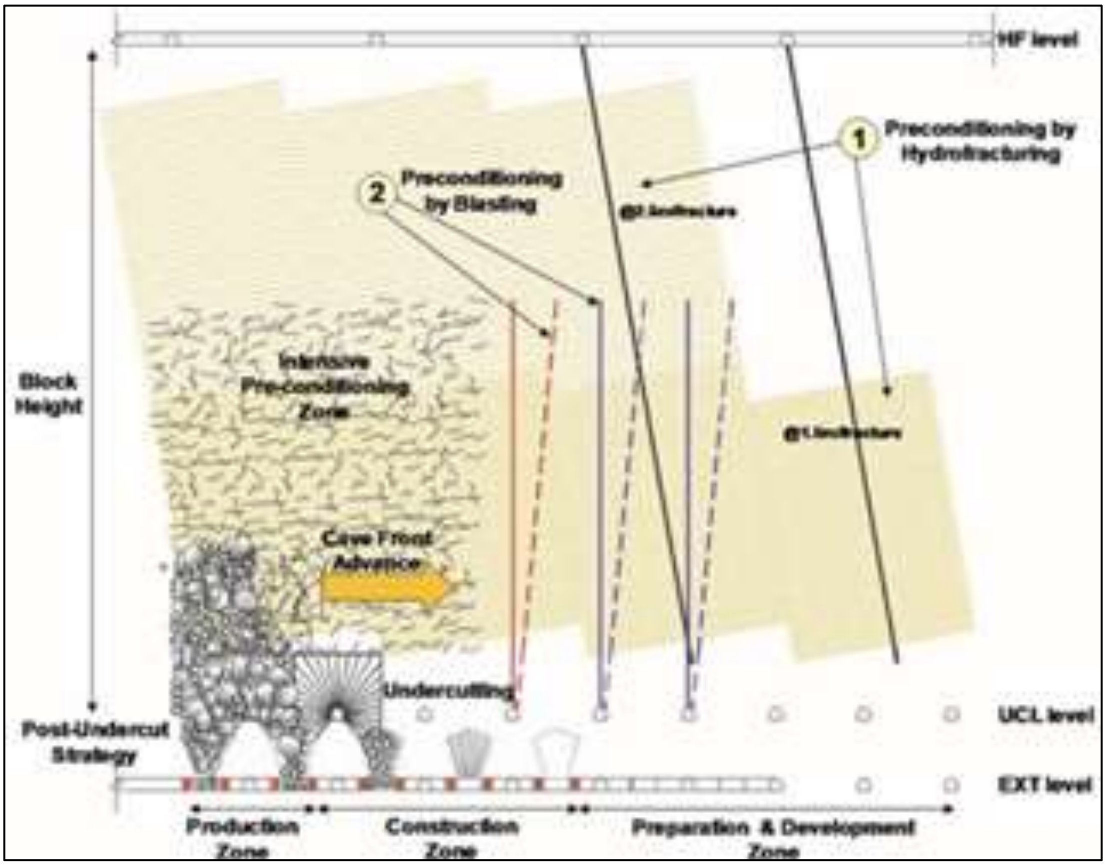


Figure 21. Cadia East intensive block cave preconditioning with explosives and hydrofracturing (Catalan, 2012)

Explosives

Currently, the major application of explosives preconditioning in underground coal longwall mining is in the Upper Silesian Coal Basin (USCB), shared by the Czech Republic and Poland. Mining in the USCB is at depths of 800-1200 m in massive sandstone and conglomerate strata. The rockburst risk in such conditions is extremely high. De-stress blasting in immediate roof rocks is considered to be the most effective preventative control. All blastholes are drilled upwards at angles of 13-25^{o} from the horizontal. Drilling is carried out in both the maingate and tailgate with holes varying in length from 47-90 m. Hole diameter is generally 95 mm at a spacing of 10 m. Holes are charged with plastic gelatine explosive (heat of explosion 4100 kJ/kg) with sand employed for the stemming (Konicek, 2018).

Because of the lack of a free face, the use of conventional explosives is limited in a confined environment to a fairly small radius of highly fractured material around the borehole. The use of high energy gas fracturing enables the extent of fracturing to be extended significantly. This process involves the use of substances which produce less detonation shock but rather sustain a gas pressure for a prolonged period to extend fractures.

Hydraulic Fracturing

Hydraulic fracturing was first used in Australian longwall coal mining at Moonee in the Newcastle Coalfield to help break up overlying conglomerate to mitigate severe wind-blasts. Holes were drilled into the roof from the longwall face. This technique, although slow and tedious, did succeed (Hayes, 2000). More recent Australian success has been achieved at Narrabri, Gunnedah Coalfield, using vertical surface holes in massive strata to initiate longwall caving (Jeffrey, 2013).

Enlow Fork in the Northern Appalachian Basin, Pennsylvania, a state-of-the-art longwall operation, first attempted hydraulic fracturing in 1998 using oilfield techniques to break up massive sandstone channels in the immediate vicinity of the longwall face, and has used variations of the methodology ever since. Enlow Fork leads the coal world in this application (Su 2001, Akinkugbe 2007, Su 2012 and Lu 2014).

The direction in which hydraulic fractures develop is a function of the stresses and pre-existing planes of weakness within the rock mass. In a uniform rock mass, a hydraulic fracture will develop in a plane that is perpendicular to the minor stress. When structure is taken into consideration, the fractures may develop along these. What happens depends on the minimum energy required to propagate the fracture. Where there are significant planes of weakness (structure) it is unlikely that pre-conditioning by hydrofracture will be required. The focus of hydrofracturing for pre-conditioning is on breaking up massive strata. The stresses that exist within such strata are therefore critical to the success of the operation.

If the minor stress is horizontal and aligned perpendicular to the longwall block, then vertical fractures will form which will be in the wrong direction to promote failure of the goaf. If the minor stress is perpendicular to the future face then multiple vertical holes would need to be drilled and hydrofractured to create face breaks. Alternatively, holes for hydrofracturing need to be drilled in line with the longwall block and stage fractured to create the multiple breaks required.

If the minimum stress is vertical then the fracture will become horizontal. This is the most desirable orientation for a fracture in preconditioning. The fractures formed can be large in area and form a series of separated plates which are more likely to fail in bending. Multiple levels of hydrofracture may be undertaken quite simply in a single vertical hole to break a massive unit into multiple plates. The process for this is to drill and case a hole and then to perforate it at the first level and to hydrofracture this. The hole is then filled with sand to cover the first perforations and then perforated again and hydrofractured. This process may be repeated to achieve the desired degree of fracturing.

If the stress field is isotropic and the rock homogeneous then the fractures may form in random directions.

The limitations on hydrofracture are the area of the fracture that can be achieved and the orientation of the fracture. The latter is governed by the stresses and any structures that may exist. The size of fracture is basically limited by the ability of the rock to absorb the hydrofracture fluid versus the flow rate and pressure that can be supplied by the hydrofracture pump. Thus porous, permeable strata require hydrofracture fluids that will seal the hydrofracture surfaces with a filter cake and be more viscous as this will also limit fluid penetration. As with any complex process, there is a trade of which parameters will give the maximum benefit, in this case the largest area hydrofracture. This is the art of the hydrofracture design, which has been the subject of intense study within the petroleum industry for the purpose of flow stimulation. The difference between petroleum applications and those used for preconditioning are that in the latter there is no need to maintain fluid conductivity within the fracture. This simplifies the fracture process significantly.

GOAF BEHAVIOUR

We need some model to predict how the goaf will behave behind the longwall. The question is how will the rock break up? This is dependent on the stresses within the goaf and the strength of the material within it. Prior to the goaf forming, the rocks will carry both vertical and horizontal stress. As the coal is removed by the longwall, the vertical stress holding the roof up is removed. There are then several possibilities as to how failure may occur. These include tensile and shear failure.

Tensile failure may occur perpendicular to the free face of the roof of the goaf. It is induced by gravity alone and leads to slabbing. There may also be tensile stress associated with local stress concentrations near the installation road.

At the start of the goaf, there may be a compressively induced shear brought about by the high lateral stress in the roof rock combined with a lack of confinement. Compressive initial failures can be extremely violent when the roof is strong. They can have all the characteristics of a uniaxial specimen failing violently in a universal test machine. In the absence of high compressive strength, shear is a potential failure mechanism brought about by gravitational load acting at each end of the bridged roof. In a low lateral stress environment, the goaf roof rock may simply drop out with steep sides. This form of collapse gets wider and higher as the longwall progresses. The width and height of such a failure may be the piston that drives a wind-blast event.

Once the goaf has formed, there is no compressive stress at the goaf edge near the roof nor further up into the goaf until the rocks begin to form a compressive arch or bridge. Compressive stress related shear failure may take place within this arch. Near the goaf edge the lateral stress has been relieved and the stresses may be stylised to have the form shown in Figure 22.

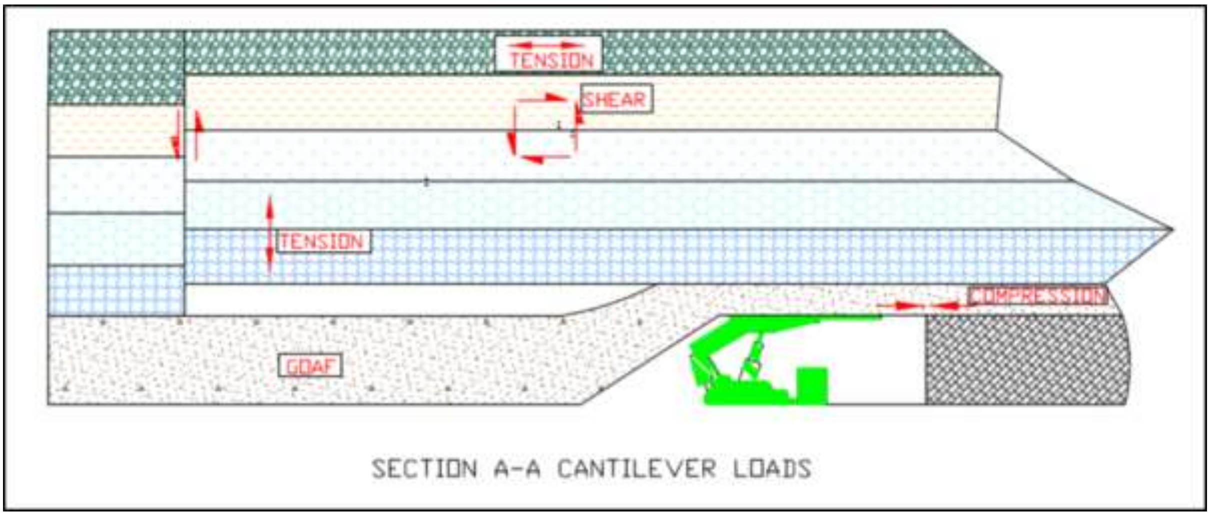


Figure 22. Stylised view through longwall face and goaf showing stresses

Figure 22 shows a section through a longwall face and goaf. It shows an immediately collapsed weak roof behind the powered hydraulic support. Whether this has fallen depends on the rock type. The types of stresses in roof above it are shown. Tension may be developed within the hanging mass. This may cause the rock to pull apart. This will normally occur on weak bedding planes.

It is quite possible that these bedding planes may have already been separated by shear developed by the release of lateral strain. If the rock laminations contain varying strain levels, shear may develop between them even while they are ahead of the face, though the lack of confinement after the passage of the face would make this more likely. Alternatively, the rock on the goaf edge may shear from the material above because the rock above it is constrained within an arch or a not properly disrupted goaf. Shear may also occur by the self-weight of a cantilevered section of rock. The shear stress within this is caused by the overhanging mass. This vertical shear has a conjugate shear that will act along the bedding plane. Uniform section elastic beam theory suggests that the shear stress has a parabolic profile with a maximum which is 1.5 times the average shear at the mid height of the beam.

The layers of strata act as plates which may be joined by cohesion and friction but are likely to separate by tensile or shear stress on weak bedding planes. Once separated, they behave as individual units that may be loaded from above. The likely failure mode is then one of tension induced by bending. If the layer of strata is jointed through without infill, it will not have tensile strength. However there are many layers of strata in coal mines which are not jointed. Some of these have very significant tensile strength.

A massive, thick, stratigraphic unit can support its own weight for a substantial distance into the goaf. Simple elastic linear beam theory can be used to describe the distance before it breaks in tension or in shear. The formulae for these are given in Equations 1 and 2 respectively. Equation 1 defines the length of the cantilever subject to a tensile stress failure at its top. Equation 2 defines the maximum cantilever length due to shear stress at the mid-depth of the cantilever. Each case assumes that there is no loading on the end of the cantilever from the goaf.

$$l_b = \sqrt{\sigma_t d/(3\rho g)} \tag{1}$$

$$l_\tau = \frac{2\tau_f}{3\rho g} \tag{2}$$

Where
l_b is the length of the cantilever unit at failure due to tensile stress
l_τ is the length of the cantilever unit at failure due to shear stress
d is the cantilevering units' thickness
ρ is the density of the rock
τ_f is the shear strength of the rock on a bedding plane
σ_t is the horizontal tensile strength of the rock at the top of the unit.

The question is, what defines the beam or plate thickness? Is it defined by shear along bedding planes caused by pre-existing differential strain between them or is it due to tensile failure across the bedding plane due to gravitational load?

The equations used are based on linear elastic theory though it should be realised that the rock is probably anything but linear in its behaviour (Gray, Zhao and Liu, 2018). In addition to nonlinearity and anisotropy of elastic parameters, the failure behaviour of many of the coal measure rocks are also anisotropic.

In a recent project undertaken by Sigra, the rock was tested uniaxially, hydrostatically and triaxially for elastic properties. It was also tested for strength uniaxially, in shear triaxially, and also in shear by direct shear on the bedding planes. In addition it was subject to true tensile testing both perpendicular to the bedding planes and in the direction of these. A laminated medium-grained sandstone unit with micaceous bedding planes has the typical properties shown in Table 5.

Table 5. Rock properties for a medium-grained laminated sandstone with mica on the bedding planes

Rock property	Units	Value
Uniaxial compressive strength	MPa	62
Mohr Coulomb angle of friction (axial triaxial test)	Degrees	42
Mohr Coulomb cohesion (axial triaxial test)	MPa	13.6
Cohesion (transverse shear)	MPa	2.0
Tensile strength across bedding (axial pull)	MPa	0.5
Tensile strength along bedding (transverse pull)	MPa	5.0
Secant Young's modulus (uniaxial at 10 MPa)	MPa	6517
Ratio of anisotropy (horizontal vs axial) at 10 MPa, (hydrostatic test)		3.6
Non linearity of axial modulus at 10 MPa (hydrostatic test)	MPa/MPa	395
Permanent axial offset at 10 MPa (cyclic axial compression)	με	464
Permanent circumferential offset at 10 MPa (cyclic axial compression)	με	-38

Table 5 shows that the rock is very anisotropic and nonlinear in elastic properties. The cohesion measured from triaxial testing is seven times higher than that measured in direct shear on the bedding plane. The tensile strength measured in the direction of bedding is ten times that measured across the bedding. The rock is also inelastic, showing significant permanent strain at quite moderate axial loading. These factors need to be measured and taken into account when determining rock mechanics behaviour and certainly in the failure mechanism of a goaf. In addition the pre-existing stresses and their associated strains are important. Along with material properties they determine what layer of strata will shear over which, thus determining the thickness of units that are then liable to bending failure.

The actual shape of goaf will be more complex than described by the simple beam model presented above because it has a three dimensional component around the face ends where the effects of pillars will occur.

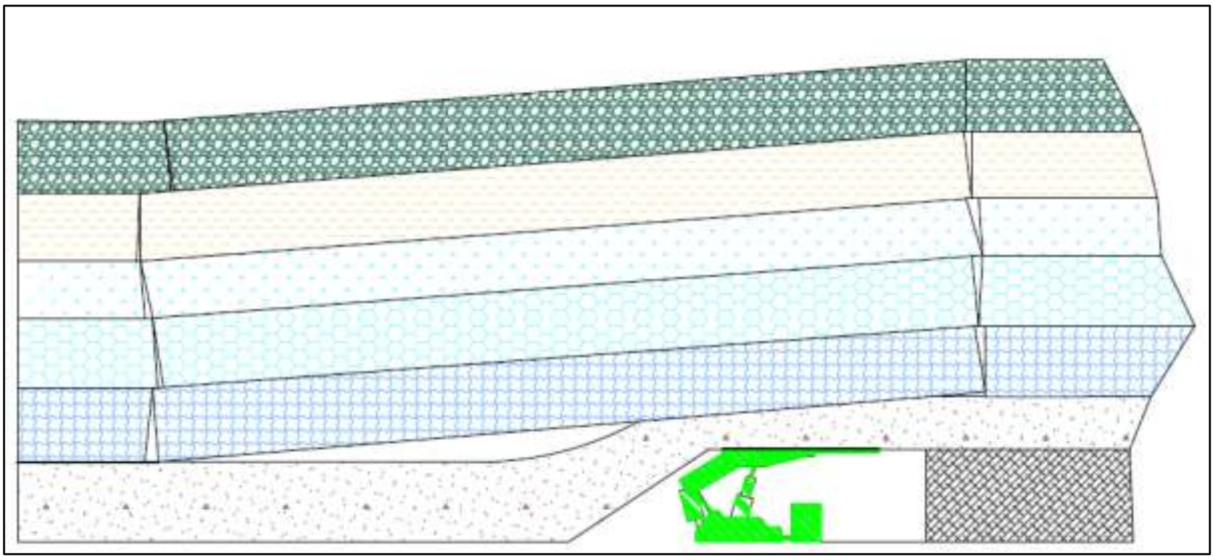


Figure 23. Block failure behind the face

Figure 23 shows the type of failure associated with massive strata. In this, the immediate weak roof has fallen but above it is a number of major slabs that have delaminated on weaker planes. These have fallen sequentially from the bottom up. They are shown here breaking just over solid coal in the face.

As such, they may by their geometry behave as a lever with a fulcrum just beyond the face, which places enormous vertical load on the coal at the face. This would cause face spall. The slabs are shown here falling sequentially with connection to similar rock behind in the goaf. Under such circumstances some level of horizontal stress may be maintained along the axis of the longwall (Gray, Wood and Shelukina, 2013) helping to provide a controlled fall as there is friction at the break ahead of the face. If the slabs do not fail evenly and sequentially the slabs can slide backwards into the goaf. This causes open voids above the face, which is associated with rock blocks falling from it on to the armoured face conveyor. Recovering from such an event becomes a major exercise.

If the blocks that form are of sufficient size they can cause major problems for the powered roof supports which have to bear the weight of the block as it breaks from the cantilever, causing a weighting problem.

The tensile failure of massive strata is a mechanism for large seismic energy release, as very little energy is used in creating the failure compared to the shear failure mechanism. This raises the likelihood of rockbursts triggered by seismic events.

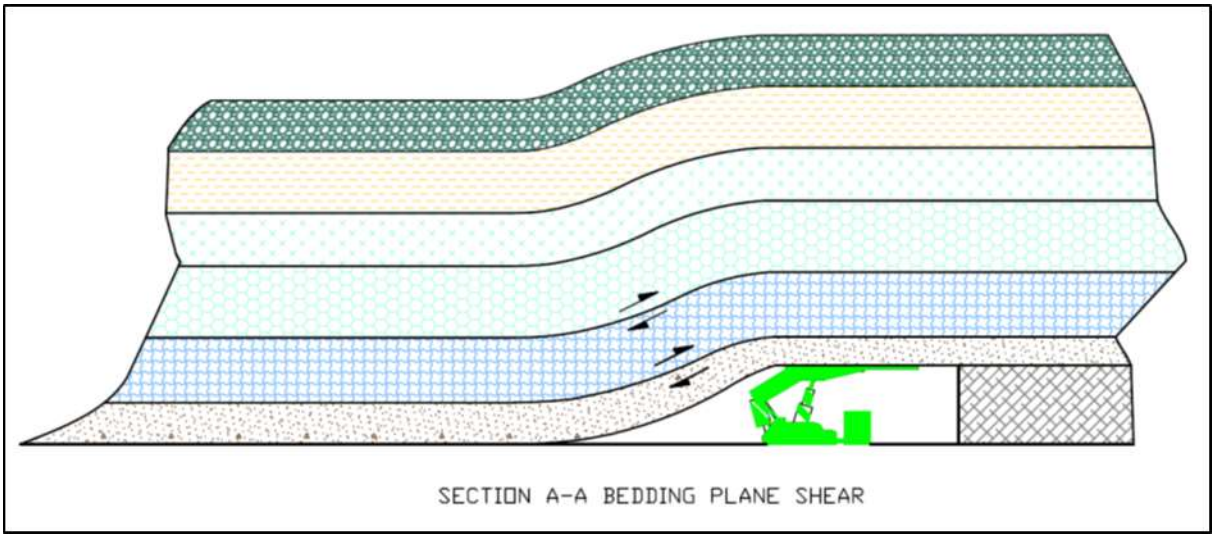


Figure 24. Goaf formation in a weak rock

By contrast to the massive rock case, Figure 24 shows the even formation of a goaf in weaker rock and higher stresses. Here shearing readily occurs on weak laminations.

The problems at the face are mirrored over the tailgate. Large blocks can move and cause problems such as shear of the tailgate pillars. The formation of large blocks carrying load from the adjacent goaf may load the tailgate pillar unduly. It is Chinese practice to use pre-splitting of a massive roof to prevent this problem (Chen, 2018). This pre-splitting is normally conducted from underground.

CONCLUSIONS

This paper describes the difficulties associated with longwall coal mining under massive strata that leads to the formation of large blocks. These problems include weighting events, the formation of cavities over the face, face spall, coalbursts, pillar failures, uneven subsidence and wind-blasts. These problems are a horror story for miners.

It is essential before mining commences to determine what the goaf behaviour will be. Will there be problems with massive units or not? This requires detailed measurement of stresses and rock properties well beyond the current practice of simply sending rock samples for uniaxial or triaxial testing to obtain failure properties that do not relate to the failure planes that will form to delineate block boundaries. Armed with this knowledge, it is then possible to start to build a realistic model that will focus on whether large blocks will form with mining or not, and what their dimension may be. This will be the key to successful powered support design.

Where the formed layer thicknesses and hence block sizes are too great, it is then necessary to consider preconditioning to ensure that the sizes are manageable. Preconditioning can be achieved through the use of hydrofracture or by blasting. The successful use of hydrofracture is dependent upon the stress regime as this will determine the fracture orientation. Where this is unfavourable, a resort can be made to blasting. This has a far more limited range than hydrofracture. It can however be increased by the use of different explosives that primarily generate gas rather than a detonation.

In conclusion, mining in massive strata is not easy and should only be undertaken with due consideration.

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