LONGWALL MINING

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INTRODUCTION

Since the 1993 edition of Monograph 12 was published, mechanised longwall mining has continued to evolve. Positive changes have been achieved in many areas including health and safety, production levels, productivity, unit costs and total operating costs. These improvements have not always been successful at every mine that has utilised longwall mining techniques, in Australia. Many improvements have, at times, been hard won through dedicated application and perseverance by miners, tradespeople, supervisors, production and maintenance staff, mine managers and equipment suppliers. A wide variety of geo-technical consultants and more recently researchers from organisations like the CSIRO and CRC Mining have assisted. Apart from much needed improvements in health and safety, profitability and return on capital have also improved. Higher production rates and productivity levels have been achieved due to the successful implementation of changes associated with new and sometimes innovative techniques and equipment.

Shortwall mining using continuous miners, discussed by Hedley (1993) has now ceased to be utilised in the coal industry in Australia. Lower than acceptable production levels and high production costs make this mining method unprofitable. Risk levels surrounding health and safety to face operators – mainly associated with geo-technical issues like soft and low strength ground – large prop free front and high canopy aspect ratios that contributed to poor levels of production and productivity and low levels of confidence from management were the main reasons for its demise.

Longwall mining will remain the principal extraction method for underground mines in Australia in the foreseeable future. Thick seam reserves with high quality coal at shallow depth and benign geological conditions will be preferentially targeted to keep costs low and output high, thereby providing optimised benefits to shareholders.

Mechanised longwall mining is ever changing and evolving with new techniques, technology, equipment, face management practices and systems appearing as a direct means to continually improve all aspects of operational and financial performance. Continued focus on these critical aspects, implementation of process improvements and ongoing development of automation will see critical performance indicators relating to longwall mining further improve. This will make the Australian underground industry increasingly viable, in the face of the long term trend of lower real term commodity prices.

Longwall retreat mining has traditionally produced coal using two methods, bi-directional and uni-directional mining cycles. The use of bi-directional cutting has always been seen as more productive on longer faces. However, as shearer power and haulage speeds have increased, thicker seams have been targeted, and environmental considerations increase, uni-directional cutting has become more competitive and can be more productive than bi-directional cutting.

In the design, specification and manufacture of a set of longwall equipment, the concept that the equipment must be designed to function as an integrated whole should be an initial and continuing goal. It is not satisfactory to select items from the various categories of equipment and then assemble them together in the hope of compatibility.

The ongoing application of new technology, coupled with enhanced operating methods, will ensure that Australian operations remain world class.

ROOF SUPPORTS

In preparing a technical specification and scope of work for the roof supports as part of introducing a longwall mining system or purchasing a replacement, a thorough design review of all documentation should be undertaken both prior to and post the tendering phase. While this review process sounds quite a logical and sensible approach to completing a thorough analysis, perhaps ideally including an independent consultant’s technical review, it takes a considerable amount of time. Quite often, an inadequate amount of time is allocated in the project schedule to completing this essential work. Consequently the decision to select is rushed. The risks and implications of failing to employ a quality management approach, or a simple technique involving a ‘plan, do, check and act’ process is not fully understood. The selection team needs to understand their rationale for the roof support design and look at how the technical choices that are made will impact on the capital and operating cost outcomes in incremental terms over the total life cycle of the roof support’s operation. Historically, too much attention is paid to the upfront capital cost. Inadequate consideration is given to learning which operational or production benefits can contribute to lower operating costs and higher profits. Careful choice of a better designed roof support system will improve these figures.

Roof support selection should take into consideration:

1. support resistance,
2. roof and floor pressure distribution,
3. cutting height range,
4. collapsed and transport height,
5. travelling access through the supports,
6. the setting and yield pressure,
7. hydraulic circuit,
8. support control system, and
9. speed of operation.

Longwall mining is used in the relatively moderate depths of 120 to 300 m in New South Wales (NSW) and Queensland and up to 650 m in the Illawarra Coalfields. The depth of cover has an impact on the desired roof support capacity and density. The presence of any coal beams of more than 0.5 m and up to 3.5 m in any panel can also contribute towards stable roof conditions. However, if the seam has a cleated nature, then the roof coal and any apparent weakness of the strata overlying the immediate coal roof demand careful consideration in the design of the roof supports. In all cases, a geotechnical evaluation of the seam and mining conditions at the mine and other mines with a similar seam and geology, in close proximity to the seam to be mined, should be made in terms of matching the roof support design to the mining conditions. To evaluate how to improve the capabilities of a two-leg shield support, the following aspects are recommended for consideration.

Support load and resistance

The relationship between the support load generated by a shield support and the stiffness of the support in terms of roof convergence prior to generating the yield load is critical in controlling the roof above the longwall. The support load expressed as the support density measured in tonnes per m² needs to be determined for each longwall mine's particular geo-technical requirements.

The ability of the shield to control the roof strata above the longwall can be expressed in a 'ground reaction curve' which mirrors the load, displacement characteristic curve for a given rock type. Before each new longwall installation, work should be completed to develop a specific ground reaction curve for that mine. The principle of its significance as developed by Wilkinson (2004) for use at BMA Broadmeadow Mine in Queensland is illustrated in Figure 1.

Using Figure 1, for Broadmeadow Mine, as an example it can be seen that prior to setting the roof support, an amount of closure due to the geological characteristics of the roof and coal seam will occur. No roof support system is able to resist this behaviour. The time prior to setting the support will also influence the amount of roof closure. Once the support is set to the roof and a realistic setting pressure is achieved, closure of the shield support

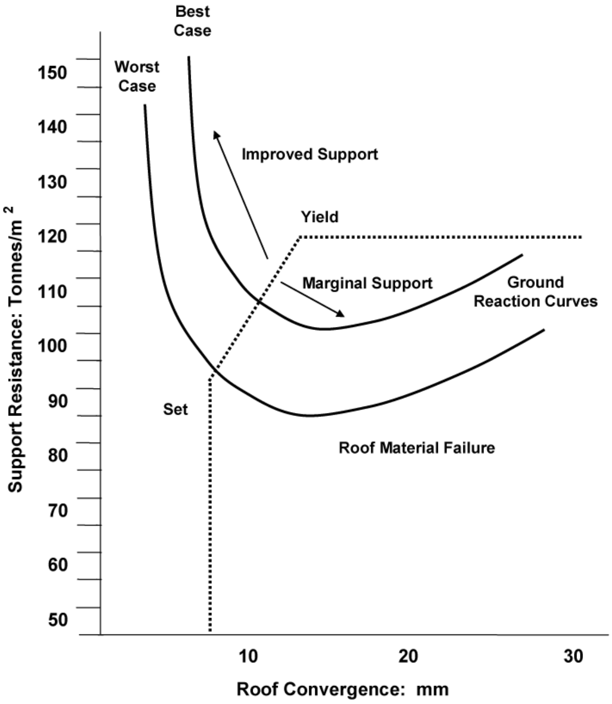

FIG 1 - Ground reaction curve – BMA Broadmeadow mine

will still occur as the roof converges, until yield pressure is reached. The closure between set and yield can be calculated for a given leg configuration and is determined by compressibility of the fluid in the leg and dilation of the leg. Once the yield pressure is reached the leg will yield as designed. Consequently, the pressure in the leg will reduce to approximately 96 per cent (+/-1.5 per cent) of the yield pressure. Pressure will again increase in the leg as convergence of the roof continues until the yield pressure is reached, and the cycle is repeated.

Wilkinson (2004) has described that the gradient of the ground reaction curve continually changes up to the point where the strata fails. Referring to Figure 1 it can be seen that at the lower apex of the curve the roof is no longer acting as a competent beam but has experienced fracturing and is not intact. The purpose of hydraulic shield supports is to prevent or delay the failure of the roof during the mining operation. In Figure 1, higher up the left-hand side of the ground reaction curve where the support characteristic intersects the curve, the roof control will be better. Conversely, the farther towards the lower apex on the right-hand side of the ground reaction curve, where the roof support characteristic intersects, the roof control will be worse.

Support stiffness is controlled by the closure of the leg between set and yield and the subsequent closure characteristic once yield is reached. A shield support with lower set to yield ratio will allow more roof closure before yield load is achieved.

Set to yield ratio is expressed as: $\dfrac{\text{Set pressure}}{\text{Yield pressure}}$

The total amount of roof closure prior to strata failure varies depending on the strata characteristics such as coal stiffness and vertical and horizontal stress. It is hypothesised by geo-technical consultants to be between 15 and 35 mm of sag.

Generally, for specific ground conditions, the desired effects of support stiffness are:

1. to control the roof as quickly as possible at a point on the ground reaction curve where the roof is in a stable condition, and
2. to create the maximum buffer in terms of roof closure and time between the 'set' condition and the immediate roof becoming unstable.

As the roof deteriorates and ground convergence occurs, the rate of convergence for a constant load will increase. The advantage with a stiffer shield support is that the rate of roof convergence is lower when the support reaches yield load than is the case for a softer shield support.

The support characteristic of a longwall shield support can be determined by:

1. the load or support density applied to the roof at both set and yield pressures expressed in t/m²; and
2. the convergence rate of the roof, relative to the floor at set and yield pressures.

For example, for Broadmeadow Mine, the approximate support characteristic for a shield design involving both 380 and 400 mm bore legs is shown in Figure 2.

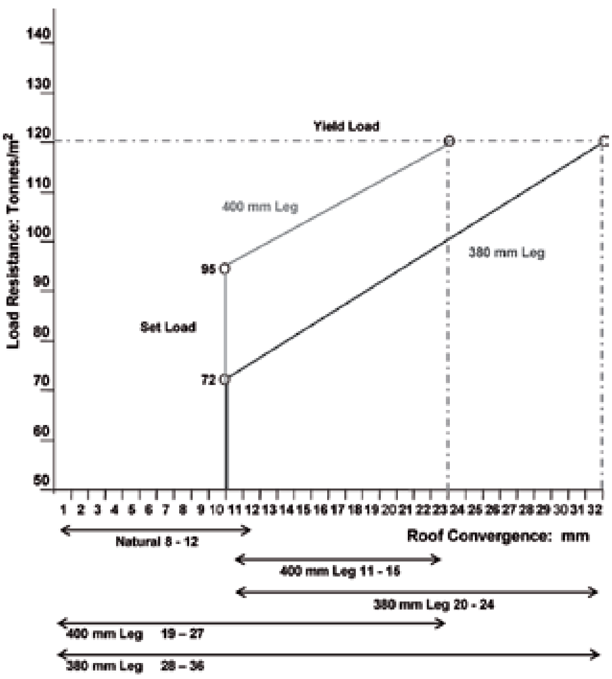

FIG 2 - Support load and roof convergence characteristics – BMA Broadmeadow Mine

Figure 2 illustrates that although the support density at yield load is the same at approximately 122 t/m², the initial setting load is larger for the 400 mm leg at 97 t/m² compared to 87 t/m² for the 380 mm leg. The roof closure that occurs between set and yield for the 400 mm diameter leg is 11.29 mm compared to 20 mm for the 380 mm diameter leg for a set pressure of 345 bar and 14.8 mm for the 400 mm leg, and 23.7 mm for the 380 mm leg at 325 bar setting pressure.

Roof support stiffness can be expressed in terms of roof support closure from set to yield pressure. Using the convergence numbers shown in the x-axis in Figure 2 it can be calculated, using the formula 1- (11.29 mm/20 mm x 100) that the hydraulic stiffness of the shield support with the 400 mm leg is approximately 44 per cent greater than the shield support with the 380 mm leg. It should be noted that a roof support with a shorter stroke will influence the stiffness and rigidity of the leg.

Figure 2 illustrates two components of roof movement prior to achieving yield load namely natural convergence and hydraulic support convergence:

1. Natural convergence is a function of the strata, the operating cycle and the characteristics of the immediate roof and floor material. Estimated ranges of convergence values have been assumed as follows:
   a) Natural strata movement caused by the relaxation of strata, after the web is cut, which cannot be prevented irrespective of the capacity of roof support. This movement is dependent on roof strata behaviour and support resistance of the coal seam and is believed to range between 2 to 6 mm.
   b) Irrespective of the capacity of the roof support, broken roof material on top of the canopy and loose or soft floor material under the shield support base pontoons have some natural compaction before yield load is achieved. This varies depending on roof and floor type, face management, equipment design and strata material characteristics. For material above the canopy the value is a conservative 2 to 3 mm and for the material under the base 2 to 3 mm.

      Although no physical measurements have been taken, there is a need to use an assumption involving a range of 6 to 12 mm. It is reasonable to take the mid point of 9 mm. The sensitivity of this parameter would obviously affect the permissible roof support closure prior to roof failure.
2. Hydraulic support convergence can be defined in two ways:
   a) Convergence of the support between being set to the roof and when it achieves full yield load due to convergence of the roof increasing the pressure in the leg.

b) Convergence of the support at yield pressure which is a function of the differential between yield pressure and the re-set pressure. The re-set pressure, after fluid has been exhausted to atmosphere through the yield valve, should be as close to yield pressure as practically achievable. The rate of pressure buildup from re-set to yield pressure is a function of the rate of roof convergence.

The calculated support convergence between set and yield for the 380 mm and 400 mm diameter legs is shown in Table 1. If the nominal pump pressure is 345 bar, the worst pressure on the face is likely to be approximately 325 bar. If a high pressure set system is used, the setting pressure could be up to 400 bar. However, as 400 bar is too close to the yield pressure, there is a need to allow for hysterisis effects between yield and re-set pressure. In this circumstance, the high pressure set for the 400 mm diameter leg would have to be lowered to 380 bar.

Therefore, the total convergence or the possible total movement of the shield support with the 380 mm leg is very close to expected roof closure at the point on the ground reaction curve where the roof material becomes unstable and rapid convergence or roof falls may occur.

Using the above logic, a larger diameter leg, say the 400 mm diameter leg, used in combination with a versatile and flexible high pressure set system would provide superior roof control compared to the 380 mm diameter leg. The downside of using a larger diameter leg for improved technical and operational outcomes is the additional capital required to purchase these larger units.

Support stiffness

Most of the shield supports are supplied by JOY and DBT and are designed to use 380 mm internal diameter legs combined with a high pressure set system. Although theoretically, this combination provides reasonable stiffness, the operational risk of the high pressure set system not being available, or being switched off for operational reasons, is high. If the system is turned off for any reason; for instance, to better negotiate deteriorated roof, there is a risk that it may not be turned on again soon enough. Such an event could result in roof falls, in front of the roof supports, that would slow down the mining process and may result in the need for expensive resin injection practices to be employed. Currently, in the Bowen Basin, perhaps due to poor operating and/or management practices, the operating experience of using these high pressure set systems has not proved a dependable solution to roof control. High support stiffness can be achieved more reliably and consistently by using a larger 400 mm diameter leg. These provide similar stiffness to a 380 mm diameter leg without application of the high pressure set system and have considerably better (approximately 29 per cent higher) stiffness than the 380 mm leg with high pressure set. As mentioned above, due to hysterisis effects between the yield pressure and the re-set pressure, the high pressure set for a 400 mm diameter leg would be best set at 380 bar as compared to 400 bar for the 380 mm leg.

The stiffness of a support and its resultant effect on roof control is critical to the operation. The 400 mm diameter legs assist in mitigating the risks of poor ground conditions and therefore are a substantial benefit to the mining operations. Positive roof control has proven to be one of the most crucial operational factors in achieving consistent longwall production. This is certainly true of longwall mines in the Bowen Basin. A stiffer support system is a big advantage and contributes significantly to increasing the potential success of longwall mining.

There is an increasing emphasis by geo-technical consultants to recommend the internal diameter of the legs be increased from 380 to 400 mm and also to incorporate all appropriate changes in leg design to further enhance these stronger legs with a larger bore. Such changes significantly increase the stiffness of the roof support by approximately 44 per cent and reduce the amount of roof convergence that can occur between setting the shield support at the standard 325 bar and reaching yield pressure. The effect of such changes is shown in Table 2.

Each mine should determine from an internal geotechnical engineer or an external consultant the allowable amount of maximum roof convergence prior to roof failure. One consultant's opinion is that the roof convergence of the 380 mm leg operating in the Goonyella Middle Seam is beyond or close to the probable practical limit of roof closure and therefore a 400 mm leg is recommended.

Another potential benefit of a 400 mm diameter leg is it can increase the yield load capacity of the support but at the sacrifice of the roof support's fatigue life. This could be advantageous if unexpected cyclical loading strata behaviour is encountered. This advantage is countered by the fact that a roof support is designed to have a 'structured

TABLE 1 - Convergence between set and yield

Leg diameter (mm)	High pressure set pressure (bar)	Leg convergence between set @ 325 bar and yield pressure (mm)	Leg convergence between high pressure set and yield pressure (mm)
380	400	23.7	9.9
400	380	14.8	7.0

TABLE 2 - Comparison of 380 mm and 400 mm legs

Parameter	Units	380 mm leg	400 mm leg
Set pressure	Bar	325	325
Yield pressure	Bar	410	410
Support density at set load (before the cut)	t/m²	97	97
Support density at yield load (before the cut)	t/m²	122	122
Roof closure (set to yield)	mm	23.7	14.8
Natural roof convergence	mm	+/- 9	+/- 9
Total roof convergence:	mm	32.7	23.8

life' of say, 50 000 cycles for a load capacity of 1100 t that occurs when the yield pressure reaches 400 bar. If the yield pressure is increased by 12 per cent to 450 bar, the capacity of the support will be increased by 12 per cent to 1232 t. However, as the structure of the support was designed for 1100 t operating at 1232 t means its 'fatigue life' is reduced by 12 per cent to 44 000 cycles.

High pressure set

A normally accepted concept is to increase support stiffness by installing a high pressure set system on the face. Theoretically, it allows the setting pressure on 380 mm legs to be increased from 325 to 400 bar. This system operates across the entire face. However, if roof conditions above one or more supports require the high pressure set to be switched off in order to advance the face, then the complete face would be without high pressure set.

Although support yield loads are critical design parameters for any longwall shield support system, the manner and consistency with which the load is applied to the roof and the behaviour of the shield support are equally important. The following characteristics of the shield support relative to the probable roof and seam conditions are considered significant:

1. Stiffness of the shield support in terms of the quantity of roof convergence prior to achieving yield load.
2. Load distribution between the canopy and roof to prevent crushing of the roof adjacent to the goaf edge.
3. Optimising canopy tip load. Generally, a higher tip load would be preferable for 'softer' roof conditions. Canopy tip contact and effective tip to face distance need to be controlled and unsupported roof between the canopy tip and the coal face minimised. Although design of longwall shield supports caters for a flat horizontal roof with a theoretical tip to face distance of approximately 500 mm, the effective tip to face distance is dependent on canopy orientation relative to the roof. The nominal roof profile is dependent on seam undulations, steps cut by operators, and material that may fall from the roof. These issues are largely related to mining conditions and operator proficiency and are beyond the control of the shield support designer. Experience shows however, that many modern, high-capacity, two-leg shield supports have a tendency to crush weak or fractured roof adjacent to the goaf edge due to the high load concentration between the top dead centre of the leg and the rear contact point of the roof at the goaf end of the canopy. This results in the canopy having a tendency to punch into the roof or lift at the rear end. Consequently, contact between the face end tip of the canopy can be affected. In weaker roof strata it is always preferable to maintain canopy tip contact and maximise tip load.
4. Canopy aspect ratio and load distribution on the canopy play a significant role in controlling canopy orientation.
5. Hydraulic integrity of the shield support over its expected service life is critical. The reliability and maintainability of the support has a direct affect on the percentage of available hydraulic load capacity that can be applied to the roof. For example, if 20 per cent of the leg and/or valves on the face are leaking or malfunctioning, the effective support load of 1050 t per shield support system is only realistically an average of 840 t.

Guaranteed high pressure set systems assist in ensuring that the shield support is set at the required pressure to place load onto the roof and achieve appropriate stiffness. These systems are often most valuable when the roof conditions are poor due to weak roof, geological disturbances and/or where high stress conditions exist. Under these conditions operators have a tendency to turn these systems off to prevent individual supports punching into the roof or roof cavities. Once the systems are turned off, they are often not switched back on when it is appropriate to do so. Therefore, it is critical that guaranteed and high pressure set systems are designed to be as flexible as possible. This would ideally enable shield operators to easily and rapidly disengage and re-engage the high pressure set on each individual support as operating conditions dictate.

One sensible option proposes that solenoid valves be fitted to each shield to allow infinitely flexible operation of the high pressure set system. This solution makes the high pressure set system operationally flexible and practically manageable and this would allow individual supports to be isolated and reconnected to the high pressure set system as required. For example, DBT have a high pressure guaranteed set system that complies with this principle by incorporating individual solenoid control of this function on each support. Another initiative involves each leg in the support being able to be mechanically isolated so the high pressure set can still be used if one leg is leaking or bypassing.

Addressing these issues proactively recognises that support stiffness is a critical issue. Use of 400 mm diameter legs and the solenoid-operated high pressure set system should improve operational flexibility when ground conditions deteriorate. This action comes at an incremental cost but it provides another solution that can be quickly initiated in an unforeseen event. If the high pressure set system does break down, the 400 mm diameter leg provides greater and more adequate stiffness than a 380 mm leg. The combination of the two will provide an extremely stiff support that should adequately control the seam roof.

Canopy control and canopy aspect ratio

The canopy orientation and stability of a two-leg shield is normally evaluated in terms of the ratio between the forward canopy projection (canopy tip contact to top dead centre of leg) and the rear canopy projection (top dead centre of leg to rear contact point of canopy) referred to as the 'canopy aspect ratio':

$$\text{Canopy Aspect Ratio} = \frac{\text{Front canopy projection (length)}}{\text{Rear canopy projection (length)}}$$

As an example, the modified canopy designs proposed by JOY and DBT for Broadmeadow Mine in the Bowen Basin dealing with this issue are summarised in Table 3.

In Table 3, an increase of 100 mm to the rear canopy projection has a significant impact on the canopy aspect ratio, changing it from 2.6:1 to 2.4:1 (7.7 per cent) while only having a marginal negative impact on support load density (2.3 per cent). The reduction in contact pressure on the total rear area of the shield support canopy is approximately 8.2 per cent lower if the rear canopy projection was increased by 100 mm. However, in weak roof conditions if the roof crushes or breaks out 100 to 300 mm back from the goaf edge of the canopy, the influence of the 100 mm increase in canopy length is proportionally greater.

Large capacity, two-leg shields have a tendency to crush the immediate roof adjacent to the goaf edge. This causes the rear of the canopy to push up into the roof and drops the front of the canopy, increasing the effective tip to face distance. Such behaviour can be detrimental to effective roof control in weak or fractured strata. One solution involves increasing the length of the rear canopy projection. This assists by reducing the canopy aspect ratio thus improving canopy control. Geotechnically, a canopy aspect ratio of approximately 2.6:1 or even lower is preferred to achieve an optimal outcome. Serious consideration must be given to this ratio in the design phase for the particular ground conditions the roof supports are to encounter.

The control of the orientation and stability of the shield support canopy is a critical element contributing to effective roof control on a longwall. Although two-leg shield supports are equipped with a stabiliser or tilt ram between the canopy and caving shield, this device is not designed to control canopy orientation once set to the roof. The load applied to the canopy by the legs is far greater than that induced by the stabiliser ram. Consequently, once the roof support commences setting, the canopy orientation changes until the load distribution over the canopy achieves equilibrium.

As the load capacity of the two-leg shield increases, so does the contact pressure of the canopy against the roof especially in the area adjacent to the goaf edge. In weak, jointed or geologically disturbed ground, the roof above a large two-leg shield adjacent to the goaf can often be observed to break or crush, causing the rear of the canopy to raise and the tip of the canopy to lower. This effectively increases the canopy tip to face distance and can result in roof deterioration and roof falls.

TABLE 3 - Comparative canopy designs

	Original design		Final design	
Parameter Description	JOY	DBT	JOY	DBT
Total Canopy Length (mm)	4505	4420	4605	4520
Front Canopy Projection (mm)	3280	3195	3180	3195
Rear Canopy Projection (mm)	1325	1225	1425	1325
Canopy Aspect Ratio (mm)	2.47:1	2.61:1	2.23:1	2.41:1

Leg design

Due to the poor history of leg reliability in many Australian longwalls, the proposed design for each mine's support must be scrutinised and known potential problem areas evaluated. In recent years, JOY reviewed the reliability of their legs and has implemented significant design improvements to address and overcome previous weaknesses. The changes in leg design include:

1. changed test regime to 60 000 cycles with much greater closure criteria,
2. increased bearing overlap on major and minor stages,
3. increased bearing strips from two to three on the piston head,
4. changed bearing material to proprietary materials like TGA,
5. improved piston seal arrangements,
6. changed blipper valve in the smaller leg cylinder to a soft seat design to ensure that it does not stick when the supports are being lowered or advanced,
7. improved wiper ring design,
8. bronze plating of gland nut threads to prevent corrosion, and
9. top cap of the narrow cylinder on a two-stage leg is designed to minimise spherical distortion.

DBT have not experienced problems to the same extent. In particular, there have been fewer wear strip problems which may be attributable to their longer bearing overlap dimensions on the major and minor leg stages.

Walkway dimensions and ergonomics

In thicker seams with mining heights greater than 3.8 to 4.0 m, it is normal practice to use the rear walkway to ensure the safety of the face personnel. Therefore, the width of the rear walkway at waist level, needs to be considered from an ergonomic aspect to allow for lamps and self rescuers worn by underground personnel.

The horizontal dimension across an average person's waist when carrying a lamp and a self-rescuer on their belt is between 620 to 700 mm. Allowing for some practical clearance, a rear walkway width of $\geq$ 750 mm at waist height is required to permit ergonomically effective passage of people along the face so their productivity is not unnecessarily impeded. The walkway should be free of obstructions such as hoses, cables, speaker boxes, valves and contents. The width of the rear walkway at foot and ankle level also needs to be considered. Consequently, the length of the base of the roof support needs to be analysed and perhaps lengthened through the rear walkway area and maybe the canopy length increased by a similar amount to maintain the geometry of the lemniscate linkage. Alternatively or in combination with any base modification the desired width of the rear walkway can be achieved by the optimisation of the design of the mounting bracket for the valves and controls.

When the shield supports advance up to the AFC, the front walkway can be ergonomically difficult to negotiate. Features like, the slope on the toe of the pontoons, the location of the base lifting lug, the height of the spillplate and cable tray relative to waist level of an average person, may mean it is necessary to slowly walk sideways in front of the supports when the face is closed up. In lower cutting height operations, below 2.8 m, close attention to these aspects and others during the design phase should occur to ensure people can walk along the front of the roof supports safely and efficiently even when the supports are closed up.

If in the future, the shearer and/or AFC are upgraded resulting in an increase in the cross section width of the face equipment, the following changes are probable:

1. The cable tray and spill plates may have to be increased in height to clear the toe of the base during advance.
2. The width of the front walkway will be reduced between 100 and 200 mm, rendering it more difficult to move along efficiently when the shields are advanced. This loss of ergonomic efficiency will have an impact on production and the productivity of face management, operators and maintenance personnel. Loss of this amenity can have dire consequences and end up increasing costs rather than achieving the opposite objective.

Torsional rigidity

At operating heights of greater than 4.0 m, there is considerable potential for sideways movement of the support canopy relative to the base. This movement is due to accumulated tolerances in the lemniscate linkages, pin tolerances and torsional movement of the structures.

The impact of torsion on the roof support can be:

1. leg damage due to excessive torsion on the roof supports,
2. damage to side shields caused by overlap of adjacent canopies, and
3. premature cracking of lemniscate structures due to excessive torsional loads.

Generally, the original equipment manufacturers OEMs agree that the torsional rigidity of the shield support could be increased by designing the lower lemniscate links with a torsion box structure between them to increase linkage rigidity. Torsional rigidity can be further improved by reducing the accumulated tolerances through improved fabrication techniques and reducing the clearances between bores and lemniscate pins by machining the pins.

Both major OEMs have proposed similar solutions to improve torsional rigidity. These include:

1. machining of lemniscate pins and line boring of bores to reduce clearance tolerances;
2. reducing fabrication tolerances to improve the 'fit up' gaps between the lemniscate links, base and caving shield; and
3. using torsion boxes between the upper links on thick seam two-leg shields.

Side shields

The configuration and operation of side shields sparks a considerable amount of debate especially amongst operators and management operating above a 4.0 m working height. In one situation, JOY limited the side shield stroke so that the width of the support canopy did not exceed the nominal support centre spacing of 1756 mm when fully extended. This side shield design is an 'L' shaped configuration with a hydraulic travel of 100 mm. With a nominal canopy width of 1650 mm and support centres of 1756 mm, this provides no over-push, ie (1650 + 100) = 1750 mm.

Implementing this design principle appears to obviate potential damage to roof supports caused by cumulative over-push of canopies. JOY claim their experiences of roof supports operating in thick seam counters the impact of poor face management practices which can cause misalignment of the shield supports to the AFC.

One implication of JOY's design is that gaps up to 100 to 200 mm between adjacent roof support canopies could occur. This could create a risk of injury due to material falling from the canopy. Keeping the rear walkway clean would also be compromised. In a low to medium seam thickness of less than 3 m, JOY's proposal should not have serious implications. However, for operating heights of 4 to 5 m, the height from which material can fall and the gaps that can open up between adjacent supports become far more significant.

There are many longwall faces operating at 4 to 5 m high which have roof supports with over-push of side shields and where roof support alignment to the AFC is controlled through effective face management practices. However, if the over-push facility on a shield support is inappropriately managed, it can result in a scenario where the canopies became progressively offset from the bases of the shield supports due to the cumulative affect of canopy side shield over-push.

DBT propose a diametrically opposite approach with a canopy of minimum practical width (1600 mm) and side shields which extend 300 mm with an effective over-push of 150 to 1900 mm. The DBT side shield is a spring applied, hydraulic retract design which should negate the possibility of operator malpractice of pushing the roof supports canopies in the opposite direction to face creep. Since these side shields are spring applied with a load of approximately 1.5 to 3 t when extended, they do not create shield steering problems. The DBT side shields are spring applied and hydraulic retract so that the force applied to the adjacent support varies depending on the extension of the springs. This does not provide an effective means to push supports along the face but does negate the possibility of 'wind-up' in which condition the canopy is not aligned with the base in the vertical plane. Wind up occurs due to manual operation of side shield push in the opposite direction to face creep; this causes canopies to be pushed sideways resulting in stresses being placed on legs and seals.

An option involves considering adding a base steering ram to push supports sideways during shield advance. This applies load to the support closer to the centre of gravity and should not cause 'wind up' between canopies and bases. Installing a side steer ram into the centre of the base, usually below the rear walkway pushes against the adjacent base during support advance. Devices of this type have been effectively used in Eastern Europe for many years on steep seam faces. The application of base steering may also reduce the need for some of the features of canopy side shields. Prospective buyers should seriously evaluate this so that the side shield system is designed with the facility for greater travel and also consider a facility to limit the travel via a simple mechanical stop arrangement.

The principle of using canopy side shields to steer the longwall face shield at operating heights in excess of 4 m is flawed as there isn't enough force or strength or rigidity in the upper portions of the roof support structures to effectively achieve this requirement. Due to the geometry, centre of gravity and the width at the top and bottom of the two-leg thick seam shield, this practice can contribute to canopy and base misalignment problems and result in extensive and expensive mechanical damage to the roof support and in particular the legs.

The preferred normal practice is for shield supports to have between 50 and 100 mm of side shield travel in excess of that required to cover the nominal support centre spacing. This is primarily used to ensure full canopy side shield contact when the supports are negotiating undulations in the seam. Where the side shields are spring applied, the side load applied to the side shield is normally well below 7 to 8 t, and does not have the ability to effectively steer the face. In the event that the side shields have a hydraulic extension function much larger loads in the order of 20 to 30 t can be generated and these side shields can be used to steer the face. In this case, inappropriate management and operation of this facility can create canopy to base misalignment.

It should be normal operating practice that a shield support when advanced is free to realign the canopy to the base and that the side shield load applied to it during advance is equal on both sides.

Future developments

Future production capacity or reliability requirements should be considered as part of the roof support selection process. This may entail operating with a new shearer and/or AFC, which could contain the latest generation higher capacity technology. This may considerably widen the distance from faceline to the rear of the canopy compared with the existing suite of equipment.

It would not be desirable to accommodate this larger equipment by stepping the roof supports back 200 to 300 mm from the face. However, a compromise of increasing forward projection canopy length by 100 to 150 mm and making provision to step the roof support back by a further 100 mm should be assessed as part of its future acceptability. If a wider and higher capacity AFC is installed, the telescopic relay bar must have adjustment holes to allow the shield support to be brought forward 100 mm to close the tip to face distance. This facility will help maintain or offset the potential increased tip to face distance. The sensitivity of tip to face distance on roof control can be tested by stepping some of the roof supports back in 100 mm increments and monitoring the impact. Such tests can be achieved using an adjustable telescopic relay bar.

Possible cost implications

Incorporating all the required design improvements has definite cost implications. In terms of the total value of the longwall installation and the total capital investment of the project, the proposed costs should be justifiable in terms of the value added, risk minimisation and increased flexibility. All of which should provide higher production and lower costs. Additionally, these incremental upfront costs are but a fraction of the total cost of ownership for a set of roof supports. Poor production rates due to poor equipment design and selection will cost the owner of the mine far more each year than a small extra upfront capital cost. A once-off saving is false economy when compared to selecting better designed and critically assessed equipment that provides improved health and safety, gives greater production, higher operational productivity and reliability and assists in achieving a lower cost per tonne.

ARMOURED FACE CONVEYER, CRUSHER AND BEAM STAGE LOADER TECHNOLOGY

The selection of an armoured face conveyer (AFC), crusher and beam stage loader (BSL) is determined by objectively analysing the technical, mechanical, electrical and structural engineering aspects of the equipment being offered, assessing the geo-technical information available for the mine and/or using geo-technical information gained through the use of this equipment at similar or neighbouring longwall operations. Obviously, commercial considerations especially total capital cost and perhaps warranty will influence the final decision to purchase a particular set of equipment. In selecting an AFC, BSL and crusher the purchaser should focus on the total cost of ownership over the entire life of the asset.

The maingate drive arrangement contains the motor and gear reduction for the maingate drive sprocket and the fabrication for the connection to the stage loader. The discharge onto the stage loader can be a side discharge or an end discharge. Installations which have large coal lumps use a side discharge arrangement which ploughs the coal from the AFC onto the stage loader. However, if slabby stone is to be handled an end discharge could be a better choice. Virtually all AFCs delivered at the present time are side discharge.

A side discharge arrangement ploughs the coal from the AFC and guides it onto the stage loader. This type of system is able to handle large lumps of coal but the chain and flight performance can be reduced if slabby stone is encountered. The important features of this arrangement are:

1. the shape and the location of the plough plate;
2. the shape and cutaway of the AFC decking plate immediately beneath the plough plate;
3. the position of the stage loader in relation to the centre line of the conveyor belt;
4. the attachments of the stage loader to the AFC drive frame base; and
5. the structural strength of the underframe of the maingate drive frame, particularly the points of attachment for the roof supports and the connection to the AFC pan line.

The key elements of the AFC include the head and tail frames, line pans, gearboxes, sprockets, couplings and shearer haulage systems. Today, the design of this equipment is simpler and more reliable using well proven computerised design technology and manufacturing methods that aim to provide more continuous production while contributing to a safer and more ergonomically efficient face operation.

With advances in computer technology, the design of the components results in equipment being built to more exacting tolerances. Casting technology has improved and now, allows tighter tolerances and reduced use of welded fabrication of clevises, dog bone housings and line pan connections which have previously been inherently unreliable.

All suppliers of these products routinely use risk analysis methods and work to continually improve their equipment over that currently in operation by gathering and analysing operational data usually with some assistance from mine operators.

The tailgate drive is constructed to the lowest possible profile to allow the shearer to cut past the drive frame approximately 300 mm measured at the centre line of the drum. The support attachment beam of the tailgate drive and the adjoining pan section provide an attachment point for the three tailgate supports and must be robustly constructed to accept the loadings.

The face side furnishing assists in the cleanup as the conveyor is pushed over and may provide a platform for the shearer rollers. The goaf side furnishings provide a means of attachment for the roof supports to the conveyor, a means of mounting the spillplates and attachment of the chainless haulage components. The spillplates are constructed to provide a trough for the shearer cablehandler and mounting brackets for the hose and cable-runs to the centre of the face and the tailgate.

Line pans use different grades of abrasion resistant steel to improve the life of the upper deck plates. One supplier can provide a pan with an exchangeable upper trough. Dogbone ratings have been 300 t for some years and now have ratings up to 450 t. The sigma section design optimises guidance for the flight bars to minimise friction and add strength to each line pan. Use of automatic welding equipment improves product efficiency and weld reliability. Deck plate thickness can be varied to suit specific mine requirements. Pan joints or ends are now overlapped and can be machined to reduce noise, aid with an unobstructed flight path and improve deck plate alignments to enhance pan performance and operating life. The use of 48 mm chain is almost the industry standard especially on the newer or upgraded longwall systems.

Sprockets used at each end of the face tend to be single unit construction comprised of a drive shaft, inboard bearing and sprocket rings which are lubricated automatically in a fully sealed unit. Good design and fabrication methods are essential and involve enlarging the pocket profile on the sprocket to provide increased chain and sprocket life and allow the sprocket to be used on either end of the AFC. Use of a cradle support provides a safer and simpler assembly, simpler maintenance and easier replacement.

Gearbox designs tend to be completed in house by the OEM. All employ computer modelling techniques to complement their vast amounts of operating experience and knowledge. This affords the end user the choice of helical and planetary design with a proven power rating of up to 1 MW. In the foreseeable future, drives will be capable of 1.2 MW.

Drive systems predominantly use either CST gearboxes or Triple T fluid drive systems. Both systems provide high torque starting capability for long and high capacity AFCs. These systems allow motors to start up under no load and also reduce the impact of voltage drop in mines where the high tension power supply lacks robustness. Microprocessor control of the drives results in torque being progressively applied to the AFC chain as it accelerates to full speed. The use of CST gearboxes or Triple T Drives effectively controls the enormous forces generated during operation and especially at start up of the AFC and has the added benefits of increasing chain, sprocket and gearbox life.

The shearer haulage system attached to the AFC is the mechanical interface aimed at optimising shearer performance. A variety of haulage systems have been developed by the major OEMs. The different models offered by each OEM have benefits which can include retro-fitting existing pan systems with newer developed systems. The new rack systems are stronger, the sprocket teeth have a longer life due to the wider rack that allows the teeth to be made wider. Racks can be forged or fabricated. More recent designs offered have a heavy duty forged rack system designed to provide longer sprocket and trapping shoe life that can better manage severe seam undulations. A most attractive feature involves a direct drive system that increases the tunnel clearance under the shearer and a higher production rate past the shearer.

The BSL, crusher and mobile belt tail end are also critical to the achievement of a safe, efficient and productive longwall system. Optimum performance and reliability of this integrated unit is achieved through a design that complements the operating parameters of the other equipment, namely the type of material being handled and the roadway conditions.

The BSL capacity is designed to be greater than the AFC. This is achieved by increasing the width, installing a higher chain speed and by decreasing the flight spacing. The chain sections, flights and flight connections should be sufficiently robust to withstand the impacting of the crusher. There are important features to be considered in the design of a stage loader:

1. the section between the transfer point and the crusher must be flexible;
2. the section of the stage loader in contact with the floor must be underplated to prevent floor stone being gathered up by the return chain;
3. the outbye end of the crusher section should be designed to form a plough or a skid to plough aside loose floor material;
4. the drive assembly sub-base must have both horizontal and vertical movements to allow the stage loader to run along the conveyor boot end;
5. the attachments of the stage loader to the AFC must allow for horizontal angular displacement both sides of 90°; and
6. the pathway of the flight chains should be as straight as possible, particularly the return chain where it passes under the crusher.

The BSL is inherently simple and provides high availability through the use of heavy-duty components in areas like pan side sections and high impact flight pans. The beam behind the drive frame is self supporting and incorporates heavy duty line pans with spill plates. The floor mounted section contains the crusher and has a vertical articulating hinge on the outbye end. This part of the BSL can follow the contours of the main gate roadway without inducing excessive bending into the system. Inspection pans are provided to allow access to the bottom chains.

Crushers are supplied in various sizes, capacities and strengths to meet a wide range of operating conditions. All crushers operated in Australian are direct driven through a gearbox operating at high speed and with high energy. Crushers have to withstand the heavy duty and are fabricated with thick steel plate to withstand the impact of crushing the material. Vertical adjustment of the roll crusher shaft is achieved using hydraulic cylinders to make rapid adjustments to suit the size of the material being crushed.

The final component making up the AFC, crusher and the BSL unit is the mobile boot end. These units come in many different configurations, all with the objective of allowing the longwall to retreat without stopping the belt and the coal flow from longwall face. Different models include a 'Matilda' style unit as well as self propelled crawler and skid mounted units. The self propelled boot ends either use their own hydraulic pump or are powered from the roof support hydraulic system. The crawler mounted units have heavy duty track pads and have drives with internal planetary gears. Roof jacks and belt wipers are standard equipment as are the levelling jacks located on each corner for lining and levelling the boot end.

Recent developments in AFC design

There is a trend to separate wear components from structural components. For high-wear areas, very hard wear-resistant materials are being used in increasing amounts. This results in longer service life and far lower overall pan wear albeit at higher cost.

Maximising the contact surface between the flight bar and the top portion of the sigma section profile minimises the surface pressure from the flight bars onto the high-wear areas that occur especially when the AFC is being snaked during its advance and also when negotiating undulating ground conditions. Further developments and refinements in the bottom race have resulted in a large increase in the contact surface areas thus reducing the wear on the flight bar shoulder. This optimisation of contact surface areas has the additional advantage of optimising friction levels and therefore decreasing power required to drive the AFC chain. Also, the surface life of the flight bars and sigma profiles are extended and operating noise levels are lowered.

In new designs, the strength of the dog-bone has increased with breaking forces of up to 450 kN. This has forced sophisticated design improvements of forge castings such as the critical areas of the dog-bone housing and the pan itself. Consequently, dog-bone pockets now have longer service life, less danger of damaging the conveyor or the dog-bone housing. Most importantly, despite the increased strength of the dog-bones, the dog-bone is still manufactured to break first as the preferred sacrificial element during high stress spikes.

Improved design of the pan lines has maximised their stability and flexibility. Line pans can articulate up to ± 6 to 7˚ vertically and ± 0.8 to 1.2˚ horizontally. This improves the steering of the AFC allowing shorter snakes when required and better adaptation to undulating seams. These design and manufacturing improvements provide a more precise fit between pans, minimising the overall wear of pan ends, flight bars and the chain. Maintenance requirements are further reduced and trouble free operation of the chain in both directions is enhanced.

Active debate continues between all stake holders in the longwall industry about the relative merits and demerits of using cast AFC compared to rolled high alloy sigma and composite welded pans. After all the differences in material used and their design and construction are investigated it is extremely difficult to determine which type is more objectively superior. Limiting such a determination to these manufacturing variables is looking at a relatively small part of the cost of owning an AFC, BSL and crusher. The determination needs to also focus on other elements that contribute to the total cost of ownership over the life of the asset. This should include assessing the operating and maintenance practices at a mine. Clearly, if these practices are 'best in class' probably through the use of quality management techniques like Six Sigma, an AFC that is handling softer coal with lower ash, clay and silica content will have reduced wear rates. Clearly, wear rates of 0.8 mm per million tonnes at a mine should result in 25 per cent longer AFC life than at a mine whose wear rate is 1 mm per Mt. This assumes common operating and maintenance practices are employed at each mine. Clearly, this is a big assumption and difficult to assess in an objective manner. The debate about which construction is superior will continue for some years until an objective approach is adopted to determine which is more valuable to the mine operator over the whole life of the AFC.

Top-deck thicknesses have increased up to 50 mm giving a longer potential life, however, motor drives have proven ratings of up to 1 MW and couplings and gear boxes have improved. In the end it is chain technology (eg chain size, strength, metallurgy and profile) that will be the critical factor in successfully owning, operating and maintaining an AFC, BSL and crusher that provides a life in excess of 20 Mt, handles over 4000 t/h and is used on face lengths exceeding 300 m.

The next step is to develop and manufacture the technology to have AFC, BSL and crushers operating cost-effectively and routinely across a wide variety of longwall applications including face lengths of 400 m. It's not far away!

SHEARER

Shearers used for cutting coal have evolved enormously over the last fifteen years. Most developments have resulted in bigger and more powerful machines weighing up to 100 t. High capacity haulage systems now provide higher cutting speeds to complement the increased cutting power (up to 1 MW) in the ranging arms. Currently shearers have significantly better control, communication and diagnostic systems. New levels of automation have improved the efficiency for a wide variety of cutting sequences especially through the use of 'state based control' systems that can, if programmed, automatically position the ranging arms, and control shearing speeds at different positions along the face and at specified times. These controls have been achieved by installing sophisticated algorithms and feedback loops into the shearer. Also, the advent of modern control and monitoring systems across the total longwall system has allowed two way communication protocols to be established between the key pieces of equipment but with the shearer now being the central and most important component for optimising production rates from the longwall under a variety of circumstances or programmed 'status' points.

Despite these improvements, the basic components that build up a shearer have not changed except for mechanical, electrical and electronic design improvements. Shearers have these fundamental components:

1. heavy duty mainframes that sit on four 'legs' and two down drives onto the haulage components (this structure may incorporate a roll steering arrangement);
2. top or bottom mounted ranging arm cylinders;
3. sloughing plates to deflect coal falling from the face onto the face side of the shearer so it can be loaded out by the face conveyer;
4. two ranging arms;
5. two haulage units including motors;
6. direct down drives;
7. transformer box;
8. power pack;
9. pump motor for hydraulic power;
10. electrical control box which normally contains the shearer control system, (eg DBT COMPACT System), the shearer automations system, health monitoring package and the haulage control systems; and
11. auxiliary electrical material which includes the electrical control system, transmitters, audible pre-start warnings and methanometers.

The ranging arms contain the gear train to transmit the power to the cutting drum. The important features of this expensive part of the machine are:

1. the ability to allow the drums to reach out into the tailgate and maingate and cut the clearance for the AFC drive frames,
2. the structural strength capable of withstanding and transmitting the cutting forces back to the body of the machine,
3. the final planetary drive to the shearer drums capable of accepting the full load torque of the motor,
4. the design of the outside diameter minimised to allow the best possible vane depth to be achieved in the cutting drums,
5. the provision of hollow shaft venturi ventilation from the face side of the cutting drums, and
6. the location and design of the hydraulic ranging cylinders such as to cause minimum restriction to the coal flow onto the conveyor.

The spiral-vane cutting drums must be designed to suit the cutting and loading characteristics of the coal seam in conjunction with drum rotational speed, shearer linear speed and gauge length of the picks in use. Drum selection should take into account:

1. number of vanes or starts;
2. number of picks/line;
3. distance between the pick lines;
4. variable pitch angle of the vanes across the drum;
5. angle of wrap of the vanes;
6. depth of the vanes to be adequate to load the coal but not unnecessarily deep so as to create zones of stagnant coal;
7. shape and the clearance of the face side of the drum;
8. structural design;
9. selection and type of the cutter pick and, in particular its length;
10. maximum cutting speed of the shearer;
11. revolutions per minute of the cutting drum; and
12. water spray system and the size of water passages.

Dust suppression is achieved by water spray systems supported by a high pressure boost pump. The primary spray system consists of sprays in the shearer drums, which provide pick face and pick body flushing. These sprays are generally of a quick-release type with 1 to 1.5 mm spray jet size. The drums may also have a spray passage through the pick body which provides a spray immediately behind the pick cutting tip. The water supply to the drum sprays is usually limited only by the pressure rating of the rotary seals in the ranging arm and perhaps the oil cooling elements in the gear cases. The aim of the dust suppression system is to contain the dust between the shearer and the coal face. The spray nozzles are placed to direct the spray water in the same direction as the ventilating air.

A second circuit of the spray system supplies water to a series of sprays placed along the top of the machine and along the ranging arms to form a screen of spray water endeavouring to contain the dust against the coal face. These sprays are positioned to spray in the direction of the air flow. The operating pressure to this circuit is 1 MPa with very fine spray nozzles.

New technology being introduced but not yet widely used in the underground coal industry to aid communications between system components includes wireless ethernet which is a new proven innovation for communication between control systems. In addition, there is now an ability to fit an inertial navigation system as an integral unit into a special compartment as a research and development initiative to automate face alignment and ultimately horizon control across the entire face length.

Other factors to be considered when purchasing a shearer from an OEM include completing or obtaining:

1. Compatibility trials with other provided equipment.
2. Mini-build for operator and maintenance training.
3. Overseas freight.
4. Landing and clearing charges.
5. Inland freight.
6. Documentation relating to equipment approvals and certification to ensure the shearer meets statutory compliance requirements. Other documentation includes parts manuals, general arrangement drawings and finally maintenance operating manuals. These materials are now supplied in both hard copy and in an electronic format.
7. Competency based training of operators and maintenance personal following on from a site-needs analysis and risk assessment.
8. Project management for supply of the shearer from placement of the order through to delivery and commissioning the equipment. Once again, the application of risk management techniques and the consequential outcomes need to be considered.
9. Payment of import duty which is currently charged at three per cent of the imported value unless a concession to remove this duty is made under an Australian Government initiative known as the Enhanced Project By-law Scheme (EPBS). This duty also applies to the importation of other equipment that makes up the longwall operating system.
10. Site service engineering commitment that is normally required for the compatibility and mini-build and for the site installation of this equipment, and may include underground installation and commissioning, and the initial three months production during which time acceptance trials are conducted.

Often, the shearer is inspected prior to the end of each longwall block and especially during the mining of the first longwall. Assistance is provided with the first longwall transfer and the first weeks of production on the second longwall panel. Usually payment for this service is made to the OEM at their agreed schedule of rates.

Cutting methods and sequences

With older control technology and equipment, increased production is difficult to attain. However, in recent years many cutting options and features have been developed in the design software that allow production increases for little or no additional cost. In many cases, mine management may not be aware that these options or technologies exist. It is important for mine management to keep updated on these technological improvements.

Looking back ten years, the cutting cycle options available were bi-directional or uni-directional cutting employed on the retreat longwall extraction method. Nothing is gained by remaining with the same old system 'because that is all we have ever used', when alternatives are available and may prove more efficient and cost effective. Changing the cutting system may provide the solution, or at least another alternative, to maximise productivity, and improve the environmental and face management practices.

Technology has changed the way the industry can manage longwalls. Today, mines commonly use bi-directional cutting or uni-directional cutting with a backward, forward or reverse snake, depending on geological, environmental or management conditions. Recently, mines have been looking at the half web system and variations on this method as an alternative to the old systems.

The difficulty is, to determine which system is best for a particular mine. Only personnel familiar with the mine environment and current longwall technologies will know the benefits, risks and costs that will provide the correct answer. A mine that has always used one system may not know whether another system would better suit their conditions and provide health and safety benefits and production increases at an acceptable cost.

The majority of longwall operations in Australia use a uni-directional cutting technique that provides simple operation, environmental benefits, minimal manning and improved horizon control. However, in a cost-driven, supply-sensitive and thus competitive marketplace, using the conventional or traditional mining sequences or cutting cycles may not allow the equipment to realise its production potential. As companies strive for a competitive edge, the market leaders seek to find new methods to gain small productivity increments that will positively re-position them on the industry segment cost curve.

The use of bi-directional cutting has always been seen as more productive, especially on longer faces. As shearer power and haulage speeds have increased, thicker seams are targeted and mine environmental issues require greater consideration, uni-directional cutting has become more competitive and in some cases can be more productive.

In addition, levels of automation, computer power and programming have increased. Computer power on longwall faces has enabled support control systems to become more sophisticated, allowing further alternative cutting systems to be developed. An example is the half-web cutting cycle, used in one particular form at Twenty-Mile mine in the USA.

The half-web cutting system can be used as a viable alternative to conventional systems. Rutherford (2005) has developed a cutting variant on the system that is suitable for Australian thick seam conditions.

The half-web system is basically a uni-directional system of extraction in mid face, with bi-directional gate sequences. Faster shearing speeds and cutting cycles may be possible as the 'shuffle' required for the bi-directional cutting is not necessary and loading on the shearer drums is reduced.

The half-web is achieved by pushing the AFC forward 50 per cent, after the supports have advanced to provide a half web cut for the return shearer run. Using mid seam drum positioning, a pre-splitting effect on the coal occurs, thus reducing lumps. In the right seam conditions and if managed correctly, this technique can equalise the coal flow in each cutting direction. The half-web system requires a modern support control system, such as the Joy RS20 or DBT PM4 system (or newer generation systems) to operate effectively. Support hydraulics need to be well maintained, highly efficient and controlled with tight discipline. Consequently, significant cutting cycle benefits should arise resulting in an increased cutting rate.

In the last 12 to18 months, trials have been completed at a number of mines using the half-web system over a number of shifts and significant benefits and improvements have been seen in the following matters:

1. production cycle times (up to 40 per cent improvement),
2. reduced lump size and blockages on the AFC and BSL,
3. coal handling,
4. equipment loading and wear rates,
5. environmental management, and
6. face management and control.

However, there has been a poor response to the use of this innovative system. Perhaps the industry is still too traditionally focused and slow to change and there is a lack of understanding of how the system could operate in Australian conditions. One area of concern is the requirement to have a support control system available 100 per cent of the operational time. Control system reliability is now extremely important as is the need to train operators, tradespeople and engineers to be competent in the use and maintenance of these more complex systems and programs.

In the past, roof support control systems have been used until their maintenance efficiency has reduced productivity to such a low level that the longwall is no longer cost effective. Renewing the support control system, at a cost of over $4 million depending on face length, will increase production and productivity levels. However, the support structures and hydraulic system must be suitable to operate at the higher cycle times provided by these latest generation control systems. If not, the investment may be ineffective because the down-stream or consequential flow-on effects have not been properly evaluated as part of a robust management process.

Mine operators should not shy away from purchasing new longwall equipment on the basis that existing equipment it is only ten years old. The longwall technology has advanced substantially in the last ten years and the Australian industry must invest in this technology to remain competitive with open cuts and new developments overseas. The Australian industry must look at its current operations more critically, actively consider embracing the new technology especially electronics used in outside industries and positively manage the development and implementation of automation and control systems onto new or upgraded equipment.

Bi-directional cutting system

Bi-directional (Bi-di) cutting is described as ‘the full web extraction of the seam in the one pass along the longwall face’. Each time the shearer cuts from one end of the face to the other, a full web of coal is extracted. As the shearer cuts along the face, the supports are advanced and then the AFC is advanced following the supports.

To enable full extraction to occur in each direction, the shearer must be ‘shuffled’ into the next cut at each end of the face, by running back around the snake and then back into the gate. The ‘shuffle’ sequence takes time and can increase wear on the equipment if the snake is too tight. This need to make the ‘shuffle’, makes Bi-di cutting inefficient on short faces of less than 180 m length. Depending on shearer speed and support system capabilities, gate end turnarounds (time spent double cleaning floor and changing shearer direction), and time spent doing the ‘shuffles’ are critical to cycle times.

Bi-di cutting can be environmentally less desirable than alternatives if the system generates high volumes of dust, either from the shearer or support advance. Operators should restrict activities on the return side of the shearer for health and safety reasons and it is necessary for the support control system to automatically initiate the support cycle. The development of automation associated with new roof support control systems can eliminate the need for a roof support operator to work on the return side of the shearer. However, it is essential that the system is well maintained and operated.

Bi-di cutting may give added support in poor ground, allowing ‘double chocking’, in some instances. In thick seams, the full extraction may not be possible without larger diameter drums and slower cutting speeds, which may have a negative impact on production levels.

Critical factors that may limit the use of Bi-di cutting are:

1. the age and reliability of the support control system;
2. two shearer operators may be required;
3. horizon control needs to be high quality;
4. AFC/BSL and outbye system capacity may determine cutting speeds;
5. lack of flexibility in the cutting cycle, requiring operators to have a good knowledge of the system problems and faults; and
6. the need for additional operators if automation is at a low technical level.

Beneficial factors of Bi-di cutting may be:

1. reduced demand on support system requirements (especially in thick seams),
2. the ability to support the face better in poor conditions,

3. greater benefits in thinner seams where clearance under supports is critical, and
4. snaking in both directions will keep face creep to a minimum.

Bi-di cutting is shown in Figure 3 in which the following cycle time can be estimated:

Cycle time 200 metre face @ 10 m/min

= main cut + shuffle + turnaround

= 200/10 + (2 x 32)/10 + (3 x 3)

= 35.4 min

Uni-directional cutting system

Uni-directional (Uni-di) cutting is described as 'the extraction of the web in two passes across the longwall face'. In basic terms the shearer must pass across the face twice to extract the web. On the first cutting pass, the supports are advanced and on the return pass the AFC is advanced.

Uni-di cutting removes the need to 'shuffle' the shearer into the next web at each end of the face, reducing the time spent at each end. This is offset by the need to cross the face twice for each cut, requiring the shearer to travel faster to equate to Bi-di cutting.

On a 200 m face the shearer speed must be increased to >13 m/min to cycle at the same rate as Bi-di cutting at 10 m/min.

Critical factors that may limit the use of Uni-di cutting are:

1. the increased speed of the shearer may increase demand on the support system,
2. the need to have a faster haulage speed on the shearer,
3. additional operators may be needed if automation is at a low technical level,
4. support advance in the tailgate has the same environmental effects as bi-directional cutting (if a reverse snake is used), and
5. creep may be caused by snaking in one direction each shear

Beneficial factors may be:

1. training requirements maybe reduced because the system is simpler,
2. only one drum needs to be controlled across the cycle,
3. more environmentally desirable because support advance is designed to be on the return side of the operator (depending on snake design at the tailgate),

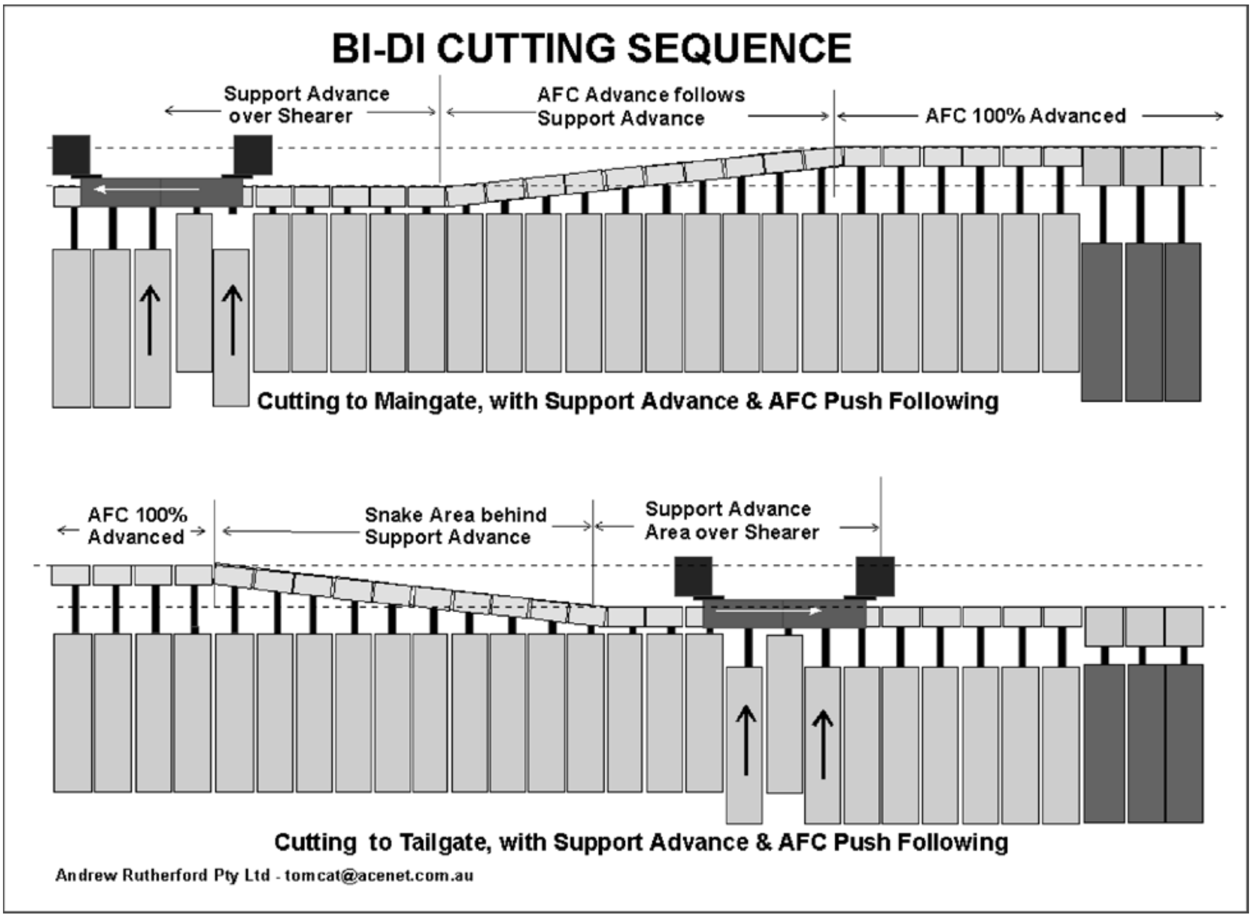

FIG 3 - Bi-di cutting sequence (after Rutherford, 2001)

4. loading on the equipment can be more easily regulated,
5. AFC/BSL loading and outbye system can be regulated by varying extraction on each pass (especially in thick seams),
6. horizon control is less critical as the AFC can be pulled back to re-grade the floor, and
7. there is greater flexibility in cutting cycle and support control system operation.

Uni-di cutting is shown in Figure 4 in which the following cycle times can be estimated:

Cycle time 200 metre face @ 10 m/min

= main cut + turnaround

= 2 x 200/10 + (2 x 3)

= 46 min

Cycle time 200 metre face @ 13 m/min

= main cut + turnaround

= 2 x 200/13 + (2 x 3)

= 36.8 min

Shearer speeds are now commonly over 20 m per minute, allowing Uni-di cutting to be competitive with bi-directional cutting on longer faces.

There are a number of variations on Uni-di cutting that can be used for environmental reasons, usually for reducing personnel exposure to airborne dust.

Forward snake

The conventional Uni-di cutting system has a 'backward' snake at the tailgate, requiring the shearer to cut near the roof going into the tailgate. Therefore, the roof supports must be advanced on the intake side of the shearer operator when cutting into the tailgate.

In some situations, the dust created from the support advance makes this system environmentally undesirable.

To alleviate this dust problem, the system can be run with an 'advanced or forward' snake at the tailgate. This removes the need to advance supports on the intake side, but requires the AFC to be snaked in 'reverse' (towards the maingate), or normally with a 'double snake' being formed on the face as shown in Figure 5.

Conventionally operated Uni-di cutting systems often switch to a 'reverse' snake to reduce the rate of creep towards a particular gateroad when snaking in one direction all the time. In general, the Uni-di cutting option is more tolerant of operator and equipment faults or problems and for this reason is more widely used.

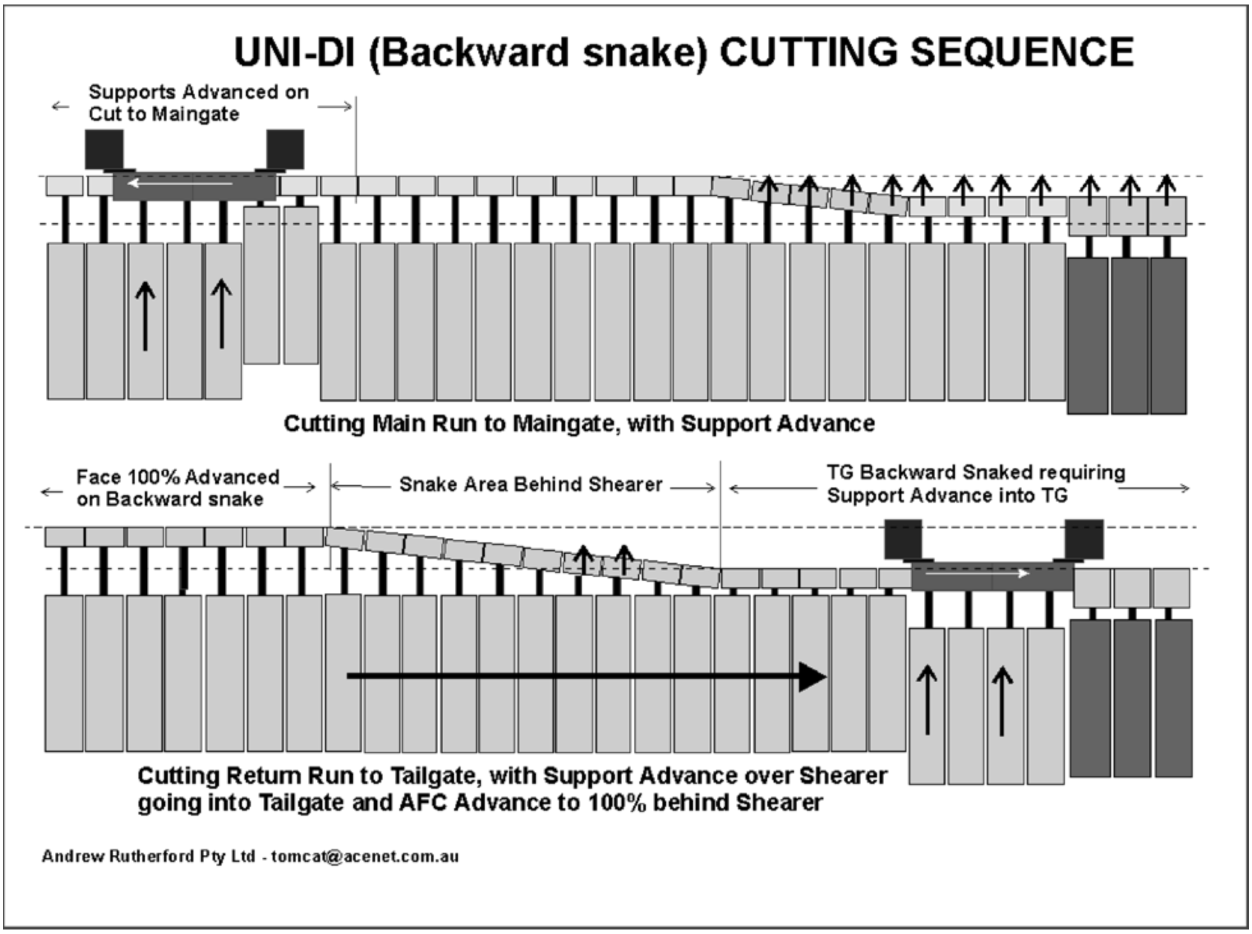

FIG 4 - Uni-directional cutting sequence with backward snake (after Rutherford, 2001)

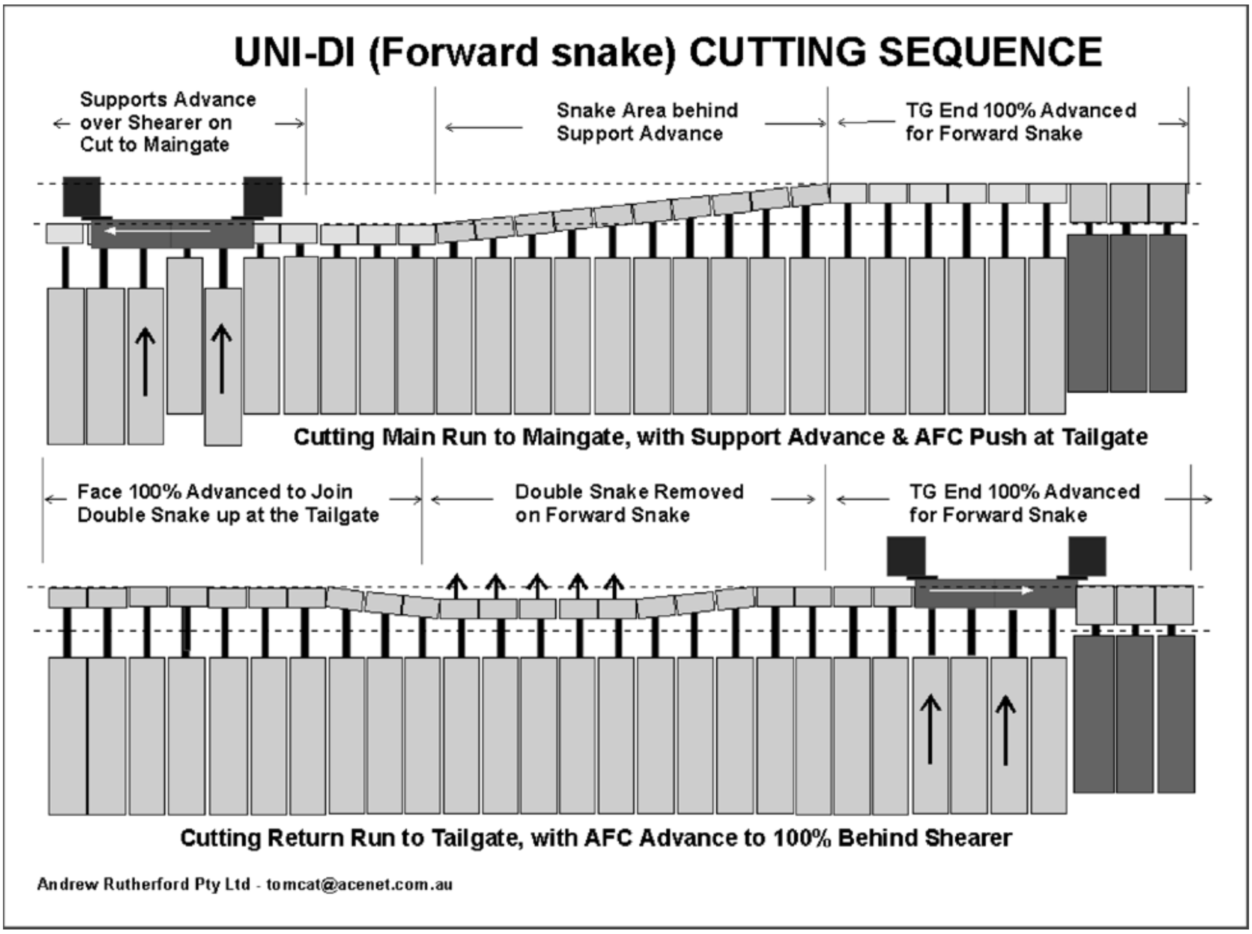

FIG 5 - Uni-directional cutting sequence with forward snake (after Rutherford, 2001)

Half web

The half web cutting system has been very productively used at Twenty-Mile Mine in the USA for several years. The system provides a reduced loading on the equipment and allows the ground stress levels or weighting on the face to break the coal. This outcome provides higher production rates as the shearer speed can be increased because of a lower cutting load per pass. These particular geological conditions allow greater production levels to be consistently achieved. The benefits from implementing this system at Twenty-Mile Mine allowed an investment; in higher capacity equipment (with faster shearer speed, faster support operation and better outbye systems) to be justified to take advantage of the conditions,

Australian conditions do not match those of Twenty-Mile mine. For this reason and perhaps because of a lack of understanding of the system, the half web has not been used until recently in this country. Trials have been completed at Dartbrook, Metropolitan, Kestrel, South Bulga and Moonee, all with positive results. However, for various reasons the system has not been readily accepted.

Half web is a mixture of a Uni-di (mid-face) and Bi-di (face-ends) system. It can be considered a Uni-di system because it requires two passes of the face to extract one web of coal.

The roof support control system must be of advanced technology to enable the efficient use of the Bi-di configuration for the cut to the maingate, where the 50 per cent push follows the support advance. Following this, the uni-di system is employed for the run back to the tailgate where the AFC is pushed to 100 per cent following the shearer. It therefore requires a different or more complex program to operate in high levels of automation and control.

The environmental benefits of reduced dust exposure remain as for the Uni-di system because all the supports are advanced on the return side of the operator.

The use of bi-di operations requires support operation reliability to be high. The support control system must be effecient, modern and reliable, eg the RS20 system from Joy or the PM4 system from DBT.

The support electro-hydraulic system must incorporate reed rods and check valves in the push rams to accommodate the 50 per cent push. A reliable monitoring system can be a benefit to operators where supports do not advance correctly.

The potential advantages of the system are:

1. distances travelled by the shearer are the same as with Uni-di cutting;
2. environmental conditions are superior to both Bi-di and Uni-di and equate to the 'forward' snake conditions in Uni-di;

3. assuming mining conditions are acceptable, shearer speeds can be increased as less coal is cut in the main cut, increasing potential cutting and production rates;
4. the cutting profile is used to reduce lumps by pre-cutting the seam on each run up and down the face;
5. the 50 per cent push from tailgate allows a greater distance between face and AFC spillplates for moving lumps more easily;
6. drum positions can by varied to allow the tonnage cut to be more evenly distributed on each pass of the face, a great benefit in thick seams;
7. half web can be varied within the support control system program to allow the advance to be varied to suit mine conditions and potential problems;
8. snaking distances are shorter and less severe, reducing wear and damage potential;
9. snaking distances are halved and so cutting into gates is quicker;
10. creep may be better controlled as snaking is done in both directions;
11. as in a Uni-di system, the majority of horizon control cutting is done with the maingate drum so no tailgate operator is required; and
12. it allows auto-steering to be used as in Uni-di cutting and therefore shearer operations remain similar to existing conditions.

The potential disadvantages are:

1. as in a Bi-di system, it is a slightly more complex operation due to AFC push in both directions;
2. can be complex when manually operating the system and may require the operator to work on the return side of support advance;
3. if coal strength is weak, the tip to face distance can increase when top coal falls, adding to the potential for roof falls in front of the roof supports;
4. the system does give the potential for lumps to be trapped on the tailgate side of the shearer which can slow operations; and
5. as in the Bi-di system, roof support and AFC advance occurs behind the operators on the cutting run to the maingate.

There are two variations of the half web system and each has particular advantages in certain conditions.

Kaiser half web

The Kaiser half web system (similar to that used at Twenty-Mile mine) is probably best suited to lower height seams and is used to precut the seam to reduce loading on the shearer and allow a faster shearer speed during the main cut. In thinner seams, the loading on the AFC is not balanced in both directions, unless the drum diameter is matched to the extraction height. This may not be possible in hard cutting conditions or in very low seams.

The Kaiser system is used at Twenty-Mile mine with shearer speeds at >30 m per minute and generates up to 1 Mt per month production. The system is ideal for their particular conditions, with coal bursting across the face, and precutting the centre utilises the strata effects to maximise cutting rates.

The system of precutting the face produces an undercut that allows roof and floor stone to be trimmed more easily on the return run. An additional benefit is that the precut can split lumps that may fall off the face, having a beneficial effect on production by reducing downtime for lump breaking across the face, behind the shearer or at the maingate.

The potential benefits of the system are:

1. reduced loading on the shearer compared to conventional systems,
2. a faster shearer speed can be allowed,
3. the shorter snake allows the shearer a higher speed further into the gate,
4. reduced lumps on the AFC, and
5. there is higher potential output when shearer drums are designed and matched to seam height.

Thick seam variant

The thick seam variant of half web system developed by Rutherford (2001) is well suited to Australian thick seams, and has many advantages over other methods. It has the following benefits:

1. improved load balancing on the AFC/conveyor system compared to other systems;
2. reduced loading on the shearer and AFC compared to other systems, due to the half web and half seam cut;
3. shearers can operate at high speeds with reduced loading and power requirements;
4. reduced lumps on the AFC;
5. the system supports slumping faces by leaving a bench;
6. higher output potential when shearer drums are designed to match seam height; and
7. clean-up is improved because depending on drum design there is les material to be loaded.

Precutting the face increases the free faces available to break out the coal, reduces the loading on the shearer drums and the production of large lumps from a slumping face by splitting them before they spall from the face.

The half web system illustrated in Figure 6 can support a slumping face by leaving a bench. By varying the half push, any roof coal that falls can be retained on the face side to allow the shearer to cut prior to loading. Cycle times can be estimated as follows:

Cycle time 200 metre face @ 10 m/min

= main cut + turnaround (Bi-di cut)

= 2 x 200/10 + (2 x 4)

= 48 min

Cycle time 200 metre face @ 15 m/min

= main cut + turnaround

= 2 x 200/15 + (2 x 4)

= 34.6 min

The half web can allow a faster shearer speed than available for Bi-di or Uni-di, especially in the thicker seams when loading needs to be balanced to optimise the full conveyor system capacity. For example, the cutting profiles show that for a given thick seam, Uni-di cutting would generally take 66 per cent of the seam on the main cut, whereas half web can balance this to 50 per cent, depending on coal characteristics. If the shearer speed is equated to volume cut, then the potential production increase will be 32 per cent. Furthermore, depending on the drum design the cleanup might be improved due to the reduction in material to be moved across to the AFC.

With reduced loading on the shearer, with fewer lump problems and balanced output, the potential to run faster in both directions allows half web to be a higher and more productive system, especially in thick seams, and should be considered by all operations with modern support control systems

General considerations

To obtain the best from a longwall system, designers must plan for maximum output with shearer operating speeds >30 m per minute. This will set new challenges for the roof support and AFC manufacturers who must design systems to match the pace of the shearer. Such changes increase the imperative to integrate further automation across the entire longwall system. This development will increase production and productivity rates, improve health and safety for all face operators by reducing their personal exposure to airborne dust and proximity to ultra high pressure hoses and allow standardisation of the longwall operating system by reducing variation through less manual intervention.

In Table 4 a comparison has been made between the cutting systems and identifies the reasons why Uni-di has been the preferred method of cutting in recent years and how the half web method may improve productivity options in the future.

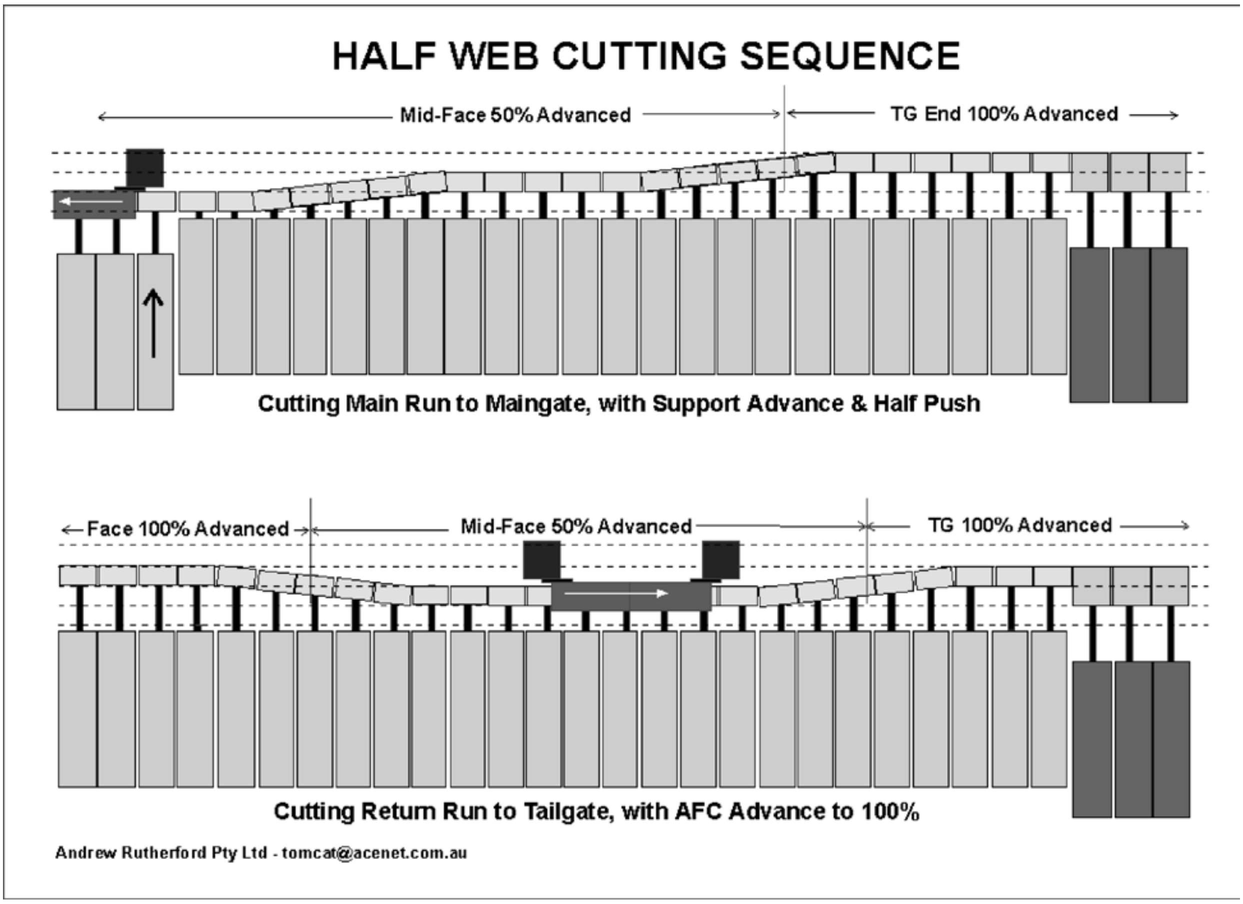

FIG 6 - Half web cutting sequence (after Rutherford, 2001)

TABLE 4 - Summary of system benefits and difficulties

Cutting System	Bi-directional Cutting	Uni-directional Cutting	Half Web Kaiser Cut	Half Web High Seam Variant
Relative complexity of system	Most complex	Simple	Complex because Bi-di cut is needed with shearer	More akin to Uni-di
Cycle time on 200 m face @ 10 m/min	36	46	46	48
Faster shearer speed needed for same level of production	No	Yes - >13 m/min	Yes - >13 m/min	Yes - 15 m/min
AFC loading:				
▪ To maingate	Standard	Reduced	Reduced	Lower and balanced
▪ To tailgate	Standard	Substantially lower	Substantially lower	Lower and balanced
Shuffle required	Yes	No	No	No
Support advance on intake side	Yes	Yes (in TG backward) No (if forward snake)	No	No
Snake length	Standard	Standard	Half standard but two half snakes	Half standard but two half snakes
Double snake on face	No	No (for backward) Yes (for forward)	Yes (only 50%)	Yes (only 50%)
Balanced AFC loading	Yes	No	No (unless drum diameter matched to seam)	Yes (high seam especially)
Reduced loading on shearer	No	Yes (depending on seam section taken on main cut)	Yes	Yes
Precutting face	No	No	Yes - middle of face	Yes (top & middle of face)
Bench to support face	No	Yes	Minimal	Yes
Reduced slumping in front of shearer	No	No	No	Yes
Reduced lumps traveling to maingate	No	No	Yes	Yes
Improved loading of loose coal	No	No	No	Yes
Geological benefits				
▪ Poor roof can be double chocked	Yes	No	No	No
▪ Weak floor can be easily monitored and corrected	No	Yes	No	No
Wider clearance on face side for coal	No	No	Yes	Yes
Variation of cutting volume:				
▪ Vertically	No	Yes	No (in thin seam)	Yes
▪ Horizontally	No	No	Yes	Yes
Can be manually operated	Yes - but with airborne dust risks	Yes - simple	Yes (but with air borne dust risks)	Yes - but with air borne dust risks
Likelihood of lumps on TG side of shearer	Yes	No (except with backward snake at TG end)	Some	Less likely
Tip to face distance changed	No	No	In weak coal the undercut can fall and increase the tip to face distance	In weak coal the undercut can fall and increase the tip to face distance
Autosteering be used on shearer	Yes	Yes	Yes	Yes
Is creep affected by system	Less	More	Less	Less
Can single operator be used on shearer	Yes - if auto-steering is operating	Yes	Yes (if auto-steering is operating as Bi-di)	Yes (as per Uni-di)

LONGWALL TOP COAL CAVING

Longwall Top Coal Caving (LTCC) has been developed and operated in China for the last 20 years and with over 100 faces in operation producing over 200 Mt it is a mature technology in Chinese conditions.

In Australia, there is a significant potential for an underground thick seam mining method such as LTCC to be introduced. Currently there are measured thick seam resources of 6.4 Bt situated in both New South Wales and Queensland. The Australian Coal Association Research Program (ACARP) recognised the potential and funded a study into the feasibility of introducing LTCC into Australia (Cai *et al*, 2004).

The technical challenges involved in the introduction of LTCC need to be addressed pro-actively before it will gain proven recognition in Australia. The successful introduction of LTCC at the Austar Mine in the Hunter Valley in 2006 by Yancoal Australia Pty Ltd will boost the interest in the introduction of this innovative mining method in NSW and Queensland. LTCC could provide the industry with a means of effectively mining thick seams in a safer and more productive manner.

As outlined by (Cai *et al*, 2004) the major perceived benefits of the LTCC method for Australia include:

1. Reduced operating costs are possible as the LTCC method increases the longwall tonnes, per metre retreat and also per metre of gateroad development. This method will significantly reduce development costs and lessen the risk of longwall development not being completed to the required schedule. Other unit costs such as labour, power and face equipment maintenance will be significantly reduced.
2. The LTCC method provides greater resource recovery: as it offers a viable means of extracting up to 75 to 80 per cent of 5 to 9 m thick seams. These seams constitute over 50 per cent of Australia's measured and indicated underground mineable thick seam resources. Currently, single pass longwall is only proven to be viable at a few mines in Australia at an upper height of 4.5 to 4.8 m. Improved resource recovery increases mine life, aids sustainable development and improves 'life of mine' financial performance through the ability to depreciate major project infrastructure costs and financing over a larger mineable reserve.
3. Mine safety is improved as the cutting height is lowered relative to the high reach single pass longwall method. This results in improved roof, face alignment and roof support control, smaller and less expensive equipment and improved spontaneous combustion control, in thick seams, through removal of the majority of top coal from the goaf. However, managing airborne dust and strata control in the maingate area will require significant efforts.

Resources

The Australian resource database shows in excess of 6.4 Bt of underground thick seam coal (>4.5 m) in the 'measured resource' category, rising to 17.5 Bt when 'indicated resources' are included. Of this resource, 25 per cent is located in NSW and 75 per cent in Queensland. So from a resource viewpoint, there is significant potential for the application of LTCC in Australia.

Based on Chinese experience, the most important parameters for the success of LTCC are coal strength and mining depth. In addition for an effective application, the coal seam must be easily breakable (friable) or cave easily with small blocks that have weak to moderate strength of less than 15 Mpa. Analysis has shown that 57 per cent of the relevant seams in Australia are reasonably weak with a strength of less than 15 MPa and 29 per cent are moderately weak with the strength of between 15 and 25 MPa The remaining 21 per cent have strengths exceeding 25 MPa.

Figure 7 shows the potential LTCC seams according to their depth of cover.

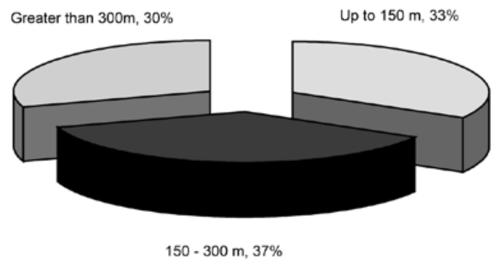

FIG 7 - Average depth of the potential LTCC seams

Operation

Longwall top coal caving is a method of extracting thick seams greater than 4.5 m. The method employs both cutting of the lower portion of the coal seam accompanied by caving and reclamation of the 'top' coal. Coal is first cut from the longwall face using a conventional shearer and AFC arrangement working under hydraulic face supports that incorporate a rear coal conveyor and cantilever/ flipper arrangement located on the lower portion of the rear canopy. Face cutting heights are generally in the range of 2.8 to 3.0 m. The roof support is advanced forward after the shear and the rear conveyor remains in place in preparation for the caving sequence. The caving sequence allows the broken coal at the rear of the supports to flow from the goaf onto the rear conveyor which conveys it to the gate end transfer. The flow of coal onto the rear conveyor is controlled by retracting the rear cantilevers of selected supports exposing the rear conveyor to the goaf coal which 'caves' into the free space. Once an area has been caved the rear cantilever is extended back out into the goaf stopping any further influx of goaf material. A secondary caving process may be repeated at the same position if further coal is present before the rear conveyor is finally advanced forward under the rear canopy of the support ready for the next shearer cycle.

Depending on the conditions in the mine, various caving sequences are employed to maximise the top coal recovery. In many cases, top coal caving is the primary production mechanism from the face rather than coal cutting, with overall face cycle times also dependent on caving rates rather than shearing rates.

Gaining a fundamental understanding of the theory and principles behind the caving process and the importance of coal strength and vertical stress relationships is critical to achieving a successful application. Learning from the Chinese coal fields experience must occur so that this knowledge can be translated and applied to Australian conditions.

Challenges to introducing LTCC

There are three main challenges to introducing LTCC into the Australian industry.

Geological and geotechnical matters

Scientific and engineering studies and detailed investigations are required to determine a particular site's potential for LTCC. To begin, there is a need to understand the theory of LTCC and apply it to that particular site's geological and geotechnical enviroment.

The process of fracturing and crack evolution in the top coal is critical to the success of LTCC and is dependent on abutment pressure and coal mass strength (Zhongming *et al*, 1999). Top coal fracturing occurs through shear failure and tensile cracking. Poor fracturing will cause larger blocks to form and poor caving through the rear AFC will result. Excessive fracturing will in turn cause roof control issues ahead of the roof supports. The fracturing process begins ahead of the faceline when the coal seam is acted on by abutment stress. Secondly, the top coal with little or no horizontal confinement undergoes horizontal dilation once acted upon by vertical stress before final caving at the rear of the LTCC supports.

Estimation through modelling of the degree of fracturing occurring during this cycle is at the core of predicting LTCC production. Figure 8 is a simplified illustration of the stress regime surrounding an LTCC face and its effects on the top coal.

Caving assessment

The Chinese Research Institutes have developed numerous methods for assessing the caving characteristics of their mines based on empirical methods, laboratory testing and experience. The methods for calculating these characteristics rely primarily on the following parameters:

1. coal strength,
2. vertical stress,
3. top coal thickness,
4. interburden/stone band thickness, and
5. degree of fracturing.

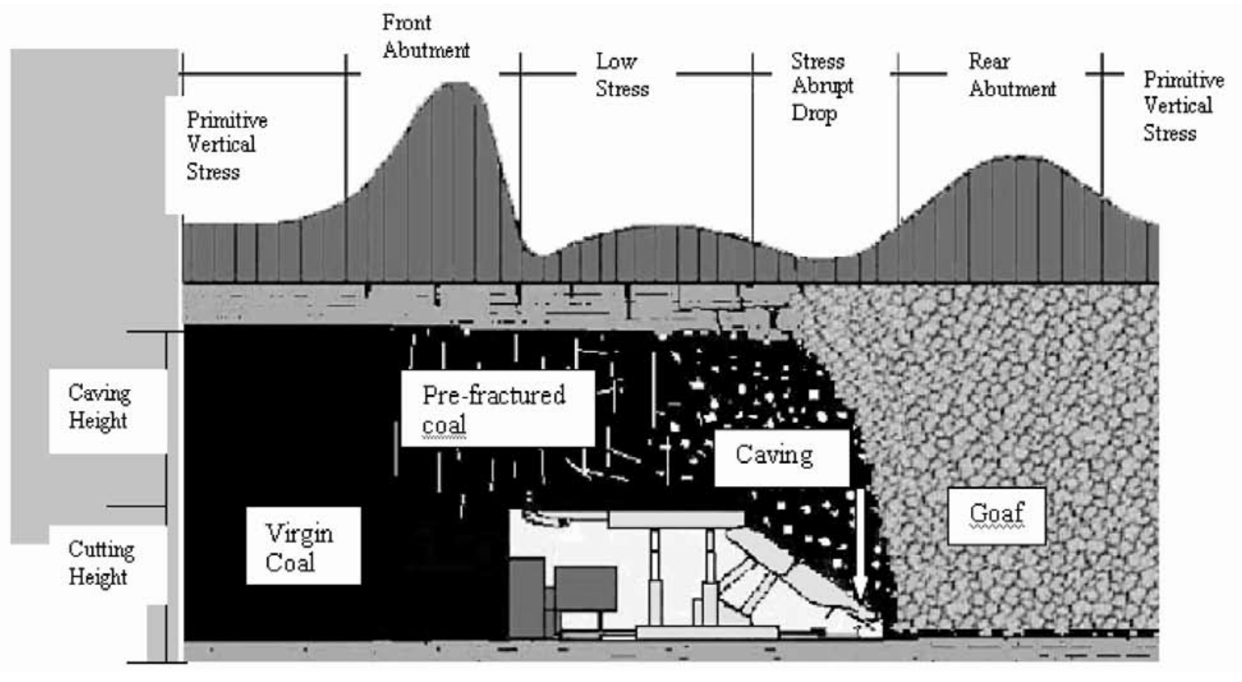

FIG 8 - A conceptual model of the LTCC method (after XU, 1999)

Other methods incorporate support resistance values or more detailed coal characteristics such as coal cohesion in the calculation. Once the caving conditions have been assessed the appropriate equipment is selected and the caving technique applied based on the calculation of the caving index for the deposit. The index relates to seam recovery and a generic assessment of the conditions. Table 5 prepared by Zhongming Jin (2001) illustrates the relationship between cavins index and recovery for different mining conditions.

Australian mining conditions are quite different from the Chinese coalfields, Chinese LTCC mines are deeper at around 600 m with moderately hard coals whereas Australian mines are relative shallow at 250 to 350 m and with relatively softer coals.

Parameters such as the 'degree of fracturing' are difficult to observe, measure or calculate and cannot be determined for Australian mines. Tools must be developed to enable a comparison of Australian mining parameters against current Chinese LTCC measurement indexes. One possible method involves developing the relationship between plastic strain modelled in the top coal of an LTCC face against the Chinese caving index for the same conditions. Using this approach a relationship between known Australian parameters and the Chinese rating index can be developed.

Modelling of the caving process is crucial to understanding and predicting the caving behavior that can be expected under Australian conditions. Such modelling would enable a specific classification system to be developed and the performance of the roof supports in these conditions to be assessed. Modelling of LTCC is complicated as the strata must be modelled and assessed through the various loading stages that result in increasing fracturing of the top coal, from being intact to fully fractured and expanded. This requirement necessitates using two different approaches known as continuum and discrete element modelling to represent the two coal conditions.

Mining environment

Use of the LTCC method causes changes to the mining environment particularly in airflow patterns, gas emissions and airborne dust generation. Historically in China, LTCC uses four legged shield supports with a rear AFC which significantly alters ventilation behaviour on and around the face.

Additional airborne dust will be present on LTCC faces due to the addition of the caving cycle that produces larger quantities of dust. The two distinct cutting and caving operations involved in LTCC are staggered by a distance of up to ten roof supports. As working the caving sequence involves operators being located on the return side of the shearer, dust exposure issues arise because currently operators cannot remotely operate the caving cycle from the intake side of the shearer. Bi-di cutting sequences followed by a caving cycle every second shear may be employed to reduce an operator's exposure to dust. Using this method in Australia will require new techniques to keep the dust generated during the caving cycle away from the shearer operators and towards the rear of the supports. Bi-di cutting may also require the use of a higher capacity rear conveyor to maintain caving output as it occurs only every second shear.

Automation of the roof supports, AFC and caving process to remove personnel completely from any dusty conditions is the best solution and will require the involvement of equipment manufacturers, researchers and mine operators alike.

Chinese LTCC mines have typically lower gas contents than Australian mines. Gas make into and out of LTCC goaves may be significantly different compared to a conventional longwall goaf in the same conditions due to the larger coal extraction volume and its effect on the surrounding strata and coal seams.

Similarly gas accumulations and ventilation dead spots may occur on and around the rear AFC. Maintenance and repairs in these areas will need to be strictly controlled and ventilation in these less accessible areas altered by installing curtains or air movers for specific tasks.

Modelling of gas and dust problems and the formulation of mitigation strategies have been performed for numerous examples of similar face ventilation, dust and goaf gas issues on current faces with successful results. Computational Fluid Dynamics (CFD) modelling could be applied to the LTCC method as a technique for identifying airflow problems around the shearer, along the rear conveyor, for determining oxygen concentrations in the goaf and designing control strategies.

TABLE 5 - Caving index with corresponding recovery percentages and description of conditions

LTCC Classification	1	2	3	4	5
Minning conditions	Very good	Good	Medium	Bad	Very bad
Caving index	> 0.9	0.8 - 0.9	0.7 - 0.8	0.6 - 0.7	< 0.6
Seam recovery (%)	> 80	65 - 80	50 - 65	30 - 50	< 30

Equipment design and performance

The design of LTCC equipment and the measurement of its performance is another area critical to the success of LTCC. During the last 20 years Chinese equipment manufacturers have developed several generations of supports as more was learnt about the caving process and the role played by LTCC supports. Chinese LTCC mines typically employ four-legged supports with either rigid or articulated canopies. The supports are typically rated from around 600 to 800 t. Very few mines employ powered supports at the gate ends preferring manually set props and link bars. For an Australian application to proceed, a workable solution involving use of powered support in these areas must be reached. Access to face areas for maintenance functions must also be considered carefully in the design process, as the gate end areas are congested with the additional rear AFC drives. Designing equipment to operate in various ground conditions especially in the maingate end of the face will be important to determine the optimal support design for safe production. For example, DBT in conjunction with Yancoal Australia's Austar Mine are developing an innovative roof support design to control the roof at the maingate end of the face. Time will tell how successful this solution is in soft coal.

Irrespective of the equipment design, initially, an equipment manufacturer and the mine operator must consider the design life of the equipment for the LTCC application. This particularly applies to the shields. A short design life will allow for new equipment designs to be introduced more regularly according to their performance in the field and give the opportunity to take advantage of new technologies. Equipment with a long design life will be more expensive to purchase and will limit the frequency of new equipment designs and technology being introduced. The path chosen will depend largely on the mine operator's confidence in the assessment of the caving conditions and equipment performance prior to mining.

Automation of face functions for increased safety and productivity will be an essential step in the application of LTCC into Australian mines. As with all longwall faces the general guidelines for an efficient face are maintaining correct face alignment and horizon control. With LTCC, these rules will become more important due to the subtleties of the top coal fracturing and caving process. Poor face alignment can seriously impact on production from caving as can a loss of horizon. Control of these parameters through automation will ensure greater continuity of production

The application of automation to the caving process would also remove a major hurdle to introducing the LTCC method into Australia. The caving operation currently requires an operator to spend time at each support during caving operations listening and looking for changes in the caving material as it makes its way onto the rear AFC. Automation of the caving process will involve programming the timing of the retraction and extension of the rear cantilevers. The key step in automation is to replace subtle human observations by developing artificial sensors and neural networks to detect changes in the caving process and to develop a productive automated caving control system.

In summary, the challenges outlined in this section are by no means a complete list but are based on numerous findings and field observations of LTCC. In order to introduce LTCC successfully into the Australian mining industry several challenges exist,from the understanding and application of fundamental science through to solving everyday engineering issues. The challenging areas of geology and geotechnical assessment, mine environment and equipment design along with the suggested methods of meeting these challenges must be addressed to enable the process to move forward. By doing so the issues can be dealt with and in many cases the solutions derived for the LTCC case can be applied back to provide improvements in safety and productivity to current conventional single pass longwall faces.

STATUS OF RESEARCH AND DEVELOPMENT INTO LONGWALL AUTOMATION

A Landmark Longwall Automation Project was carried out by CSIRO and The Cooperative Research Centre for Mining Technology and Equipment (now CRC Mining) under the direction of the Longwall Automation Steering Committee (LASC), a sub-committee of the ACARP Underground Research Committee. The overall goal set by LASC was to develop systems that would result in:

'A longwall face that will operate automatically within pre-defined parameters to enhance health and safety and production consistency, to lower operating costs and improve return on capital'.

To achieve the project goal of longwall automation, ten outcomes or work areas were identified, and the research program structured around them. These areas were:

1. Face alignment focusing on lateral direction control and face geometry.
2. Horizon control looking at maintaining cutting horizons as required by the mining process.
3. Open communications by developing open architecture between system hardware and software components.
4. OEM involvement/commitment by creating mechanisms for exchange of information between the project team and equipment manufacturers.
5. An information system consisting of a monitoring station, automatic sequence design and operator displays.
6. Production consistency and reliability focusing on condition monitoring and reliability, and then on optimizing coal flow and finally on collision avoidance between components.

7. Determining the redefined functions of face operators and subsequent training requirements.
8. An implementation plan for introducing automation system components at strategic selected sites.
9. A commercialisation plan for transferring the proven automation technology to the industry.
10. A progressive automation implementation plan for all longwall mines in Australia by identifying the status of their existing technical specifications and the potential to implement appropriate levels of automation.

A brief description of the outcomes of the first seven of these work areas is contained in Appendix 1.

In general, the major contributions from the three-year $4.31M ACARP Landmark Longwall Automation Project to longwall mining operations have been:

1. New sensor development for closed loop control of face equipment.
2. Integrated operation of face components through open communications systems.
3. New data flow and management methods and technologies.
4. Identification of skills and qualities of people required for automated longwall operations.
5. Development of new on-line condition monitoring and fault detection technologies.

In addition, other successful technical results from the automation of components included:

1. Automatic face alignment was achieved using an inertial navigation-based sensor on the shearer to accurately measure face geometry and feedback signals used to move the roof supports.
2. On-line measurement of creep implemented with creep information incorporated into face alignment corrections.
3. Bench-testing of the Inertial Navigation System (INS) based enhanced horizon control.
4. Broadband communications system using wireless Ethernet to the shearer was robust and reliable on an operational basis. A full commercial installation can now proceed.
5. New results were obtained for locating coal face features with the use of thermal infrared-based horizon control.
6. A longwall information system which integrates information from multiple systems and sensors and provides high quality visualisation and control interfaces was developed.

In the reliability and condition monitoring component of the research program, it was confirmed that longwall technology does not have the same maturity as equipment in other parts of the mining industry, eg surface mining or preparation plants. The typical utilisation of a longwall face is less than 50 per cent across the industry. Half of the delays can be attributed to time associated with face equipment failures. Initiatives in the Landmark Project outcomes recommend this issue be addressed through engineering and design changes and through better equipment selection tools. This includes focusing on increased precision, automation and development of on-line condition monitoring and fault detection techniques.

Following these significant outcomes, a recommendation has been accepted by the Australian Coal Research Board to commit an additional $2.4 million to progress these outcomes into a development and demonstration phase lasting two years at Beltana and Broadmeadow Mines. This research has the following goals:

1. Develop proof of concept outcomes in the Landmark Project to commercial prototype stage.
2. Progress new findings and concepts from the Landmark project to experimental or proof-of-concept level.

LONGWALL IMPROVEMENT OPTIONS

A significant cost driver for longwall operations is panel width or more specifically the development driveage required per longwall tonne produced. Apart from the cost of roadway development, every gate road required has costs associated with items such as the maingate conveyor, secondary support, roadway maintenance, ventilation structures, stone dusting, statutory inspections, pumping, pipes and cables. Reducing the number and frequency of gate roads will directly reduce the cost to prepare a longwall panel for production. The counter side of this argument is the additional capital investment required to span the increased width of the longwall panel.

It should be noted that longwall faces wider than 300 m have been operating successfully for a number of years in Europe and the USA. The trend to introduce face lengths greater than 300 m is increasing as engineering issues associated with increased power requirements for the AFC, higher chain strengths, control and monitoring of starting and running multi high-voltage motors, other incremental improvements in critical areas around the AFC, roof supports and shearer are proven and carefully implemented

Whatever the reason for the opportunity, the option to widen the face has a multitude of benefits to the mine operator. The circumstances surrounding each opportunity will most likely be different for each mine. However, in the end there should be a clear economic justification to increase the face length rather than just focus on tactical or strategic reasons.

Clearly, the easiest justification to widen a longwall face revolves around the additional value added by increasing the annual sales volume to the marketplace. As the revenue stream for a mine is normally its most sensitive financial key performance indicator, additional value can be achieved through increasing annual production levels. This assumes there is no deterioration in the supply and demand balance and consequently the price received, for the additional coal from a wider longwall face.

The Australian underground industry has a long history of poor performance in achieving panel development schedules that meet longwall production continuity requirements. All interested parties understand the risks posed by either slow development or the increased development requirement of narrower longwall panels and are aware that, widening the longwall face will reduce the number and frequency of gate roads required and clearly lower the production, marketing and financial risks.

ACARP commissioned a Scoping Study across most longwall mines in Queensland and New South Wales to highlight the increasing concern that initiatives implemented in the industry, to increase panel development rates in the last five to ten years, have been quite ineffective. The report (Gibson and Associates, 2005) highlights the challenges and problems, that need to be addressed over the next five to seven years. The report also assists people to further asses the benefits of increasing the width of their longwall panels as another means of proactively addressing this problem.

Key business risks

As part of assessing the benefits of extending the width of the longwall face, the key business risks for this initiative should be identified. Some of the incremental risks that may need closer assessment as part of a risk management process include issues like:

1. inability to steer the face,
2. periodic weighting on the longwall face,
3. equipment not being delivered on time,
4. unknown impact on seams above and below the target seam,
5. higher gas emissions,
6. inability to effectively ventilate the panel,
7. the impact of increasing supply into the marketplace,
8. wider faces causing unacceptable higher amounts of surface subsidence, and
9. would current ‘licence to operate’ or other statutory approvals be jeopardised.

While there are additional risks, associated with wider faces, there are significant advantages including:

1. reduced roadway development,
2. less exposure of personnel to health and safety risks,
3. increased production efficiency, and
4. lower operating costs.

Benefits

Where no production increase is sought, the prime basis for justification of wider faces is to gain savings in operating costs as a result of fewer gate roads and reduced longwall retreat metres. This initiative will reduce operating costs in the areas of panel establishment, gateroad development, conveyor drive installations, demobilisation requirements and maintenance support activities. This strategy will enhance the mine’s ability to move operations down the cost curve whilst being able to continue to supply coal at the required sales level.

As part of a project initiative in early 2005, an internal study was carried out at Broadmeadow Mine on the merits, issues and benefits of increasing the face length from 200 to 322 m. As part of this assessment, a review of productivity data, collected by the Joint Coal Board and Coal Services Pty Ltd, was made indicating that longwall face width does have an impact on productivity. It is important to recognize that analysis of this data is limited by the wide range of longwall configurations that make up the data set and the differences in coal clearance capacity between the longwall faces.

In order to gain meaningful comparison a Key Drivers Index was created that combined the main drivers of longwall productivity into a single number in an attempt to make the information more relevant. The drivers used were face width, cut height, system capacity and depth of cover. When this value was plotted against average longwall tonne per annum for Australia’s leading longwall mines, there was a very good correlation evident as indicated by Figure 9.

In Figure 9, Broadmeadow has been plotted on the same correlation to indicate the impact of a wider face on average production. In addition, Beltana and San Juan Mines have been plotted for 2004 (there is only one full year of production data available as the 2005 data is not available yet) to indicate the potential advantage offered through a punch layout system and/or automation both of which will be in place at Broadmeadow. Consequently, Broadmeadow productivity is seen as having a significant upside to that available from conventional longwall operations.

It is noted that Kestrel is an outlier from the general industry trend. However, it is understood that the mine utilises its two sets of longwall equipment such that there is effectively no time lost due to longwall panel relocation or commissioning, as this is undertaken by a contractor whilst the preceding panel is mined. The contractor commissions the longwall over the first 50 m of the panel before handing over the longwall to Kestrel personnel. Without the Kestrel data point the coefficient of determination (R2 value) is 0.94.

It is important to note that the data presented above is essentially a snapshot based on average depth of cover for each operation over a period of up to five years and the actual position along the chart changes as depth of cover changes. Increasing the longwall production

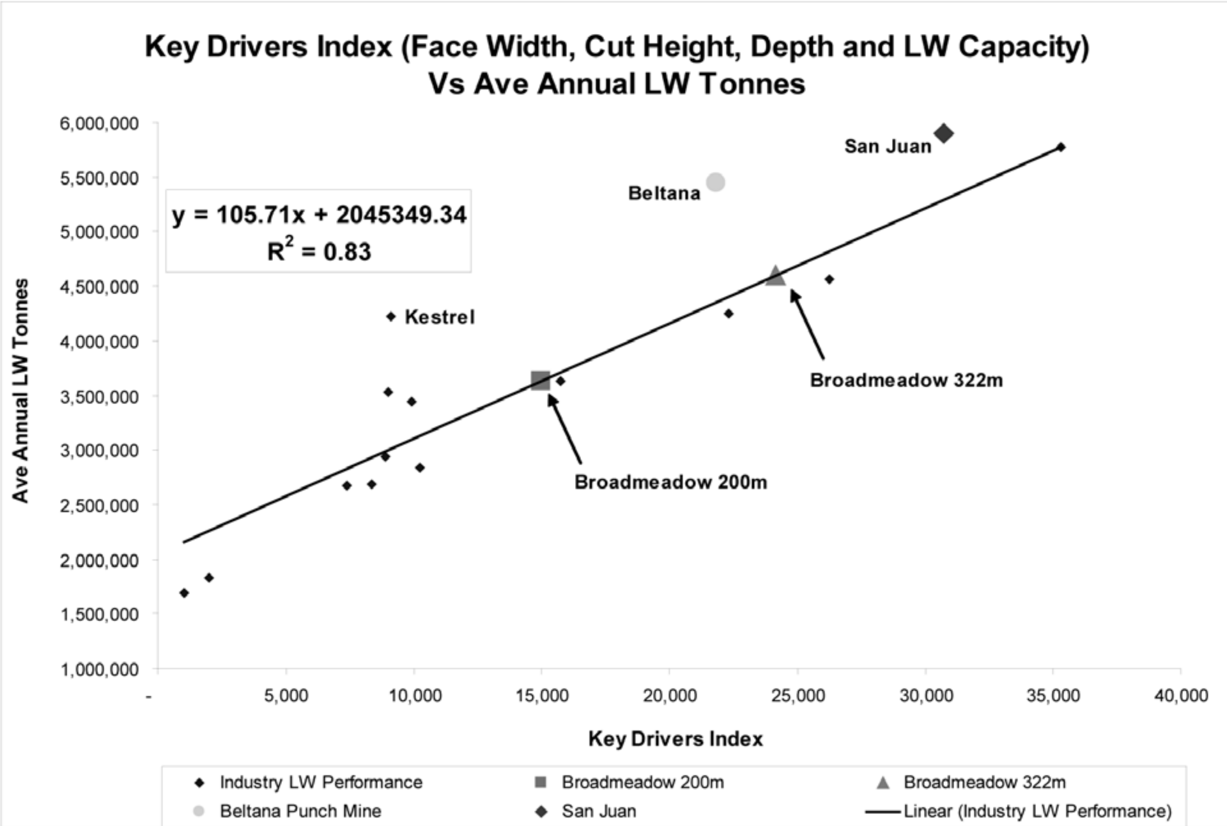

FIG 9 - Historical industry longwall performance

capacity to match the Punch Longwall Mine design and equipment capability of the wider 322 m face has a significant positive impact on the average annual tonnes that Broadmeadow is capable of in comparison to the rest of the industry. The Broadmeadow points have been plotted at a depth of cover of 180 m.

Average run of mine (ROM) production (including development) from Australia's best performing longwall mines over the past seven years is shown in Figure 10. The data excludes the first year after start-up, extended strike periods and the final year of longwall production (ie not full year production).

ROM production from Broadmeadow is forecast at the upper end of this range. However, the logic is that the punch mining approach has significant advantages over conventional mine layouts. Beltana Mine, operated by Xstrata, uses a punch longwall layout and is the site for the CSIRO Landmark Longwall Automation Project. The mine produced 5.7 Mt in 2004 with production curtailed for five weeks due to restricted railing capacity.

Details of the coal clearance systems installed at each of the mines covered in Figure 10 is shown in Table 6.

In general terms, it can be seen from Figure 10 and Table 6 that the potential for lower operating costs are achieved by the mines with large mine production and with larger coal clearance systems designed to meet the outputs of high capacity longwall equipment.

TABLE 6 - Coal clearance capacities

Mine	Coal Clearance System (t/h)
Okay North	6500
Moranbah North	6000
Newlands	5500
Kestrel	5000
South Bulga	4000
Crinum	3500
Oaky Creek No1	3200
Wambo	3000
Ulan	2800
Southern	2500

The cost reduction identified in the Broadmeadow Study means that based on the same sales profile for the two different face widths, there is no need to assess the likely productivity improvements as a result of widening the longwall face. Suffice to say, that the longer travel distance across the wider 322 m face and the reduced percentage time lost at the gate ends to mine the equivalent amount of coal will increase the degree of latent production capacity above a 200 m wide face.

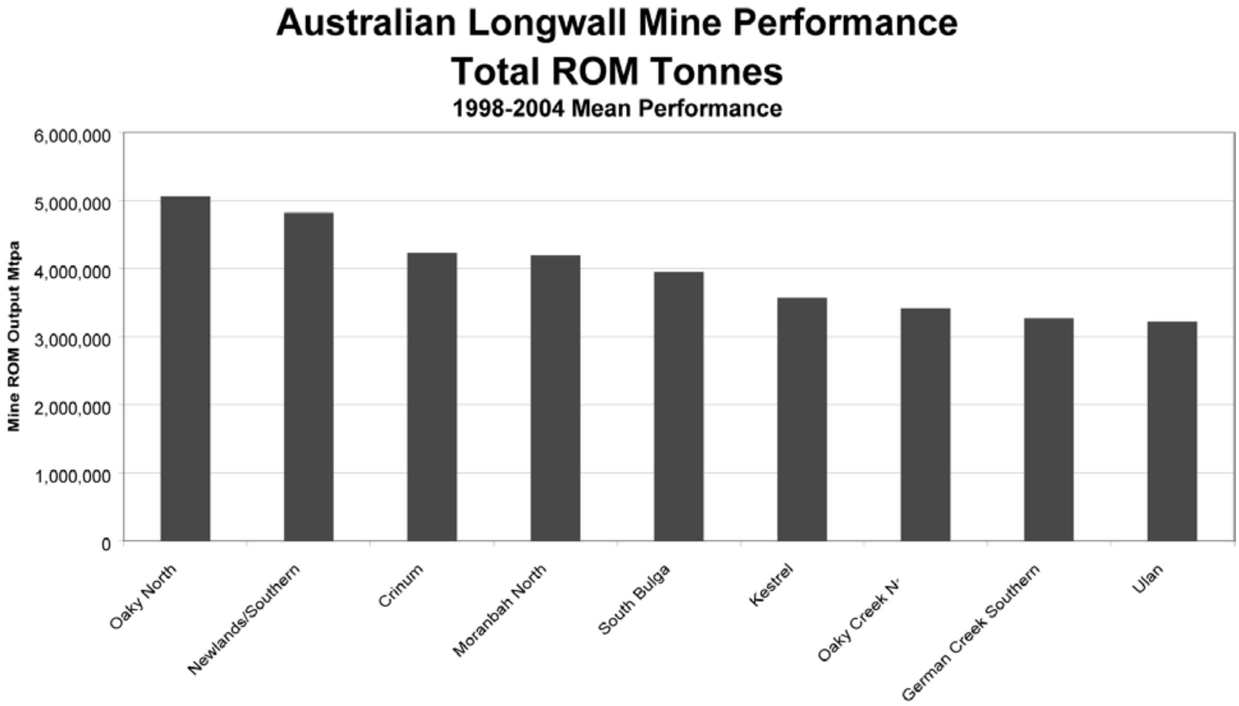

FIG 10 - Annual production from top quartile of Australian longwalls

The underlying assumptions were based on the life of mine plan and the ability of the 322 m face to more easily meet this productivity level than the 200 m face and with potentially more production time available, if required during the financial year.

A counter argument to this line of reasoning arises if the reduced retreat rate were considered to cause a higher risk of geotechnical instability on the longwall face and manifest itself as a reduced availability of the longwall unit. The latent capacity as a result of the more efficient cutting cycle would offset this reduced availability to some degree. For example, an investigation recently completed for Broadmeadow Mine showed that extending the longwall face by 60 per cent to 322 m while maintaining the same production profile resulted in a 60 per cent improvement to the net present value over 30 years. Roadway development was reduced on average by approximately 3500 m per annum, reducing the risk of longwall discontinuity. In addition, there are 15 fewer longwall moves over 30 years and the time between each move is increased by 5.5 months. There is incremental capital expenditure required to extend the face but it is clearly offset by lower operating costs over the evaluation period.

Tables 7 and 8 show the key differences in development requirements and longwall continuity. Table 7 contains information based on a life of mine comparison while Table 8 breaks this down into approximate annual averages for each case.

For Broadmeadow Mine and no doubt others, that decide to increase their face width, there is an opportunity to re-visit the issue of owner-operated development units compared to using a panel development contractor now that there is a reduced dependence on annual development requirements.

Implementing an option to increase the width of a longwall face results in improvements and benefits that can be summarised by reduced development requirements each year and a direct consequential reduction in the health and safety risk to personnel through the elimination of exposure to all activities associated with panel development. Increased elimination of risk per unit of time is the best means of improving safety at any operation.

The economic benefits to this example can be summarised as a reduction in operating costs and business risk. If an embedded option exists to increase production levels following this wider face length, then the value proposition is significantly improved in net present value terms. However, this option may require additional capital to upgrade the coal clearance and other infrastructure to match the design capacity of the longwall.

TABLE 7 - Vital statistics of life of mine layouts

	200 m Case	322 Case	Difference
Number of gate roads	50	35	-15
Number of longwall panels	48	33	-15
Development metres	493 012	358 783	-134 229
Longwall metres retreat	148 257	99 826	-48 431
Longwall cycles	185 321	124 783	-60 538
Longwall tonnes (Mt)	163.0	172.5	9.5
Total tonnes (Mt)	174.3	180.7	6.4
Lw rom tonnes/dev metre	331	481	150

TABLE 8 - Annualised vital statistics of life of mine layouts

	200 m Case	322 Case	Difference
Development metres per year	11 879	8,328	-3,551
Longwall metres retreat per year	3,414	2,189	-1,225
Longwall cycles per year	4,268	2,736	-1,531
Longwall tonnes (Mtpa)	4,459	4,557	0.098
Total tonnes (Mtpa)	4,782	4,783	0.001
Ave time between lw moves (mths)*	10.9	16.4	5.5
*LW move time for 322 m face is 42 days (27 days for 200 m face) – days lost to LW moves over 30 years are basically the same			

OPERATING, DESIGN, METHODS AND PLANNING ISSUES

Manning

Changes in work practices, roster systems and automation have allowed a gradual reduction of manning to operate and maintain the longwall mining system. Shearer initiation systems are universally available for use on all longwall installations. However, across the industry, these and other technological improvements have not been fully utilised to achieve their maximum output capacity, to lower manning levels or to deliver potential productivity improvements.

Ventilation and dust control

Ventilation is required in longwall mining, as in any type of underground mining, to produce an environment suitable for employees to work in comfort and safety. The ventilating airflow must be able to remove harmful airborne dust from the working area, dilute any seam gases that may cause a hazard and provide air of sufficient quality. The specific requirements are addressed in the respective Queensland and New South Wales Acts and Regulations and also within the Ventilation Management Plan developed by each mine.

Ventilation within the longwall panel can be provided by a number of systems and ventilation patterns.

1. The R type system involves a set of intake and return headings either side of the longwall block. Such a system must be used when no main return headings are located at the inbye or extracted end of the panel. Roadways are usually left standing around the extracted part of the block to provide some minimal bleeder ventilation around the goaf edge. This system can only be used in seams with very low gas emission rates as the goaf is not well ventilated. Goaf gas emissions are returned via the tailgate area of the longwall. Typical quantities used in these panels are > 20 m^3/s.

2. A variation of the R type system is used in seams with gas contents of 5 to 10 m^3 per tonne, moderate gas emission rates and lower permeability. The longwall face ventilation still follows the R type path. However, extra bleeder return headings are maintained behind the longwall goaf to provide positively ventilated migration paths for goaf gases. The bleed returns from the goaf are diluted with 20 to 30 m^3/s of intake air from the panel intakes, depending on gas emission rates. The longwall face is ventilated with > 25 m^3/s of air of which a small portion flows past the goaf from the tailgate to ensure all the goaf gases migrate to the bleed returns and do not encroach on the tailgate.
3. The Z or Y system is usually used in seams with high gas emission rates of >10 m^3/t and higher permeability levels. The main returns and the main intakes are arranged at opposite ends of the longwall block with the main returns being behind the longwall goaf. This pattern provides the greatest pressure differential across the goaf and gives the most flexibility for the use of intake air to dilute the gas emissions. The pattern has been described as a Z or Y system, depending on the direction of the intake airflow in the tailgate headings. The maingate is provided with >25 to 30 m^3/s of air with further dilution of the return air by a regulated intake flow down the tailgate roadway.

These are the three main systems used in longwall ventilation but are by no means the only possible combination of intake and return airways.

Panel design, methods and operation

Rapidly increasing production rates achieved by modern longwall faces have put a great deal of pressure on the development rates realised by conventional panel development systems.

The development of gateroads and companion roadways simply delineates the block to be extracted. Unless the mine has the luxury of a substantial lead time on development, has good mining conditions and the development activities are profitable then the gateroad configuration should be as simple as possible. However, there may also be a number of physical conditions at the mine that warrant a totally different system to that which the longwall itself requires.

Roadways developed for gateroad use must be stable over periods of 18 to 24 months and capable of withstanding loadings imposed immediately during development, secondarily while standing during completion of development, and finally by way of the front abutment from the longwall operation. Companion roadways are similarly subject to these loads as well as side abutments and the back abutment. Support installed during development must be capable of withstanding all such loads because premature failure, particularly in the maingate, creates lengthy delays while the face-end moves through. Ground support in the maingate is limited by the space required for the movement of the maingate facilities such as pumps, cables, transformers, monorails and the BSL. Intersections are the main source of deterioration in most longwall gateroads, by virtue of the area exposed to the influence of the front abutment pressure.

Most mines have little or no lead time on development. Gateroad completion must be consistent with the transfer and start up of the longwall face with sufficient time to prepare roadways for longwall use. Multiple entry driveages are a liability with respect to advance rates, particularly if one or more of those roadways are only required during development and serve little or no useful purpose to the longwall. Cutthroughs are likewise an expensive delay during driveage.

The overall layout of the maingate, tailgate and companion roadways can very much determine the success or otherwise of the longwall face. Maingate panels adjoining previous goaf areas are subject to intense abutment loads usually resulting in roof deterioration and floor heave. The rehabilitation of prepared maingates is an expensive and time consuming process. Where goafside maingates develop stability problems due to the proximity of the previous goaf, the options are to increase the pillar size between the gateroad and the goaf or drive the maingate independent of the companion roadways and as late as possible to reduce the time the roadway stands. Driven at narrow centres, 18 to 20 m alongside a companion roadway, this fresh maingate can be subject to stress relieved conditions. Enlarging the pillar width between the previous goaf and the maingate has the disadvantage of increasing the amount of coal sterilised within the longwall area. Where conditions become unacceptable the only other option is to place the maingate on the virgin side and take whatever steps are necessary to stabilise the tailgate alongside the previous goaf area. Secondary support in the tailgate is far easier than in the maingate and this configuration is the most widely practised.

Retreat longwall mining is common to all Australian installations. The final design of the operation is dependent upon the geological conditions and the economic and environmental constraints imposed. The most significant factors to ensure the success of the operation are the design of the longwall panel configuration and the selection of the roof supports to be used. In practice, almost every conceivable combination of gateroad placement and configuration has been tried in the operating and planned longwall faces in Australia.

The design of the ideal longwall panel configuration is achieved by considering:

1. the overall stress environment to which the operation will be subjected,
2. equipment limitations and capital cost,
3. geological constraints, and
4. lease constraints.

Extraction thickness is generally set by considerations other than maximising the coal volume within a longwall block. If a portion only of the seam is extracted, the reason may be on account of roof or floor support problems or for coal quality considerations. Further, the inability of a longwall panel to traverse major faults or extensive dykes can impose constrains on longwall block dimensions.

The length of the longwall block is very often determined by coal lease boundaries or lease conditions which stipulate the leaving of protective coal barriers or pillars beneath certain surface features such as dams or weirs.

With regard to the stress environment, the criteria for chain pillar design and panel dimensions have the greatest influence. Chain pillars are required to protect the gate roads from abutment pressures created by adjacent panels. The design of these pillars is critical for floor stability, and roof stability. Poor design can cause problems resulting in adverse roof and rib conditions and additional surface subsidence. The two types of chain pillars utilised are yielding, or narrow, pillars and rigid pillars. The yielding pillar is designed to yield and fail after it is no longer required.

The adverse effects of yielding pillars were described by Hebblewhite (1983) and can be summarised as:

1. detrimental front and side abutment effects on roof conditions in the gate roads thus necessitating expensive roof supports, and
2. possible premature failure of the pillar caused by excessive rib spall.

However, the benefits of using yielding pillars are:

1. maximum recovery of coal resources,
2. improved face conditions, and
3. more even surface subsidence as caving is able to extend across the adjacent goafs.

Utilising wider chain pillars will provide gateroads with greater protection from the effects of adjacent goafs. However, linear panel development advance rates will be slower, overall coal recovery will be reduced and the possibility exists of isolated goaf areas which could inhibit caving and create heavy mining conditions elsewhere.

The width and length of longwall pillars have significant influences on stress abutments, goaf formation, support requirements, surface subsidence and other factors. The face length is required to be sufficient to allow full caving, bulking and reconsolidation of the overburden strata. The goaf must be able to support the super-incumbent loads so that excessively large stress abutments will not form ahead of or on the longwall face. The selection of these two factors is based mainly upon experience at the prevailing mine and/or at other locations with similar conditions.

Designers may now utilise computational modelling methods to simulate the behaviour of rock masses under various extraction geometries. There is a large range of methods which provide a numerical modelling capability with the most useful being finite difference, discrete element and finite element type, codes. These programs are used to model the immediate roof and floor strata with its bedding planes, joints, and the stress regime. They can also analyse prospective roof and rib support requirements to understand the effect on the strata stability and assess the level of deformation. Successful computer modelling requires a very sound knowledge of the strata properties, strata deformation mechanisms and in situ stresses in the area to be defined. The modelling approach selected must closely simulate those conditions. Field monitoring to verify the results obtained from modelling is necessary prior to application in the mine design.

Panel lengths have steadily increased over the years with various mines having successfully used booster or tripper drives to extend panel conveyors. With improvements in equipment designs and the ever growing need to minimise the development metres per tonne of longwall coal, panel lengths are no longer being limited to match replacement or overhaul of equipment.

Longwall recovery and installation

The installation and recovery phases constitute a major factor in the overall efficiency of a longwall operation. Generally they are a time-consuming and labour intensive operation, during which the colliery is subjected to a substantial drop in coal production and an excessive burden on transport systems. Considerable effort continues to be put into both improving the efficiency and shortening the time of longwall changeovers. While the methods employed by the various operators differ in their applications they are all subject to the same basic principles, considerations and constraints.

To ensure a longwall changeover is completed as quickly and efficiently as possible, considerable pre-work is required in the fields of planning, project management, equipment design, operational organisation and site preparation.

The initial specifications for the longwall equipment should stipulate that the design will facilitate efficient changeovers. The main factors to consider in selection from this viewpoint are:

1. the extent of dismantling and re-assembly necessary and the ease with which it can be accomplished,
2. the relative ease of face-to-face transport, and
3. that the changeover time is the optimum time for major maintenance to be carried out on longwall equipment.

A further consideration is the possibility of duplication of various pieces of equipment. Thus a second component may be installed in the new face site prior to the other unit being recovered. An AFC and/or a shearer or component exchange of machinery parts, shearer gearboxes and haulage units, can save time in machinery overhauls.

The actual changeover operation should be modelled on the results of a thorough assessment of previous changeover techniques, both locally and overseas. Account must then be taken of the resources the company is willing to make available in terms of both capital and manpower. With the above factors in mind, an overall philosophy for the longwall move can be evolved.

One person should be made responsible for the overall project coordination of the longwall move with the various mining, mechanical and electrical departments. In conjunction with mine management, the coordinator must ascertain the extent of overhauls or repairs, supplies, handling and transport equipment that will be required to support the overall move philosophy.

The next phase is to provide a proposed schedule of all operations involved in the changeover. This involves compiling a detailed and comprehensive inventory of all tasks to be performed, including an estimate of time to be taken as well as any sequence relationships between various tasks. This involves preparing a chart with the tasks commonly listed down the page and with the estimated time requirement for each task scaled across the page. This method allows a graphic indication of project progress and status as regards the program, but provides little opportunity for consideration of alternative plans.

The application of the critical path method (CPM) to the longwall move is a well known and used practice that applies network analysis techniques to evaluate various alternatives. These systems readily lend themselves to computer applications, which enable management to explore numerous variations of techniques, cost and resource allocation in order to plan for optimum moves in terms of time and efficiency. With PCs loaded with software, like Primavera P3 or Microsoft Project, the task of planning and managing a changeout is much easier. Progress can be readily updated against the plan during the move.

The final planning phase involves locating and determining the procedure for terminating the longwall face. Two major factors need to be considered in planning for this phase:

1. ensuring that sufficient room is available for the removal of the longwall equipment, and
2. ensuring the availability of the roof and rib support necessary to maintain a secure workplace.

The critical width of the recovery area is the clearance from the roof support base to the completed face line rib. This clearance must not only allow passage of a support lengthwise down the face, but also allow for the turning of a support from its in-line position to one of forward and parallel to the face. Increasing use of diesel powered mobile equipment can impose minimum width requirements particularly at gate end entries onto the face.

Part of the planning process has to include the preparation of the roof control plan at the take off point and should aim to:

1. minimise time delay and cost,
2. maintain the integrity of the roof after roof support removal,
3. ensure sufficient ventilation exists along the face for the duration of the recovery, and
4. prevent roof deterioration in front of the roof supports.

Some additional roof support is generally necessary for a minimum of 5 to 10 m from the finish position. The degree and type will depend upon the prevailing conditions and may consist of roof bolting, strapping, meshing, timbering or a combination of systems. Roof bolting is generally parallel to the face in rows a multiple of shear widths apart and is most effective when installed near to the newly cut face line. Strapping can also be incorporated with roof bolting to give a better support if flaking is a problem. If the roof is prone to breaking into small pieces, the use of plastic link mesh can provide an effective means of keeping the recovery area clean. The mesh comes in rolls and is progressively fed over the roof support during the final stages of longwall production. The mesh can be reinforced with roof bolts, or straps and its use is common practice with rows of wire ropes.

The sequence of recovery is largely site dependent and influenced by the face to face transport route, respective gate road conditions, component maintenance requirements and availability of replacement or spare equipment.

As a rule the shearer will need some form of surface overhaul and it is advantageous to remove it from the face as soon as possible. This can normally and best be accomplished by removal from the tailgate end since only the tailgate drive section and perhaps several face conveyor pans need to be removed before access can be gained to the shearer. Removal from the maingate will generally require a minimum removal of the maingate drive and at least a portion of the stage loader or, if a cutthrough is not adjacent to the face line, may involve removing a portion of the BSL, boot end, monorail and perhaps part of the pantechnicon, if one is used. Obviously, if the tailgate area is inaccessible no alternative except removal from the maingate end will exist.

Access routes to the recovery area should be maximised to allow recovery work to proceed concurrently in more than one location and so reduce overall recovery time. To illustrate this concept, the worst case may be no access to the tailgate and no cut-throughs adjacent to the face line. The removal sequence in this case is somewhat predetermined:

1. BSL,
2. maingate drive,
3. shearer,
4. AFC,
5. tailgate drive, and
6. supports, starting at the tailgate first.

A more ideal case would be access to both gate roads, thus with concurrent removal of items it is possible to save time, the sequence would be:

1. maingate and tailgate drives and maingate transfer concurrently,
2. shearer via the tailgate and BSL concurrently,
3. AFC, and
4. roof supports.

Conceivably, even more working area could be provided by having prepared stalls driven prior to the longwall completion, enabling recovery to be carried on simultaneously from multiple sites.

The roof conditions in the main and tailgate areas will be a major factor in whether cutthroughs can be supported opposite the face finish position or whether standing support such as tin cans, lock and links or timber chocking will be required. The use of this type of support will preclude access to the tailgate. Consequently, the order for removing the equipment is largely dependent upon site specific conditions.

The AFC, BSL and crusher are withdrawn in units as large as the transport system will handle, thus reducing the amount of dismantling and reassembly required. This saves time and exposure to manual handling tasks.

The shearer is accessible for recovery once the face conveyor drives have been removed from either end. While it is preferable to transport the shearer in one piece, clearance, weight and length limitations usually require that one or both drums are removed. A shearer transporter purpose built exclusively for recovery, transport and installation of the shearer has been very successful. This involves an extension of the shearer chainless haulage system onto a track or trackless mobile machine or flat-top, whereby the shearer can tram, under its own power, directly from the face line onto the transport unit.

The roof support recovery contains these distinctive phases:

1. withdrawal of support from its face position,
2. transport along the face line, and
3. temporary support requirements.

Supports are removed via the tailgate or maingate, depending upon access. Generally, if tailgate access is available, recovery will commence with the maingate end supports.

When withdrawing a roof support from the face line, it has to be lowered, drawn toward the completed face line and turned through 90° for transport under the remaining supports. Various methods are employed by Australian operators, each dependent on the equipment available. Generally, the support is dragged out of line and turned by the use of diesel or electric vehicles. Specialised mobile equipment for support recovery is widely used in Australia and includes the Petitto Mule, an Eimco 936 or heavier LHD and other specialised chock handlers and transporters.

Removal along the face line is either by towing or carrying, again dependent upon equipment and other limitations. When towing or winching is employed, and the floor is not competent, skid paths or means of taking the weight of the toe of the support may be necessary to prevent undue break-up of the floor.

Temporary support of the recovery area during support removal generally consists of props, wooden chocks, trailing or buttress supports or a combination of systems. The aim is to only support enough roof to enable the next support to be removed and ideally to keep the advancing goaf at least one support width from the last remaining support.

Provided sufficient ventilation can flow through the face, there is little value in keeping access through the face line at the expense of time in setting extra support.

Recovered longwall equipment may be destined for transport:

1. directly to the installation area,
2. indirectly to the installation area via the surface or an underground repair area,
3. to a storage area, or
4. a combination of the above.

Similarly, the method of transport may be tracked, trackless or a combination. Each system's merits make it appropriate for particular operations. The transport phase is usually of critical importance on a longwall move and without careful planning lengthy time delays can result.

Generally speaking the ground conditions in the installation area are more easily controlled than those in the recovery area. Installation roads are commonly 6.5 to 9 m wide to provide adequate room for installing the roof supports and the AFC. They are supported with conventional roof bolts, rib bolts and mesh. Usually secondary support consists of mega and/or cable bolts. The ground support is installed as per the design developed by the geo-technical engineer and the strata management plan.

The degree of support necessary in an installation heading is dependent upon the prevailing roof conditions and the time the heading is to remain open prior to longwall production commencing. This support is placed in conjunction with a roof support with an E frame attachment.

In some instances the prevailing ground conditions may not permit the driving of a wider than normal heading. Historically, the driving of a relief heading has proved successful in reducing this problem in some circumstances. This concept relies upon the driving of a cave heading parallel to the proposed installation heading and relatively close to it, at say 18 to 20 m centres. This close distance encourages the cave heading to fail. It is usual for some stress relief to occur and hence to create improved conditions for the adjacent installation heading. Generally speaking, the longer the cave heading is driven prior to the installation heading the better the destressing effect.

In coal seams with moderate to high gas make, some problems may arise during the development of the installation heading. With wider headings and consequently larger cross-sectional areas of roadway, velocity is reduced relative to normal headings and this may lead to methane layering in cavities or at the face.

Access provided by a particular layout will largely dictate the installation sequence. As with the recovery area, the layout should allow maximum access so that provided site conditions permit installation work can proceed concurrently in multiple locations. The ideal layout should allow access to both ends of the installation heading and also allow for build-up and installation of the maingate equipment without hindering other operations. Cut-throughs driven from a heading behind and parallel to the installation heading can give access at various points along the face and may be important if supports need to be installed in a particular sequence which may not necessarily match their recovery sequence.

Another factor that can affect the sequence of installation is the provision of spare equipment, which can be installed prior to longwall recovery commencing. Obviously, longwall changeover times will be reduced if, for example, a spare AFC is installed prior to commencement of the move.

Barriers to the provision of an ideal layout may include:

1. limited available development time prior to changeover commencing,
2. access to the face being restricted by a relief or cave heading driven behind the face for stress relief, and
3. roof conditions not allowing the forming of small pillars used to give multiple access points.

The options discussed above represent a sample of the various techniques used, and attempt to highlight significant variations used by different operators to cover particular circumstances. It must be remembered that longwall changeover philosophies are constantly evolving and rarely does the operator perform succeeding moves in exactly the same manner. Longwall relocation techniques are constantly improving. They are time consuming, are inherently dangerous due to the handling of many pieces of heavy equipment in confined spaces, their cost is high and the time taken represents a loss of highly profitable production.

ACKNOWLEDGEMENT

The editors wish to thank N Aziz for his review of this chapter in the previous volume.

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APPENDIX I

Brief description of seven outcomes of the study by the Longwall Automation Steering Committee.

Outcome 1: Face Alignment and Creep Control

The project goals were firstly to automatically maintain a designed face alignment by measuring the three-dimensional position in space of the shearer and to use this information to appropriately control the movement of the powered supports. Secondly, to implement creep control by measuring the position of roof support structures in the main gate and use this information to adjust the face geometry.

During the project, a method of longwall retreat measurement was invented and a third goal became to develop a system to directly measure longwall retreat distance.

Achievements in this outcome included:

1. Automatic measurement of shearer position using the Landmark Shearer Position Measurement System (SPMS) on a production basis.
2. Accurate shearer position information is now routinely used at Beltana No 1 Mine.
3. The Honeywell or Litton INS hardware at Beltana has not failed over three longwall blocks.
4. Only one computer failure occurred due to flash memory failure and controls have been implemented. The failure was not due to the operating environment.
5. Supervised operation of closed loop for face alignment has been realised.
6. Automatic measurement of creep distance has been demonstrated and creep correction information has been manually applied to the face at Beltana.
7. A system to automatically measure longwall retreat progress has been prototyped and successfully tested.

Outcome 2: Horizon control

The goal was to achieve automatic horizon control by responding to actual changes in seam profile. For general applicability, the horizon control strategy needed to account for the variability in strata and operating conditions between Australian longwall mines.

Achievements in this outcome included:

1. enhanced horizon control of the vertical shearer track using INS based measurement has been successfully bench tested,
2. optical marker band detection has been successfully demonstrated underground,
3. thermal infrared detection of coal interfaces and detection of marker features in the seam have been shown underground, and
4. a procedure for automatic horizon control for crossing fault conditions has been developed.

Outcome 3: Open Communications

The goal was to develop a communications method to facilitate the flow of information between landmark-developed sensors and systems and critical components that form the longwall system.

Achievements in this outcome included:

1. A standard information interchange protocol between equipment from different suppliers has been developed.
2. A broadband communications link to the longwall shearer to support automated longwall shearer-based instrumentation has been developed. The wireless communications link to the shearer has been one of the major successes of the project. Coverage along the entire 260 metre face length has been very reliable and the link is now an established part of the mine's production system. In addition, power line carrier tests have been promising.
3. The introduction of the EtherNet/IP open system communication standard and associated development of the automated longwall device and control system specifications has been completed. This has been fundamental to the overall success of the project. This has provided a solid platform for ongoing development and acceptance of an open industry-wide (non-OEM specific) standard for the interconnection of mining equipment.

Outcome 4: OEM Involvement

An important outcome was to maintain OEM commitment over a range of activities to the Landmark Project throughout the three years.

Achievements in this outcome were:

1. OEM involvement was maintained at a high level. Since only one major test site was provided for in the project scope, only two OEMs were able to participate actively in the field components of the project work program. However, as all OEMs need to be part of the delivery of longwall automation products to the industry, it was necessary to involve the other manufacturers actively in the project.
2. Initial OEM concerns surrounding the requirement for free flow of information between longwall components from different vendors were allayed with the definition of the EtherNet/IP communications protocol. The work program of the various project outcomes involved all major OEMs and served to keep them abreast of project developments.
3. CSIRO has built strong communication and technical links with the Longwall OEMs in Australia, Great Britain, Germany and the United States. This effort included a facilitation role and resulted in significant input to the project by all major longwall equipment manufacturers.
4. In particular this has been through in-kind contributions in the form of:
    1. modifications to the shearer to accommodate Landmark equipment,
    2. development of Landmark-compliant equipment control software for roof supports and shearers,
    3. provision of personnel to participate in design reviews and risk assessments, and
    4. participation in laboratory and underground testing of Landmark Longwall Automation systems.

Outcome 5.1: Information System

The original goals consisted of developing an underground monitoring station and the implementation of visualisation software for automation system operation in the monitoring station. In addition, a third objective was aimed at achieving software design and maintenance requirements for implementation in the monitoring station for a single user operation and to run with 'off-the-shelf' computer hardware. The information system was also required to report exceptions relating to the performance of face equipment in keeping with the on-face observation concept of the project. High quality 3D visualisation of face equipment and conditions would be essential to give users monitoring the system remotely, confidence that the automation system is operating correctly.

Achievements gained were:

1. a powerful tool set has been constructed for automation and general process management;
2. visualisation systems incorporating database and graphical user interface software that allows multiple users access to tailored information, have been developed and an integrated information system to merge automation, geotechnical, mine design and equipment performance has been put in place; and
3. the initial implementation has clearly shown that relevant forms of information can be made available to different people at different locations at the mine site and appropriate interaction with the system can occur from various locations including on the surface and at the maingate.

Outcome 5.2: Automation Sequence Development – Process Design

The second portion of this outcome set out to develop automation systems for shearer haulage control to enable automation of specific production sequences.

A plan to facilitate the design of operating sequences and transfer them to the shearer control system has been developed.

Outcome 6: Production Consistency and Reliability

This outcome deals with a number of topics concerned with achieving production consistency and reliability in an automated longwall system by investigating the present longwall face equipment reliability and then develop reliability projections for an automated longwall face. The aim was to investigate the failure causes and to propose different maintenance strategies to improve low utilisation of the face equipment in Australian longwall mines.

A significant study of failure causes and appropriate maintenance strategies was carried out. The outcome achieved at both test sites found that equipment related delays were the single major contributor to the total downtime, accounting for over 50 per cent of all lost production time. The largest component of the equipment downtime is attributed to delays associated with face equipment. These delays are caused by either genuine breakdowns or by erroneous condition based alarms.

Outcome 7: Redefined Functions of Face Operators

The goal was to identify personnel characteristics and attributes and then the subsequent skills required in the workforce for operation of a considerably more automated longwall face and develop the framework for a training package to assist in the design of training programs.

Achievements attained included:

1. a study of manning requirements has been performed,
2. key positions and personal attributes have been identified,
3. generic roles have been established and a training matrix has been completed, and
4. a framework for individual training packages has been developed.