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STRATA MANAGEMENT AT THE GOONYELLA EXPLORATION ADIT PROJECT

Ross Seedsman[1], Peter Brisbane[2] and Guy Mitchell[3]

ABSTRACT: The Goonyella Exploration Adit is a three heading development into the highwall at Ramp 4 at Goonyella Riverside Mine that was driven between September 1999 and November 2000. The principal was BHP Billiton Mitsubishi Alliance (BMA) and the contractor was Allied Mining Australia. Strata management began at the pre-tender phase and evolved as greater knowledge of the ground became available.

Roadways were driven 3.6m high, 5m wide in the 7m – 8m thick Middle Goonyella Seam. The cut and flit mining system was used with extended cuts of 12m to 15m and a minimum support pattern of 4 x 1.5m tensioned bolts at 1.5m centres. Extended cuts were possible, even at the maximum depth (290m), so long as an adequate thickness of massive top coal was left. Horizon control was the basic strata management tool in the project.

Extensive exploration using both drilling and 3-D seismic methods was available and proved to be reliable in detecting faults greater than 3m throw. Systems to identify the proximity to thrust faults during driveage were successful in identifying smaller scale thrust faults.

Small-scale normal faults striking parallel to the roadways were encountered, typically in only one of the three headings, and these impacted substantially on the mining rates. They were not detectable from any exploration program. The term trench roof was adopted to describe the associated roof falls – less than 2m wide, up to 1.5m high and bounded by fault or joint planes. The falls were interpreted as being vertical drop-outs from a coal roof under very low horizontal confinement.

TENDER PHASE

In 1998, as the project went to tender, the understanding of the geotechnical regime was based on extensive drilling and 3-D seismics. The following is a summary of the geotechnical understanding as it stood in 1998.

- The mine plan involves the development of three roadways in the lower part of the seam leaving about 2m of top coal.
- The adit may be in a structurally complex area, as evidenced by a rotation of the strike of the seam by 45^{o} and the identification of two large thrust faults.
- There is the possibility of numerous small-scale faults aligned across the direction of the adit.
- Driveage is to be subparallel to a mapped normal fault and one of the joint directions exposed in the pitwall.
- The top coal ply is thickly banded to massive with laboratory unconfined compressive strength (UCS) values greater than 20 MPa.
- A moderately strong stone roof is present above the coal.
- There is the possibility of bedding-parallel shear zones within the coal seam, in addition to those mapped/inferred in the roof and floor.
- The driveage direction is at right angles to the presumed direction of the major principal horizontal stress.
- The major principal horizontal stress in the coal seam is assumed to be lower in magnitude than the vertical stress (0.7 times).

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As one of three geotechnical consultants, Seedsman Geotechnics Pty Ltd (SGPL) prepared a report that identified three ground types (Figure 1) based on an approach used in civil engineering. These are as follows:

- Typical – the usual conditions associated with mining in an unfaulted coal seam with a coal roof - a 'minimum' support rule can be developed prior to mining,
- Adverse – mining through 'known' geological features such as minor faults – support rules can be formulated based on presumed conditions,
- Special – conditions when mining thrust faults or at depth – highly variable conditions that will need to be assessed underground and a support regime developed.

Special conditions were identified beyond chainage 2500m as a result of a combination of thrust zones, higher ground stresses related to the greater depth of cover, proximity of rider seams, higher water inflows and other factors. It was recognised that there could be undetected faults at any depth that would be classified as special ground.

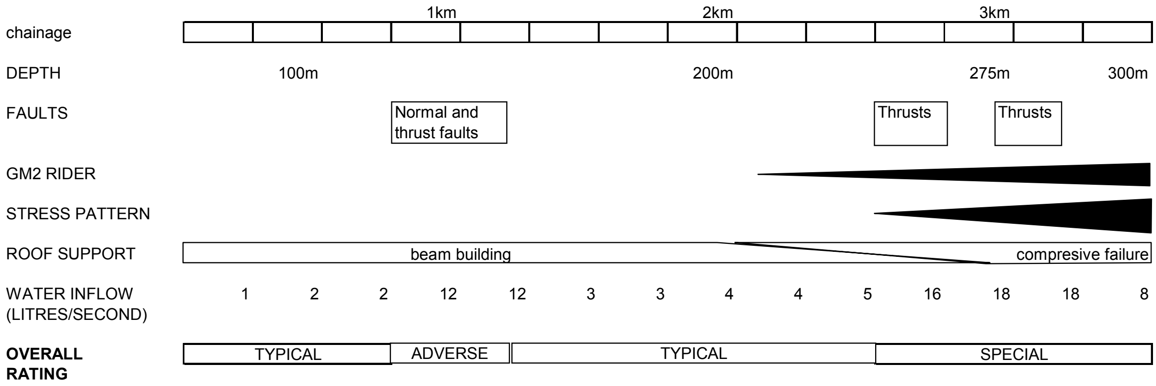

Fig. 1 Zonation of the adit at tender stage

Allied Mining Australia (Allied) was the successful tenderer. SGPL was appointed as the geotechnical engineer to both the principal and the contractor - this strategy would assist in removing the uncertainties of the ground from any contractual issues.

CHANGE TO CUT AND FLIT

Further analyses of the ground conditions, and an inspection of Moranbah North Mine, resulted in Allied and SGPL advocating cut and flit systems for roadway development. At the same time, BMA indicated that the roof and rib support designs did not need to consider the stress abutment associated with longwall retreat. These two decisions allowed major changes in the roof support design, and once these were accepted by BMA, the strata hazard management plan was developed and finalised.

The inspection of Moranbah North confirmed the interpretations of the analysis of the behaviour of coal beams. At Moranbah North, the importance of keeping a thick coal roof of about 1.5m had been recognised, as it was at North Goonyella. The Goonyella Middle Seam is comprised of a number of plies. Ply 1, the upper ply, is described as dull high ash coal with common claystone partings and occasional bedding plane shears. Ply 1 is approximately 1m thick. Ply 2 is approximately 2m thick and is described as predominantly dull coal, with few claystone partings.

The top ply is highly banded and experience shows that it is not self-supporting. Ply 2 is a massive coal band and calculations suggested that so long as 0.4m to 0.5m was left uncut it could span across a 5.5m roadway (Figure 2). If such a unit could be formed then an extended cut could be taken and hence allow cut and flit mining. The 1.5m roof coal criterion at Moranbah North was interpreted to be the composite of the 1m Ply 1 acting as a surcharge on a 0.5m structural beam of massive Ply 2 coal.

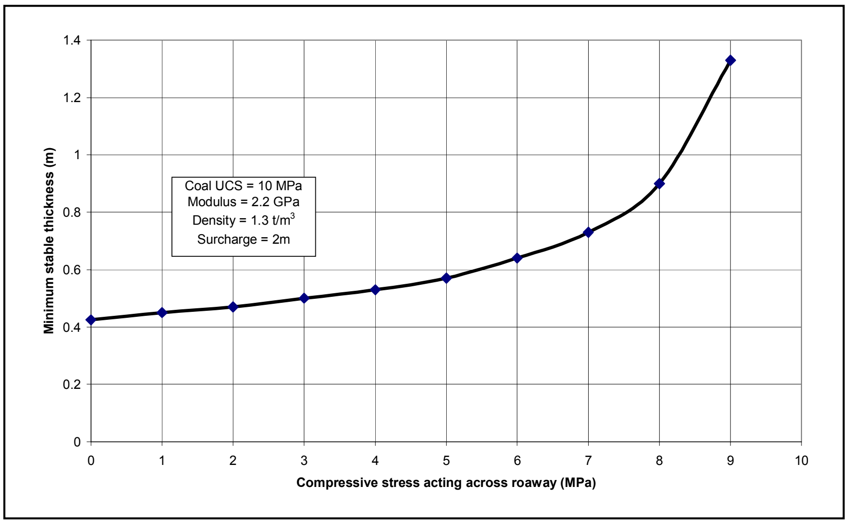

Fig 2 Analysis of a coal beam

STRATA HAZARD MANAGEMENT PLAN

Generically, there are four major geotechnical hazards that all underground operations face:

- Invalid design – the proposed mining method and roof support has not been correctly specified.
- Geological variations – the exploration programs has not revealed the full range of geological conditions.
- Installation problems – the roof and rib support is not or cannot be installed to specification.
- Time – what is stable in the short term is not adequate for the long-term.

The strata hazard management plan and its associated response plans were formulated around these four components.

Invalid design

The requirements for roof and rib support, as set by BMA and Allied were:

- Safety on installation and outbye
- Design life in the range of 5 years
- Optimum development rates

The critical design issue was the long-term stability of the coal beam, and particularly if it was available at depths in excess of those at the adjacent mines. An integral part to the plan was to validate as soon as possible the two critical input parameters – coal strength and horizontal stress.

Testing of the coal in Ply 2 in both the horizontal and vertical directions revealed the importance of incipient cleat in the coal. The design value for unconfined compressive strength of Ply 2 was set at 10 MPa.

Both the vertical and horizontal stress, as measured by overcore methods in the coal roof, were consistently found to be lower than expected. The vertical stress was approximately half that expected on the basis of the depth of cover and the major horizontal stress was approximately 0.45 times the vertical stress. The current model to explain this stress model advocates coal shrinkage as it is dewatered by the mine openings.

It was concluded that the combination of this stress field and the shape of the opening resulted in very low horizontal stresses acting in the immediate coal roof. This provided more confidence in the stability of the plunges and allowed the correct prediction of continued stable plunges in unfaulted ground at the end of the adit at 290m depth.

Variation in geology at the face compared to the design assumptions

Geological uncertainties are inherent in all geotechnical endeavors. This is particularly the case in underground coal mining as the detailed nature of the immediate roof can change significantly and not be identified by the workforce. The installation of roof and rib support requires constant checking that the assumptions on which the support design was based remain valid. This requires that the operators know what to expect and have the knowledge to identify significant deviations from that. A pocket-sized booklet was prepared that described the anticipated conditions and the support standards to be used.

The known possible geological variations that were incorporated in the support designs were closer-spaced bedding in the immediate coal roof and the presence of disrupted ground around thrust faults. The exploration programs had supplied the project with very good information on the coal plies and the presence of the larger thrust faults.

Workings drove into an unidentified thrust fault and into a zone of low throw normal faults both of which resulted in reductions in the planned development rates.

Poor installation

Each bolt plays an important part in the roof and rib support. There was a degree of conservativeness in the support design but this was for the unexpected geological conditions, and not to cover for poor workmanship.

Primary bolt installation was routinely checked and no problems were identified. In common with other operations in Queensland at the same time, it was found that the cuttable rib bolts were being broken on installation due to the high torques that were being used for the roof bolting. Once the problem was identified, a direction was issued that cuttable bolts were not to be used until a torque reduction circuit was installed in the hydraulics of the Fletcher bolter.

Time dependent failure

Roof failure or unacceptable movement outbye is a safety hazard and indicates that the primary support design is invalid. Tell tales were installed throughout in all headings.

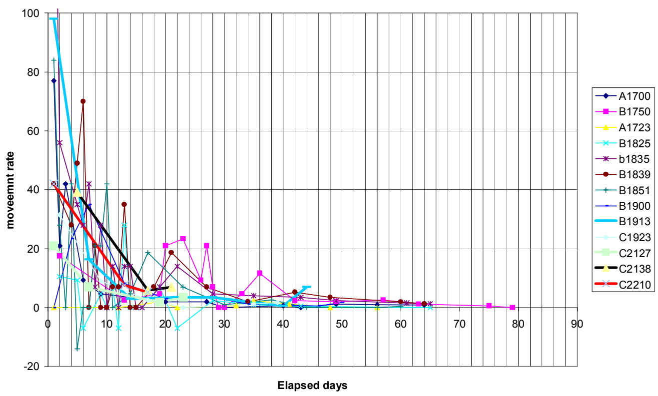

Fig.3 Examples of tell-tale measurements - movements in mm/week

The coal beam analysis gave an allowable movement of 10mm and in unstructured areas this was never exceeded. In the more structured areas, additional movement was recorded and in most cases this reduced to very low rates within ten days (Fig. 3).

In faulted ground, pretensioned cables were used with a design based on dead weight suspension. To achieve design load, it was calculated that the cables would stretch an additional 20mm after installation, so a threshold was set of 10mm for management review, and 20mm for remedial action.

It is noted that there has been one roof fall and numerous rib falls since October 2000. These are still being assessed, but at this stage the interpretation is that they may relate to an increase in vertical stress related to depressurisation of the coal over such a wide area that the overburden is beginning to reload the coal. The mechanisms used to explain slabbing in metalliferous mines are also being reviewed.

Strata reviews

A strata review team was formed that met every three months or more frequently as required. The team consisted of BMA's project superintendent, planning manger, and geologist, Allied's mine manager and the geotechnical engineer. The team inspected the underground roadways, reviewed the geological information, and developed support regimes for special ground. The team then issued an Authority to Mine to Allied, which was valid up to a specified chainage.

Typical ground was defined as that part of the driveage having a massive immediate coal roof. To achieve this, horizon control was essential and it was instilled into the work force. They all carried the simple diagram shown in Fig. 4 which was an idea copied from Moranbah North. Since Ply 2 is about 2m thick and only 0.5m had to be left, normal faults with throws of up to 1.0m to 1.5m could be traversed (so long as the fault plane itself was supported).

STRATA MANAGEMENT IN TYPICAL GROUND

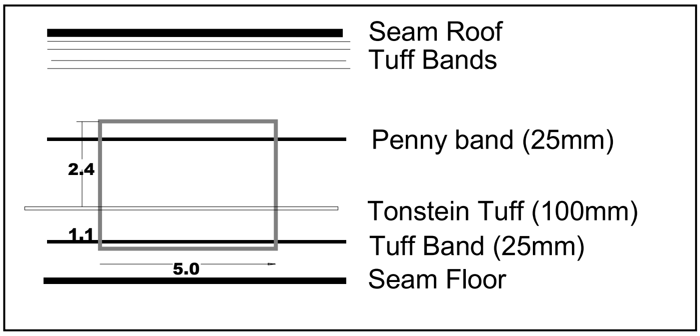

Fig. 4 Horizon control for typical ground

Roof bolting is needed as insurance against unknown structure in the immediate roof and future stress changes. With extended cut mining at Goonyella, the bolting design has a number of different features compared to in-place bolting for longwalls:

- The stability of the cut demonstrates that there is no adverse structure such as bedding or faults.
- All the stress changes that occur in the roof develop within about 5 m of the face and hence there are negligible stress changes after the bolts are installed.
- BMA had directed that longwall abutments did not have to be addressed.

To specify the bolting, a pragmatic design based on preventing time-dependent tensile delamination of the coal was adopted. In addition, the demonstrated stability of the plunge was utilised to move from straps to spot bolts. The support regime was:

- Four 1.5m long X type bolts per ring, evenly spaced across the roadway
- pretensioned to a minimum of ten tonnes
- chemical point anchored
- installed vertically or angled away from the centreline
- 1.5m ring spacing
- 400mm butterfly plates or equivalent
- either closed up to 1m spacing or increased to six bolts/strap in intersections
- maximum unsupported time for roadways – two days

EXPERIENCES IN SPECIAL GROUND

The project team knew of the experiences with faults at North Goonyella, and were fully focused on the possible impact of the known and unknown faults at Goonyella. The 3-D seismics had identified a number of large thrust faults well inbye of the portal (Figure 1) and the strategy was to mine up to them and develop support strategies after some experience had been gained with the seam. The large thrust faults were traversed without problems, because of the lessons learnt from a fall on a minor thrust fault outbye.

The geology that gave greater problems was a series of small scale normal faults.

Thrust faults

The first thrust fault was encountered at 28 cutthrough in C heading. This had a throw of about 0.3m and had not been detected in the seismic exploration or drilling. With hindsight, there was very strong evidence of its presence in a cored hole and, significantly, not in the geophysical logs. The reliance on geophysical logs (and particularly the picking of spikes in sonic logs) to identify slickensided zones was reduced.

Back analyses of the roof conditions as the thrust faults were approached from the footwall (downthrown) side allowed the formulation of a predictive model for the proximity to such faults (Fig. 5). The shallow, flat-topped roof falls began about 20m - 30m prior to the thrust plane being intersected. This model successfully identified all subsequent thrust faults and complemented the surface exploration: 3 D seismic survey were invaluable as a planning tool and was found to be accurate for faults greater than 3m, but less than 50% for smaller throws for which the observational model was essential.

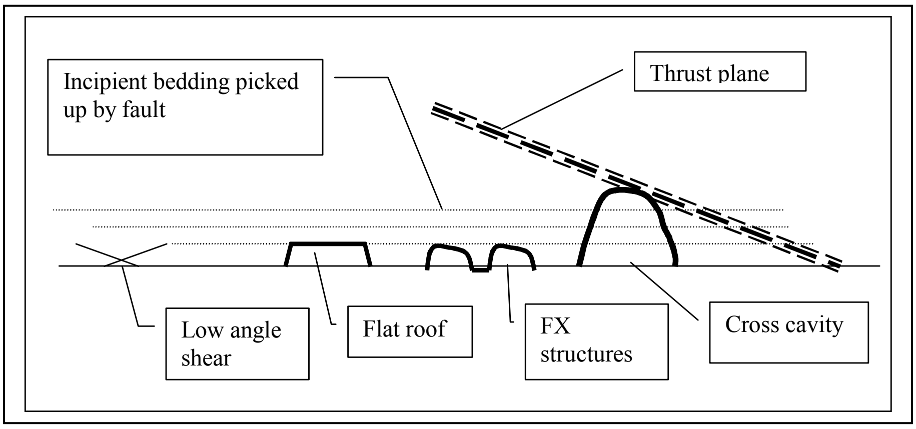

Fig. 5 Roof conditions on the footwall of thrust faults

The roof support pattern was altered once any of these precursors were observed or when thrust faults predicted from the seismic exploration were approached. The major concern was that the development roadway did not intersect the fault plane before long tendon support had been installed. In such circumstances, experience had shown that a large roof fall could develop outbye.

A design approach based on the dead-weight suspension of a fall mass was used. The fall height was estimated from the geology (thrust planes followed coal riders), and the tendon length based on the ability to anchor in low strength strata above the riders.

The sequence that was used to mine through these faults involved:

1. progressively taking 1m cuts with the remote controlled miner and installation of 2.1m roof bolts (using hand-held bolters) until a 5m cutout was formed,
2. flitting of miner,
3. installation of two angled tensioned cables per metre using Fletcher bolter,
4. flitting of bolter.

The sequence resulted in a relatively slow development rate. It was recognised that there were opportunities to increase cut-out distances, and/or to delay changing to this sequence but these were not adopted.

At the request of BMA, the initial cable design was conservative. The design assumptions and functional restraints led to a specification of 150 tonne/m of support for an 8m fall block. The fall height was based on the assumption that the rider seam and its immediate roof could be involved in any fall. Anchorage was available in the 8-10m horizon vertically above the roadway, or with 6m long angled cables anchored in 1m of sandstone or 4m of coal. This specification was later reduced to 63 tonnes/metre, which was implemented with 2 x 50 tonne cables every 1.6m - say 2 every 1.5m. In some cases, a strong sandstone unit was available within the anchorage horizon and this allowed the cable lengths to be reduced.

Horst/Grabens

At around 21 c/t in A heading, and particularly in C heading inbye of 29 c/t, the driveage encountered what became referred to as 'trench roof'. This coincided with the presence of small-scale normal faults in a horst/graben configuration (Fig 6). The headings were sub-parallel to the strike of the horst/grabens.

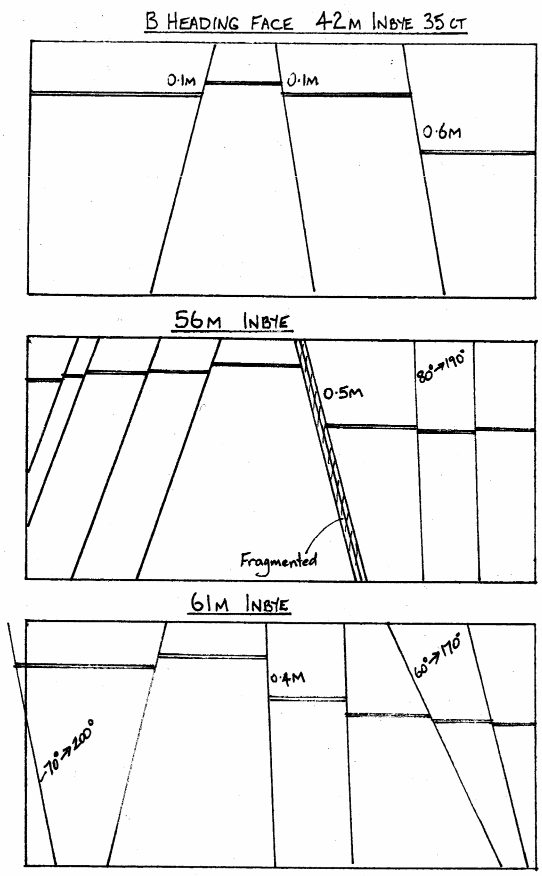

Fig. 6 Face mapping of horst/grabens

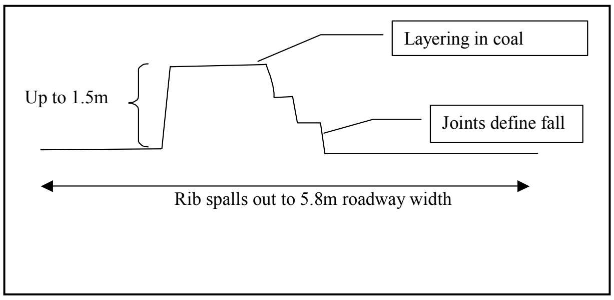

Fig. 7 Cartoon of the roof falls

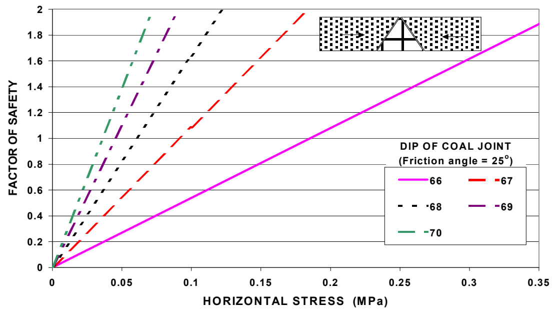

Fig. 8 Wedge analysis for coal roof with low horizontal stresses

Roof falls developed soon after the start of each plunge (Fig. 7). The falls involved the collapse of joint/fault-bounded blocks of coal from the roof. These blocks were often in the order of 300 mm to 500 mm wide and resulted in the formation of a trench in the roof. There was no rock noise associated with the falls. The longer the plunge, the higher were the roof falls – the maximum fall height recorded was about 1.5m and this happened when the plunge was extended to 5m. Trench roof was only recorded in the headings, not in the cutthroughs. The falls occur as joint-bound trenches over about half of the roadway. The fall zone migrated from the north rib to the south rib of each roadway and in this way migrated from A heading to C heading.

The falls were interpreted as being the result of low horizontal stresses acting on wedges of coal defined by the horst grabens. The low horizontal stresses resulted from the interplay of the roadway shape with a low magnitude stress field where the major principal stress is vertical. Simple wedge analysis indicated that falls were likely for discontinuities dipping at between 70° and 65°, and inevitable for surfaces dipping at less than 65° (Figure 8). The height of the falls was controlled by the onset of a more compressive regime higher in the roof.

The mining strategy was to limit the plunges to 5m, and to support progressively with 2.1m bolts, mesh straps, and butterflies. Once supported, there was no further movement. To restrict the plunge to 5m and to bolt at 1m intervals with hand held bolters was an operational decision based on the significant increase in bolting time if the falls were allowed to extend higher as a result of the plunges being taken deeper. At the same time as C heading was experiencing trench roof, the conditions in both A and B heading allowed 15m plunges. This introduced a substantial imbalance in the panel advance.

The roof support pattern adopted for the trench roof areas was empirically derived. It is focussed more on the control of immediate roof slabs and cantilevers than on the overall stability as the impression gained was that the falls rose to a height of about 1.0m and then stabilized.

More of the faults dipped to the north than to the south. As a result there were more problems with planar slides off the southern ribs – the northern ribs experienced minor toppling failures at the rib/roof corner.

SECONDARY SUPPORT

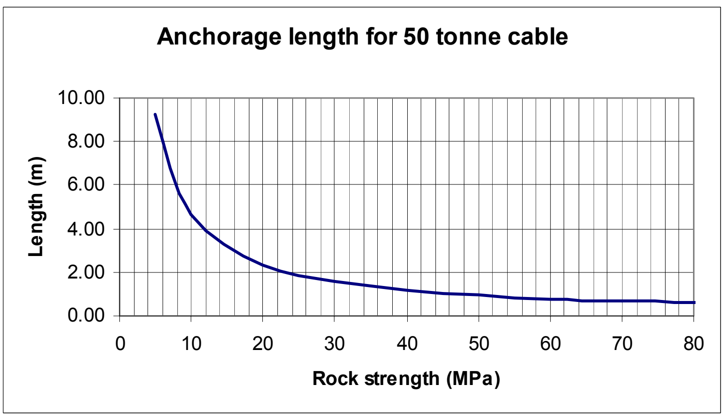

Fig. 9 Required anchorage length for cables

Monitoring identified some areas where the post-installation movement of cables exceeded the 10mm threshold. A review of the ground conditions identified that these areas were associated with low strength strata in the anchorage horizon. The distribution of overall stone roof strengths was known from sonic logs. A nomogram was developed relating anchorage length to rock strength using standard ground engineering concepts (Fig. 9). From this nomogram came the requirement for 10m long vertical cables to achieve adequate anchorage in some areas. In other areas, higher strength sandstones within the fall horizon could be exploited if angled cables were installed over the pillars.

RIBS

The high ribs required support in areas where the spacing of coal joints was less than about 300mm, and also in the horst/graben areas. The support design was based on the assumption of joint defined planar slides. The dip of the joint is an important control on the weight and dimensions of the failure block (Fig. 10). Ten tonne capacity rib bolts gave a factor of safety of two against the design blocks. Depending on the height of installation, bolt lengths of 1.2 to 1.5m were required to ensure anchorage beyond the joint plane.

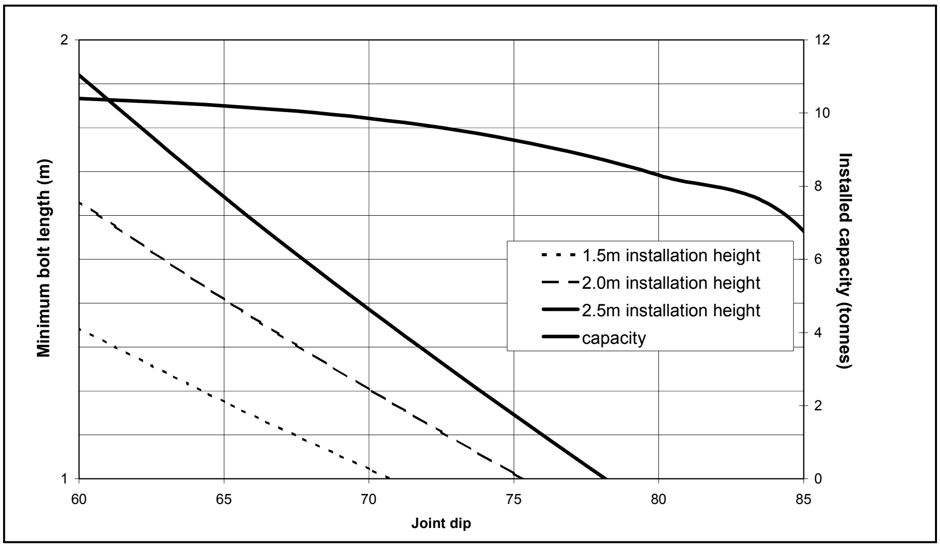

Fig. 10 Rib bolt requirements

The specification for spot rib bolts was:

- ten tonne capacity (10 tonnes/metre of rib)
- spacing of 1m or less
- minimum 1.5m length
- installed at no more than 2.5m above the floor
- point-anchored, with plate, nominal tension.

LEARNINGS

The close interaction between the mine geologist, the geotechnical engineer, the contractor and the principal allowed the rapid identification and response to changing face conditions. The drilling program and 3 D seismic studies provided essential information for planning and scheduling. With hindsight, more emphasis on coring and acoustic logging would have assisted to give a better appreciation of the nature of the ground around thrust faults and to provide data on the orientation of coal structures.

Not every geotechnical feature can be found by surface exploration nor can they be anticipated by the roof support designer. Weekly geological mapping and observations from the workforce allowed the development of models so that the roof and rib could be 'read'. Simple geotechnical models and explanations (albeit controversial) allowed everyone involved in the project to appreciate the geotechnical constraints on the development rates.

Thick coal seams have a different set of challenges related to low horizontal stresses and the role of structure. The learnings from the adit have been translated into the feasibility study for the longwall and have changed some of the fundamentals of the proposed mine layouts.

ACKNOWLEDGEMENTS

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