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Crinum Mine, 15 Longwalls 40 Million Tonnes 45 Roof Falls - What did we Learn?

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CRINUM MINE, 15 LONGWALLS 40 MILLION TONNES 45 ROOF FALLS- WHAT DID WE LEARN?

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ABSTRACT: The Crinum mine is located near Emerald in the Bowen Basin of Queensland and began development in 1994 using 2 Joy 12CM30 continuous miners in the 3.4 m high Lilyvale seam. Longwall production began in 1997 and finished the 15th and final longwall in December 2007. Over the 10 years of longwall production 40 million tonnes have been extracted and the mine has experienced over 40 longwall face, 3 main gateend, and 2 tail gateend roof falls as well as 5 roadway roof falls away from the longwall. Weak roof (less than 10 MPa in the bolted horizon) has been the principal roof control issue at the mine. However, weak, highly cleated coal, water inflow from overlying aquifers, some minor structure and a diatreme in the main headings have also contributed to the challenges at the mine. This paper describes the geotechnical experience over the 10 years, the mine’s approach to addressing the issues and the relative success of these approaches.

INTRODUCTION

The Crinum Mine is a BHP Billiton Mitsubishi Alliance (BMA) underground longwall mine located 45km north of Emerald, Queensland (Figure 1). The mine began development in 1994 and longwall production in 1997 and finished the 15th and final longwall in December 2007. Crinum has mined over 40 million tonnes and experienced over 40 longwall face falls of ground that required reconsolidation. In addition, the mine experienced five main gate and three tailgate roof falls as well as five roadway roof falls away from the longwall (Figure 2). Weak roof (less than 10 MPa in the bolted horizon) was the principal roof control issue at the mine however weak friable coal, water inflow from overlying aquifers, some minor structure and a diatreme in the main headings have also contributed to the challenges at the mine. This paper will describe some geotechnical experiences at the mine with respect to development and primary support, secondary support, pillar design and longwall support. It should be noted that these are the experiences, explanations and solutions of and for the Crinum Mine and are not presented as being necessarily applicable to other sites.

EXPLORATION

Initially a 500 m square pattern of exploration drilling was carried out over the site. From this program over 150 core samples were tested for UCS. The sonic velocity of these samples was also recorded. A sonic velocity to UCS correlation was developed for the Crinum site. This correlation was applied to the bolted horizon of the mine and contoured over the workings (Figure 2). After about 5-6 longwalls it was recognized that the observations of bad ground conditions in the mine correlated closely with the <8-10 MPa contour on the derived UCS plot. Through further experience with the UCS contour and trials of drilling density, a surface exploration borehole spacing of 130 m down each gateroad prior to development was adopted as a procedural requirement for hazard plans (Figure 3). Since that time the UCS contour has been extremely accurate in planning and budgeting for the different bolting densities at the mine and predicting areas where cable bolting will be required and budgeting quantities.

This technique is further refined using the Roof Strength Index (RSI) developed at Kestrel which incorporates depth of cover and demonstrates that weak roof at greater than 150-180 m depth is much greater an issue than weak roof at less than 150-180 m depth.

The inability to carry out 3D seismic due to a layer of basalt near the surface or predict a couple full seam displacement faults with the already dense borehole spacing resulted in at least two changes to the mine plan when encountered underground.

PRIMARY SUPPORT

Full mesh versus W Straps

The mine began with a 6 bolts per metre pattern of 2.1 m fully encapsulated torque tension roof bolts installed through W straps. Even though this initial area of the mine was some of the best roof conditions; bedding, especially cross bedding, resulted in slabs of material falling between straps and injuring operators (Figure 4).

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Within 10 pillars of entering the seam, full roof screen was employed. Subsequently the original straps have corroded and caused a hazard from dropping off the roof.

A 4 bolt per metre trial was carried out that may have been successful if it had been located in the best roof and shallowest section of the mine. Unfortunately the best roof area only existed in the initial mains area.

![Figure 1 - Location plan of the Crinum Mine](image1)

Figure 1 - Location plan of the Crinum Mine

![Figure 2 - 2m UCS contour overlay onto the Crinum mine plan with the locations of roof falls also plotted](image2)

Figure 2 - 2m UCS contour overlay onto the Crinum mine plan with the locations of roof falls also plotted

**Optimisation of the Manager’s Support Rules**

When weak roof was encountered (below 10 MPa and as low as 3 MPa) several roof bolt pattern density increases were trialed. The first being to decrease the spacing per row to 0.75 m and when this was not successful, dropping the spacing down to 0.5 m. Even though this doubled the bolt density to 12 bolts per metre the roof continued to bag and lower between the bolt closest to the rib and the second roof bolt in the pattern, usually on the stress notched side of the road first. A trial of installing two extra bolts over the normal six bolt pattern between the outside bolt and the next bolt on each side of the roadway was implemented (there after called the 6:2 pattern, Figure 5). This bolt crossed the typical line of roof failure and was very effective in reducing roof movement. Of the
10 roof falls in roadways at Crinum mine, none occurred in a roadway with the 6:2 pattern installed. In fact, in one instance (MG8, B17-18), the roof began to deteriorate immediately behind the miner in development. The deputy invoked the 6:2 pattern for the next 50 m. Reports of deteriorating roof continued outbye and between shifts a 66 m long roof fall occurred trapping the continuous miner inbye. After two weeks of recovering the 66 m of fallen ground it was discovered that the fall had pulled up at the 6:2 pattern. The inbye lip of the fall demonstrated the failure line and mechanism (Figure 6). It was after this roof fall that a decision was made to rely on the hazard plan to trigger increases in bolting pattern densities rather than wait for roof movement triggers in the Strata Management Plan to invoke changes. The reliability of the UCS contour on the hazard plan eventually required the use of the 6:2 bolt pattern in 50-60% of the development drivage. This ultimately saved significant quantity of drivage time and material cost over the life of the mine had the more dense 0.5 m spacing been used or if a blanket 6:2 pattern had been adopted mine wide. A record drivage rate of 65 m in a 12hr shift using the 6 roof and 5 rib bolt per metre pattern, whereas a record of only 45 m in a 12 hr shift was the maximum achieved on the 6:2 pattern (8 roof bolts and 5 rib bolts per metre).

Training

Roof bolting is the most critical operation at any mine. If done correctly it provides stability for all other operators working under it. Early classroom training recognized the wide range of experience and understanding that operators had with roof bolting. Therefore a one hour long training DVD including actual video and video animation was developed to properly train operators in the theory of roof bolting and best practice installation.

Subsequently automated bolting rigs have decreased the variability in hole depth, penetration rate, spin time and hold time to nearly zero. Now every roof bolt is installed nearly identically and as close to manufacturer’s recommendations as possible.

Encapsulation / Roof Bolt Tension

The 2.1 m roof bolts installed in holes drilled with standard 27 mm win bits with a 1m two speedie resin have always been unencapsulated by 100-300 mm. This is mostly due to overdrilling/reaming of the hole to 28-29 mm and the fact that using the standard calculation for required resin quantity +25% is inaccurate for weak roof conditions. This remains the case to this day.
The tensioning of roof bolts at Crinum was initially verified through testing torque with torque wrenches after installation and quarterly roof bolt installation audits in which a torque-tension measurement was carried out. This was necessary as the breakout of the nuts was 108-122 Nm (80-90 lbft) and the Strata Management Plan required 203 Nm (150 lbft) torque on the nuts. The frequency of this check was increased to each bolting rig every 30m of advance by supplying two high torque test nuts in each supply pod. This continued until automated roof bolters were employed at Crinum at which time the entire stock of roof bolt breakout nuts was increased to 203 Nm (150 lbft) allowing a torque test on every roof bolt.

Experience at the mine suggests that tensioning of roof bolts in weak, highly laminated strata has some benefit to roof support and beam formation (approximately 10-15% improvement). However the act of tensioning performs a small pull test on the installation and ensures the plate is very tight against the immediate roof. It is believed that these two factors are at least as important as the traditional "clamping the beam together" theory. It is also believed that achieving full encapsulation would have increased benefit again.
Gloving / Overcoring

During the period of industry wide concern over gloving and unmixed resin, over coring was carried out at Crinum to assess the extent of the issue at site. At the same time some of the initial Hilti One Step bolts were being trialed at the mine. The overcoring showed no gloving or unmixed resin in the bolts overcored (although some had been observed in the goaf behind the longwall). Overcoring of the One Step bolts showed perfectly mixed resin (Figure 7). The overcore from both bolts showed that the roof material cored around the unencapsulated length broke up readily during the overcoring (lack of consolidation and confinement provided by the resin, Figure 8). This may indirectly demonstrate the benefit of fully encapsulating bolts. It also demonstrated the different borehole wall profile between the modified spade bit and the wing bit (Figure 9).

RIB BOLTING

With 3.4 m high roadways and highly cleated coal, the mine started with 2 x 1.2 m steel rib bolts and butterfly plates on both sides of the roadway in the main entries and the same with 2 x 1.2 m plastic cutable bolts in the block side. After reaching about 150 m depth it was realized that 3 improvements were required. Firstly, the removal of belt structure at the maingate was becoming risky due to sloughing of the rib in the longwall abutment zone. It was decided to increase the density of cutable bolts to 3 per metre after only 2 years of longwall mining. This improved conditions, however the observation of regular shear failures of the plastic bolts especially at depth resulted in a change to stronger fiberglass rib bolts for longwall 10. Also in line with the depth of cover of 150-180 m depth of cover and pillar side rib control issues at the maingate walkway and in the tailgate travel road, the installation of welded wire rib mesh on all pillars was adopted. This resulted in greatly improved rib performance, safety and tailgate travel road conditions (Figure 10).

Rib Bolt Encapsulation

Encapsulation has always been poor on the 1.2 m rib bolts using a 660 mm resin cartridge which consistently resulted in 300-500 mm unencapsulated. It is believed that this unencapsulated length contributed greatly to a wide range of rib failures including both tensile and shear failure of the bolts, rib spall around the bolt to the depth of encapsulation, nuts pulling through the plate, etc. and was more significant than the amount of tension applied to the rib bolts. Attempts to increase encapsulation by increasing resin quantity, or decreasing bit size always resulted in premature breakout or torsional damage to the rib bolt heads. The recent development of higher strength and breakout rib bolts has resulted in the ability to fully encapsulate rib bolts using 1 m resin capsules. This reduces the already simplified three types of resin at the mine (1m slow set for cable bolts, 660 mm fast set for rib, and 1 m two speedie for roof) to just 1m two speedie for both roof and rib and 1m slow set for cable bolts and has already shown improved rib conditions on development (Figure 11).
Figure 8 - Broken overcore demonstrating lack of consolidation and confinement at the unencapsulated end of a standard roof bolt (bottom) and self drilling bolt (top)

Figure 9 - Cross section of borehole showing rifling achieved by standard wing bit
SECONDARY SUPPORT

Due to the requirement to maximize development rates to keep up with longwall retreat a practice of only installing enough roof support off the miner (6-8 roof bolts and 5 rib bolts per metre) to allow development to block out a pillar, and if secondary support was required it could be installed by contractors outbye. Due to the low bearing capacity of the roof 300 mm bearing plates were required on cable bolts as 200 mm plates would easily crush into the roof.

6 m vs 8 m Cable Bolts

The standard cable bolt length was 8 m based on initial consultant recommendations. However roof falls in roadways at Crinum have been the roughly triangular shape (Figure 12), the height of which has always been within 400 mm of the width of roadway (4.8-5.2 m high). Using the 4-6 m horizon above the natural arch shape of the roof failure as being secure, 6 m cable bolts installed at 1/3 the way across the roadways, have been employed in up to 50% of cable bolt installations, but only when hazard plans showed good anchorage in the 4-6 m horizon (Figure 13).

Post Grouted vs Point Anchor Cables

After an initial practice of installing passive cement anchored cables, the mine employed resin point anchor post tensionable cable bolts as the standard secondary support. This practice was successful in developing in, and taking the longwall through, some very weak ground. Although conditions looked bad, with a large amount of roof
lowering, no maingate roof falls occurred in cable bolted ground. When post grouting cables became popular, this practice was adopted at Crinum. Unfortunately two roof falls occurred in the maingate over the BSL in areas that had post grouted cables installed (both below 180 m depth of cover).

The main issues identified were:

- Crinum installs cables outbye, therefore if the roof has already delaminated, the failure horizon has been predetermined.
- The grout encapsulates the cable and therefore if the roof loads it at one horizon (the predetermined failure horizon), the roof only needs to move a small amount to take the cable past its ultimate strength as there is very little elongation available.
- As the cable is fully grouted there is no sign of weight on the bearing plate and together with the minimal movement required to fail the cable the deputies were not able to identify imminent danger.
- The abutment loading of the longwall is seen as somewhat of an unstoppable force. Some roof movement will occur in weak roof.

A decision was made to revert back to point anchor cables in gateroads only. Crinum had never experienced a maingate fall in cable bolted ground for the first 10 longwalls, experienced two consecutive maingate falls on longwalls 11 and 12 using post grouted cables (Figure 14) and then no falls since on the final three longwall gateroads.

Beside the ability to properly visualize roof deterioration with point anchored cables and allow significant roof movement before failure, the installation of post groutable cables allows the subsequent use of the grout tubes to inject PUR into the already existing cables (and roof fractures) instead of having to try to wet drill around the congested area of the BSL and install extra cables or inject PUR when the roof is already in bad shape at the gate end. Effectively it allows for a fourth line of action response. This has had to be actioned only two times at Crinum in the maingate intersection but potentially prevented roof falls on both occasions.

Spiling through the gate end falls along the roofline with about 14 HQ drill rods from outbye the 10-14 m long fall to up over the maingate canopies and installing steel sets underneath after chipping out the fallen material appeared to be the best and was the only method used to recover the three maingate falls at the mine. Limited material had to be removed, limited damage was done to the BSL and no false roof had to be reestablished.
The policy of not grouting cables was *not* applied to installation roadways. The intention on installation roads has been to build a roof beam and minimize the amount of roof movement and thereby maintain as much inherent strength as possible and has been the standard since LW6.

The concept of building a strong thick beam has been taken to an extreme at the new Crinum East mine where a thick layer of extremely weak roof (3 MPa) exists at the 6 m horizon and above (Figure 15). Historically 8 m and 10 m cable bolts were employed on installation roads at Crinum with height of softening extending to 5-6 m. Due to the difficulty in drilling and anchoring in the 6-10 m horizon, the design methodology of forming a thick beam was followed. 6 m long, 80 t cable bolts were installed on a dense pattern, pretensioned to 40 tonne and post grouted. Although the face road was widened to 7.8 m for line shields and up to 9.8 m wide for gateends and shearer stable, minimal additional roof movement occurred in the 6 m beam.
Top Down Versus Bottom Up

The argument over bottom up versus top down grouting of cables was demonstrated in one particular application at the mine. An install road (LW6) was driven first pass and experienced a range from about 25 mm at the gate ends to greater than 300 mm roof movement near mid face on first pass of a 4.8 m wide roadway. A roof support plan of cable bolts and trusses was designed and included bottom up grouting of the cables. The installation road cables took pallet after pallet of microfine cement especially in the area of greatest roof movement with the cement migrating into all the open bedding planes. It was reported that pumping was discontinued at one point due to the sound of cracking in the roof and gurgling and sloshing of the wet cement in the open bedding planes in the roof. When it came time for widening, the roof which had the greatest amount of movement first pass experienced the least amount of movement second pass. In fact any areas which had greater than 90 mm of roof movement experienced very little roof movement second pass and areas with less than 90 mm experienced significant roof movement (Figure 16). It was postulated that this phenomenon had two causes. Firstly, the grout was of a greater strength than the initial roof material itself and created stronger beds of significant thickness. Secondly, as roof fails it opens up bedding planes and slides along bedding planes. The rate of roof movement increases exponentially as the fractures increase and the frictional resistance is reduced by the newly created openings. Refilling those voids with grout reestablishes the frictional contacts on the bedding planes and provides a bulk material which has to go through the failure process again. As the secondary support has already been installed, the reconsolidated roof “mass” performs better than the initial roof. PUR injection into the roof works in the same way with the greatly added benefit of adhering to the bedding plane contacts and being even stronger. This experience is qualified in that top down grouting may be more applicable in moderate roof that has little initial movement and maintaining the integrity and inherent strength of the beam itself is the goal.

High Strength - High Tension Cable Bolts

An ACARP assessment of high strength, high tension cable bolts was carried out in a particularly weak area where development was being slowed by excessive roof movement. The A heading (travel road) of Maingate 7 required cable bolting after every about 40 m of drivage, resulting in the machine being flitted to the belt road and “B” heading being driven just slightly behind A heading. The B heading didn’t require cable bolting which was assumed to be due to the stress relieve caused by A heading being driven first. This was further demonstrated when the 80 tonne cables in A heading were finally tensioned to 40 tonnes and subsequently the B heading became unstable and required a full complement of cable bolts to regain stability. It was a result of this trial and the constant problems maintaining stability on installation roads that 80 tonne cables with 40 tonne tension became the standard support on installation roads.
Corrosion

It was recognized after five years of cable bolting in the main entries that post grouting in life of mine entries (+5 yrs) was applicable. This was demonstrated by approximately 35 cable bolt failures in an area of 3 MPa roof which continued to creep over the life of the mine. The failed cables showed excessive corrosion and would have benefited greatly from being fully grouted with a corrosion inhibiting grout.

PUR Injection

The Crinum mine was one of the largest consumers of PUR in Australian coal mines for two years, exceeding 300 t both years. This was primarily for longwall falls but also included installation roads and other outbye roads.

Bolted Roof Injection

When injecting in roadways a procedure was setup which included the following:

- Standing support (usually propsetters) was required to be installed in the roadway prior to injection
- Pump pressures were limited to 85 bar to reduce the risk of “jacking” the roof down.
- Electronic roof to floor convergence monitoring was employed and roof movement limited to 8 mm in any one hole injected
- No inexperienced operators were allowed under the unstable ground
- The pump was located outside the unstable area
- Low or no expansion PUR was used
- The ability to pump more than 200 kg in a single hole not intersecting a coal seam was of great benefit
- Roof extensometry was reestablished after the PUR had set and before the props were removed

Longwall Roof Fall Injection

Injection on longwall roof falls was initially under the control of PUR contractors. However after initial experience, detailed PUR support and injection plans were developed by the geotechnical engineer including number, depths and angles of holes, injection limitations, use of roof bolts, cable bolts, dowels or spiles and sequence of injection (Figure 17). Injection quantities were required to be submitted at the end of each shift and plotted on the support plan for migration.

Lessons learned at the Crinum mine included:
• Migration occurs for a maximum of 4 m ahead of the face in coal and therefore coal holes were reduced to 5 m to minimize drilling time.
• Very low expansion PUR is most effective in coal
• When trying to secure faults or fractures in weak ground, drilling through and installing steel bars or dowels is critical (this applies to cementitious grout injection as well)
• 12m is about the maximum distance possible to fully PUR encapsulate a splice or dowel
• Solid bars are better as spiles than flexible cables

When trying to preconsolidate a fault or weak ground the following experience was gained.

• Drilling greater than 50 m requires full directional drilling capabilities to ensure adequate accuracy
• Spiling weak ground requires a density of about one splice per metre
• Due to itsbrittle nature and lack of adhesion consolidating by injecting cementitious grout into broken ground is prone to longwall abutment reactivating the fractures and negating the consolidation effect. The use of steel in these holes increases their effectiveness ten fold.
• There is a huge tendency for the cement injection personnel to water down the cement mix in order to gain pumpability. A ratio of 1:1 water:cement will usually equate to about 12 MPa strength on microfine cements.

**LONGWALL FALLS**

During the two years of highest PUR consumption the mine experienced very weak roof areas (< 5 MPa for up to 2 0 m above the seam) on two different longwalls. On Longwall 7 the mine experienced 19 consecutive roof falls over the final 140 m of the longwall panel while on Longwall 9, 15 consecutive roof falls occurred (Figure 2). These falls occurred as soon as the longwall had mined past the influence of the previous PUR reconsolidation.

The learnings from these falls included:

• The number of events provided operators and managers with the experience that trying to “catch the lip” almost always resulted in a larger fall and longer recovery and therefore the decision to pull up and PUR was eventually made at the first sign of trouble and therefore longwall triggers were reacted to much sooner.
• Less than 5 MPa roof for a height of greater than 4 m above the seam is extremely difficult to control even with good mining standards
• Longwall shields must be in excellent condition to control weak roof as it is extremely unforgiving
• When injecting PUR into fractured weak roof always include steel dowels, cables or bars. Weak roof will simply rip away from the PUR leaving sheets of PUR with a thin coating of the host rock attached to it (Figure 18). The steel provides a tensile resistance to the reopening of the fracture.
• When trying to reestablish longwall production during a roof fall in which the wall will continue to mine in weak or fractured ground, spiling is an effective means of providing a false canopy.
• Several times Crinum installed 6 m and 12 m dowels horizontally just above the roofline ahead of the face, at least one if not two per shield (Figure 17), and injected them with PUR (not for the purpose of consolidation but for encapsulation and anchorage). The false roof allowed the entire canopy of the shields to get under solid roof, reestablish adequate set pressures, clear any stone remaining on the face and get the longwall back to full production before leaving the zone of consolidation thereby giving it half a chance of maintaining stability and production rates.
• Weak roof combined with weak friable coal is particularly prone to roof falls when production slows due to delays or preparation for longwall takeoff. It has become standard procedure to preinject the coal seam with PUR ahead of the face when coming to takeoff in weak roof.

**Cavity Fill**

It is accepted practice that when a roof cavity gets to a certain size on a longwall, cavity filling becomes a requirement. The material provides confinement to walls and roof of the fall to prevent the cavity from growing provides the shields with an ability to achieve some set pressure and provides protection for equipment and operators. However it also provides some assistance in the goaf. If the longwall has loaded out a lot of stone, that stone has not reported to the goaf and therefore there is more room for roof material and further caving. It is possible that this allows the goaf to progress further ahead of the face.

Although a more expensive material, the phenolic foams are much quicker and require less high risk shuttering than standard cementitious foams, thereby making them cost effective (Figure 19).
Figure 17 – PUR injection – longwall recovery support plan

Figure 18 - Side and end view of PUR which had been injected into a fracture in a longwall roof fall and then ripped back out during remobilization of abutment stresses demonstrating the greater adhesive strength of the PUR than the weak rock itself.
A learning at Crinum was how important it is to get an accurate estimate of the volume of the cavity required to be filled. Contractors and suppliers are good at looking at a cavity and saying it will require X amount of product and then if it requires more explaining it by saying the cavity was bigger than we thought, however on two occasions during roadway fall rehabilitation, Crinum was able to measure the exact volume of the cavity by using measuring poles and survey cavity measuring equipment (Figure 20). Using the expansion factor of the foam cement quoted by the supplier, which makes the product appear cost effective, these cavities should have only require a quoted amount of material (including the dense low expansion layer at the bottom of the cavity). However it was discovered during the application that nearly twice the quoted amount of material was used. This could be attributed directly to improper mixing and/or product performance. This process can be quality assured by accurately measuring the volume of the cavity and regular and random sampling and testing of the delivered product at the nozzle.

Figure 19 - Cementitious cavity fill material (left) versus Phenolic foam (right) showing the reduced amount of shuttering required.

Figure 20 - 3D plot of roof fall cavity measurement using survey instrumentation
PILLARS

Crinum gateroad pillars are 30 m wide rib to rib down to a depth of 220 m. This gives a UNSW pillar design Factor of Safety of about 1.4 which is rated as a probability of failure of about 2 in 100. At 3.4 m high and 200 m deep, Crinum pillars were showing signs of significant load and were demonstrating that they were close to their design limit. However as it was only the initial pillars in each longwall (the panels mined up hill) that were at this depth it was decided to leave the pillars at 30 m solid and install some additional secondary roof and rib support in the deepest pillars to increase the stability in this localized area. This double rib bolting and 50% overlapping additional mesh provided the necessary confinement to ensure stability and was an effective means of maximizing the resource. The benefit of this rib bolting pattern had been quantified in a previous ACARP funded rib support trial. Pillars in Crinum East will be fanned to a width of 35 m where they go down to 250 m depth of cover. The increased width will improve stability and minimize water and gas connectivity from previous goafs.

Instrumentation

Sonic Probes

Initially the mine used sonic probe extensometers to perform detailed roof monitoring. This resulted in valuable information to validate the initial roof support and mine design model. However it was realized that when the mine reached the routine production phase, sonic probe extensometers had the following limitations:

- The probes were fragile and easily damaged
- The probes were expensive to repair or replace
- Readings from one probe to another or to a repaired probe are not the same and require a recalibration of readings.
- The readings were not directly interpretable underground at the site and therefore triggers could not be acted on until readings were delivered to surface and input into a computer

Routine Monitoring

The routine monitoring and trigger response stage of the mine was better served using mechanical telltales at every standard intersection and four point electronic extensometers at critical areas such as installation roads and takeoff roadway intersections. The electronic telltales provided an accuracy of greater than 0.1 mm and immediately interpretable readings underground. It was also realized that responding to instrumentation was dependent on frequency of reading and speed of data input and therefore a procedure was set up such that all telltales and resistive potentiometer extensometers would be read by ERZ controllers and the data entered by the control room operator on shift. This included outbye telltales read by outbye ERZ Controllers on a schedule similar to stone dust and bag samples.

A further improvement was made to the traditional overlapping tube style telltales by converting to the clock it style telltales. This enabled an improved accuracy of about 0.5 mm and discontinued the practice of assuming the reading hadn’t changed or guessing at the reading when a ladder wasn’t available or was too much trouble to carry around, which was necessary in a seam height of 3.4 m with the old style telltales. An improvement to this style of telltale would be a clear plastic cover over the dials and further corrosion resistance.

A limitation of mechanical and electronic extensometers is their susceptibility to corrosion if water is present. Some initiatives were attempted to reduce this problem with little success. More successful was a regular inspection of readings and instrumentation to ensure it is still functional.

TARPS

Roof movement TARPS at the mine have always been L1, 10 mm in either horizon, L2, 20 mm in either horizon and level 3, 40 mm in either horizon. At mid mine life a mine wide telltale results review was carried out. This review showed 50% of the telltale readings (either horizon) showed less than 10mm of roof movement, 15% less than 20 mm, only 5% between 20 and 40 mm and 30% showed more than 40 mm of roof movement (Figure 21). In addition several plots were made of rates of roof movement showing acceleration points at 10 mm and 20 mm. These results validated the trigger levels set showing that below 10 mm the roof was stable and required no further action, between 10 and 20 mm it needed a more frequent inspection, at 20 mm it needed a higher level inspection and plan for secondary support because if it made it past 20 mm without secondary support it would continue to 40 mm after which a roof fall was imminent.
Blast Monitoring

Blast monitoring for vibration from open cut blasts can be very helpful in understanding the effects on underground workings. The application of 1000 m exclusion zone is not applicable for underground as the underground environment is not exposed to fly rock hazards. After sufficient monitoring of vibration from open cut blasts a graph can be developed which predicts the expected vibration (peak particle velocity (ppv)) from any individual blast based on Maximum Instantaneous Charge (MIC), distance and type of blast (pre split, overburden, pre strip, etc). Values below 2 ppv are difficult to sense by operators especially if there is any noise or activity underground. 2-10 ppv (especially 7-10 ppv) are perceivable by sound and vibration and operators should be notified of the plan and expected time of the blast. Values 10-21 ppv are loud, can create a wave of imbalance if standing on a platform, cause rattling of steel in a crib room and displace bits of dust and rib along roadways. Figure 22 shows results from the monitoring of an open cut blast within 700 m of underground workings.

Figure 21 - Mine wide telltale results used to validate alarm levels.

Figure 22 - Radial distance from open cut blast to underground workings (left). Blast vibration monitoring results from underground workings (right).
LONGWALL SUPPORT

7 day Operation

When longwall mining started in 1997 Crinum was a 5 day a week operation. Very often, after the weekend downtime, a roof fall or significant roof slabbing would develop on the first few shears Monday. By 1999 the mine switched to 7 day mining which solved this recurring problem and reduce it to after major equipment delays.

Monitoring

In 1999, Crinum commissioned the GeoGuard system of longwall shield monitoring. That system ran successfully, albeit with its own limitations, for six years. It demonstrated that no significant weightings were occurring at the Crinum mine as shown in Figure 23. If there was a caving interval it was about 10 m which was not sufficient to generate excessive loads. GeoGuard was also used to monitor shield condition, maintenance, set pressures, etc. Routine audits of longwall support performance included plots from GeoGuard which showed shields which required priority maintenance and these audits were submitted to longwall maintenance for action.

Maintenance

A general learning was that after seven longwalls shield maintenance became critical. Staging (blipper) valves became clogged with debris and often didn’t operate correctly causing the leg pressure to operate on the smaller upper cylinder reducing the set and yield force applied by the shield by 30-40 %. This was initially recognized by the lower cylinder gradually climbing up to its full extension. Once at full extension the set force of the shield is reduced even further as the lower leg comes in contact with the steel of its outer housing. Yield valves were discovered to eventually yield at a much lower pressure as the orifice which controls the yield pressure scour out after years of yielding. Faulty check valves can limit the shield pressure to line pressure only. Pin slop in the linkages can allow significant horizontal movement before resistance is applied to the roof by the canopy depending on what point in the “S” curve the canopy is set to the roof. In addition seals began to leak. Often shield electronics will make decisions based on the pressure in the MG leg. The tailgate leg pressure is irrelevant.

Support Design

It was eventually recognised that the shields at Crinum had the following design limitations:

1. The flippers (coal deflectors) have a designed rotation of only 90° to deflect coal from the 3.4 m seam and prevent it from toppling across the panline and spill plate to the walkway. If they had been designed to today’s double-knuckle 180° ability it is believed that some roof falls could have been prevented and operator protection would be improved during roof fall recovery and longwall bolt up. Because the face spalled immediately after cutting and generally buckled at the stone bands (in fact the shearer at Crinum is more of a loader and has to cut very little intact coal) the 90° coal deflectors frequently could not contact the coal face even when double chocked (Figure 24).
2. It was recognized that Crinum mine would be mining in weak roof conditions during the initial mine design and shield specification. Therefore the canopy was designed long to extend out ahead. In fact, when double chocked the canopy extends 0.57 m into the next web of the shearer. However due to the 3.3 m seam height, there was insufficient room to make a rear walkway. This meant that a walkway had to be maintained in front of the legs of the shields. The long forward canopy together with the minimum walkway results in a poor canopy ratio and a poor tip loading of the shields.
3. It is believed that having the shield legs closer to the face would have provided better support and a better canopy ratio, however there are dimensional limitations on how this could have been achieved.
Longwall Support Improvements

The following improvements were made at Crinum to improve shield performance in addition to the routine maintenance program:

1. Routine shield pressure monitoring was initiated
2. Longwall support audits were carried out by the geotechnical engineer
3. A dedicated high set pump system was installed
4. A study of the hydraulic fluid delivery system was carried out and the hydraulic supply lines were increased in size and the filter sled reengineered to reduced restrictions.
5. A shield leg changeout program was initiated
6. 19 spare shields were purchased in order to allow a future face extension but also enable a changeout and rebuild of worn shields at each longwall move prior to the face extension (enough to rebuild every shield).
7. A yield valve change out program was initiated
8. The control system was changed
9. A full retrofit is being carried out prior to installation in the new Crinum East Mine

After a tailgate fall which occurred between 915 mm diameter tin cans spaced at 5 m (in which a stress window on offset longwall start positions contributed), the mine adopted a blanket support plan of 915 mm diameter cans on 3 m centres. After a period of monitoring and some staged trials this pattern was eventually dropped to 700 mm diameter cans on 3 m centres in areas of weak roof or at a depth greater than 180 m (Figure 25). Standing support has always been installed in the maingate travel road behind the wall to prevent the requirement of installation in the dust of the subsequent operating wall. Although this limits clearance for access to the tailgate end of the wall (which the 700 mm cans improved equipment access up to a 913 loader) it allows early loading of the standing support. Roof extensometers revealed that peak roof movement in the travel road occurred 50 m behind the face and continued until 270 m behind the face. The practice of trying to keep the cans even with the wallface allowed improved conditions in the tailgate and standing support that was well and truly set to the roof before tailgate abutment approached.

Bolt up for longwall shield recovery was switched from high grade AX roof bolts to low grade mild steel bolts which reduced the snapping off of roof bolt heads by the force of the shield canopies. This had no negative effect on roof support during takeoff.

A trial of rib sprays was carried out and it was determined that any stiff highly reflective shotcrete could be used in workshops where low stress and no change in stress was to occur. However for longwall abutment areas these stiff products cracked and fell off in dangerous slabs. A more flexible product worked reasonably well on longwall takeoffs in combination with friction bolts where face spall had previously been a big problem using polymer grid mesh and resin rib bolts (Figure 26).
WATER INFLOW

Mining under Aquifers

After several years of “dry” mining the Crinum longwalls began to mine beneath a tertiary flow of water bearing basalt. Longwalls 7 and 8 showed signs of increased water make on the seals at the back end of the wall (lowest elevation). Both these walls mined within 90-80 m interburden to the aquifer with a progressively thinner layer of clay below the basalt. This is consistent with other mines in the Bowen Basin with 90m being the start of water percolation at other sites. However Longwall 9 mined to less than 70 m interburden (Figure 27) at a low point in the bowl shaped aquifer and experienced an inrush which was initially estimated at 120 L/s and settled to 75 L/s. Without an adequate pumping system in place this resulted in unacceptable pressures to build on the back end seals and these seals were subsequently opened to release the pressure and flood the installation road of longwall 10 (eventually causing a discontinuity in production during the longwall transfer).

The mine had initially been setup to handle water make from the goaf of the first few longwalls. However after several years and panels of “dry” mining this set up had not been advanced with the mine workings. The learning was to always be aware of the distance to overlying aquifers and have adequate pumping systems in place to handle any potential water make.
Flooding and then Reestablishing Workings

In addition to learning the limit of interburden distance, several things were learned about the flooding of an installation road. The Longwall 10 installation road had already been widened and was flooded to a depth of 8 m above the roofline, which included five pillars out the main and tailgate. Install roads in this area were notorious for requiring roof rehabilitation usually in the form of PUR injection. Depth of cover was 200 m. A strange wave action existed at the waters edge which was never fully understood but was postulated to have been created by the action of pumping the water, differential pressure between the MG and TG or the operation of an M20 air pump which was discovered to have been operating underwater. Due to the weak roof and rib many persons at site believed the faceroad would certainly collapse and be unusable and a new faceroad would be required. The expectation was so high that life jackets (Figure 28) were required for anyone accessing the edge of the water in anticipation of a rush of water caused by a fall of ground on the face road. Actual observations and learning included:

- The entire faceline roof stayed intact with some small areas of local flaking
- The ribline was in excellent condition
- Although flooded to 8 m deep an air pocket was trapped on the roof of the faceroad and a 100 m section never came in direct contact with water (Figure 28)
- The roof areas which did come in contact with water (when open extensometers holes were inspected) the water only penetrated a maximum of 200-300 mm into the roof strata.
- The action of the waves on the roofline scoured out any bagging roof and the preexisting gutter in the maingate was washed out to reveal a height of failure of 1 m along the chain pillar side (Figure 29).
- The action of the waves caused the broken roof material which had been sitting on the mesh to wear through the mesh and form hundreds of perfect 100 mm x 100 mm square and bedding plane thick pieces of rock. (Figure 30)
- The action of the waves washed a clay band out of the seam and caused an up to 50mm gap between the top of the pillar and the roof line for a significant distance over the pillar from the roadways. This didn’t cause any problems for longwall mining.

![Contour of Interburden between Seam and Aquifer](image_url)

Figure 27 - Interburden isopach showing thickness of Permian strata between the seam and a tertiary aquifer
Figure 28 - Water level 4 pillars outbye installation road in the tailgate (left). Installation road after being pumped out and showing the water line where a pocket of air was trapped for 100 m

Figure 29 - Pre existing roof bagging which was washed out by the action of the water
Figure 30 - Examples of roof material which eroded through 100 mm x 100 mm welded wire roof mesh during the action of the water

DIATREME

The dense borehole spacing at the mine did not predetermine the existence of a diatreme which was present in the main entry trunk conveyor roadway. Development began to observe the coal seam become sugary with an increase in mineralization. Then with little other transitional indications a 70 m by 30 m mass of broken, semi consolidated material was intersected (Figure 31). Although a few diatremes had been encountered in NSW no others were known to exist in the Bowen Basin. It was learned that roof bolt anchorage was beam formation was not possible and the zone required steel sets complete with concreted inverted to successfully mine through it. Although thought to be bad luck for the only diatreme known to exist in the Bowen Basin to occur in the main trunk conveyor road at the mine, its or another diatreme existence in the longwall panel would have been worse.

CONCLUSIONS

The Crinum mine successfully operated for ten years under less than favourable ground conditions, generally producing within the top five longwall mines in the country. This was achieved by adapting to and developing procedures for dealing with these difficult conditions. Numerous learning were realized during the life of the mine, not all of which were in line with traditional beliefs.

There is a saying “We will know exactly the best way to mine and support this ground by the time the mine closes”. The Crinum mine was almost there, but not quite.

Figure 31 - Broken semiconsolidated material of the diatreme (left) and its contact boundary with the coal seam (right).