The application of coupled bolts in managing adverse horizontal stress conditions at Oaky Creek coal

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Development drivage of Tailgate 21 commenced in May 2000. Difficult roof conditions associated with the splitting of the G ply coal away from the immediate seam into the roof were anticipated. 4.1m Coupled Bolts were considered to be a potentially cheaper and more effective reinforcement system than long tendon cables for these conditions. Coupled bolts have been used very effectively in similar conditions at Alliance Colliery. Coupled bolts were trialed for the first time at Oaky No. 1 between 18-19c/t TG21 where the interburden thickness between the roof and the G Ply seam split was between 1.4m and 4m. Indicators of slight horizontal stress (slight guttering) were observed in headings and cut throughs inbye of 10c/t. Horizontal stress conditions became significant inbye of 23c/t with buckling roof conditions occurring in cut throughs. Coupled bolts have been used in cut throughs to control roof deterioration. Significant instrumentation and analysis has been used to define deformation mechanisms and a number of different support systems have been trialed. This paper outlines the use of coupled bolts in TG21 at Oaky No.1.

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The Application of Coupled Bolts in Managing Adverse Horizontal Stress Conditions at Oaky Creek Coal

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Abstract

Development drivage of Tailgate 21 commenced in May 2000. Difficult roof conditions associated with the splitting of the G Ply coal away from the immediate seam into the roof were anticipated. 4.1m Coupled Bolts were considered to be a potentially cheaper and more effective reinforcement system than long tenon cables for these conditions. Coupled bolts have been used very effectively in similar conditions at Alliance Colliery. Coupled bolts were trialed for the first time at Oaky No.1 between 18-19cft TG21 where the interburden thickness between the roof and the G Ply seam split was between 1.4m and 4m.

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This paper outlines the use of coupled bolts in TG21 at Oaky No.1.

Key Words: Horizontal Stress, Roof Split, Coupled Bolts

Introduction

Oaky Creek is a modern large-scale mining operation, producing an average of 7 million tonnes of coking coal a year. The mine comprises three underground operations – Alliance, Oaky North and Oaky No.1 Mines - in Queensland’s Bowen Basin, a coal province internationally renowned for its high quality coking coals.

Development of the Oaky No.1 underground mine commenced in June 1989 and operations utilising longwall technology began in November 1990. Oaky Creek’s underground coal is mined from the German Creek Seam in the German Creek Formation. The German Creek seam is up to 4.5 metres thick, averaging 2.6 metres.

Mine Plan and Mining Conditions

The area of Oaky No.1 Underground Coal Mine which is the subject of the paper is between 23-34cft Tailgate 21 (TG21), (see Figure 1). Progressively worse roof conditions were experienced as TG21 advanced inbye from 1cft. The standard support

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Diagram 1 - Mine Plan

Affected Area
of 4 x 1.8m bolts per metre was increased to 6 x 1.8m bolts per metre at 16c/t. Long tendon / additional support was then needed in cut throughs inby of 24c/t due to the occurrence of buckling roof conditions.

Geology

The immediate roof in TG21 23-34c/t is an interlaminated sandstone / siltstone (see Figures 2 & 3). The sonic derived UCS of the roof in this area is around 50MPa, however the laminated nature of the strata and the presence of mica on bedding planes means that the strata has very low strength in bedding plane shearing.

Four distinct changes in roof lithology occurred over the length of TG21:

• 0-9c/t: 50-60MPa Sandstone / Siltstone with good bedding plane shear strength
• 9- 18c/t: 17-30MPa Laminated Siltstone (0.5-2m) under the seam split with 50-60MPa Sandstone / Siltstone unit above
• 18-21c/t: 35MPa Siltstone (5-6m) below the seam split
• 21c/t and inbye: 50MPa Laminated Sandstone / Siltstone with weak bedding plane shear strength (6-13m) below the seam split

There are no major faults intersected by the workings in the 23-34c/t area. The predominant joint direction is sub-parallel to cut throughs. Joints are persistent across headings and through cut throughs and spacing varies between 1m and 20m. Joint dip is close to vertical.

Stress

The depth of cover is between 200m and 212m over the length of TG21 inbye of 7c/t. The inferred vertical stress based on depth of cover is around 5MPa. Given the low variation in depth of cover it is unlikely to be the cause of the difficult conditions associated with horizontal stress experienced at the inbye end of TG21.

There are two possible causes for the difficult conditions experienced in 23-34c/t as opposed to outbye:

1) Same horizontal stress as 7-9c/t but weaker bedding plane strength and higher stress than 9-18c/t due to stress being redistributed above seam split in 9-18c/t.
2) Higher principal horizontal stress in 23-34c/t than outbye.

The second option is most likely as similar guttering was observed in headings outbye of 23c/t (compared to inbye) despite lithological changes and a significant change in cut through conditions.

The nearest horizontal stress measurement was taken around 300m from 26c/t. This was taken using Sigra overcoring from a surface core hole. The measurement indicated a principal horizontal stress of 18.9MPa and a secondary stress of 10.7MPa. This is a significantly higher ratio of horizontal to vertical stress than has been measured elsewhere on site. It may not be representative of TG21 due to proximity to
Diagram 2 - Stratigraphic Column

0m
Mudstone
50m
Pleiades I & II
Sandstone
100m
Aquila
Mudst / Sandst
Tieri I
Tieri II
150m
Mudst / Sandst
Corvus I
Mudst / Sandst
200m
German Creek
TG21 Immediate Roof Core
the end of a 4m fault. An underground stress measurement is being taken in TG21 to verify the actual stress magnitude.

The stress direction in TG21 could be determined from underground mapping of gutturing in headings in cut throughs. Guttering typically occurred on the right hand side of headings and the left hand side of cut throughs (looking in the driveage direction). The headings were driven on a bearing of 222°. The large difference between heading conditions (0-7mm displacement) and cut through conditions (up to 44mm) indicated that the principal horizontal stress direction was close to parallel with headings and perpendicular with cut throughs i.e. 357-0°.

Monitoring

Rock-It dual height wire extensometers and Sonic Probe extensometers were used to measure roof deformation.

Sonic extensometers and Rock-Its were installed in TG21 cut throughs and intersections in the 23-34c/t area. Rock-Its were also installed in headings.

Typical roof deformation results from the sonic extensometers are shown below (Figures 4-6). It is important to note that whilst large deformations occurred in the cut throughs and intersections, very low deformations (0-7mm total) occurred in the headings.

Figure 4  TG21 26c/t D Heading Intersection (Hole Through)
The deformation data indicated that most of the softening was occurring within the bolted horizon, and up to a height of 2-3.5m. This would suggest that improving the effectiveness of the primary support could have provided marked improvement of roof conditions.

Many of the monitoring points showed a number of accelerated deformation stages. An initial surge in deformation tended to occur in the first 24hrs. After this deformation rates would stabilise until a second surge. The second surge occurred any time from 1.5 days up to 38 days. A third surge only occurred in 31c/t C Heading intersection. In the majority of cases the second and third surges in deformation were
smaller than the first. The additional surges were often associated with the
development of a new zone of softening higher in the roof. It is likely that the surges
were due to the redistribution of horizontal stress higher into the roof after the lower
section failed.

The variation in deformation across the roadway was analysed by comparing
monitoring results from the same locations but at different positions in the roadway.
The results indicated that the maximum deformation in hole through intersections
occurred on the cut through centreline and at a maximum distance off the heading
centreline towards the cut through. It can be concluded that the deformation occurring
in hole through intersections was due to failure in the cut through extending into the
intersection, rather than as a result of the increased span of the intersection.

Total deformation in break away intersections was higher than in hole through
intersections and cut throughs. In the case of 330/t the second surge in deformation
occurred in the cut through first (15/2-19/2), and then afterwards in the breakaway
intersection (19/2-27/2). This would indicate that whilst the larger span of the
breakaway intersections has had an impact on deformation, the failure which is
occurring in the cut through is also driving deformation in the intersection.

Cut through monitoring data showed that when cut throughs were driven between
headings maximum displacement occurred on the centre and right hand side of the
roadway. In contrast when the cut throughs were driven into solid coal maximum
defORMation occurred in the centre and left hand side of the roadway. The conclusions
which can be drawn from this are described in “Driveage Sequence”.

**Driveage Sequence**

Conditions observed in cut throughs and monitoring data indicated that different
failure mechanisms were occurring as a result of changes in the pillar driveage
sequence. The hypothesised failure mechanisms are described below (see Figure 7):

1. The first failure mechanism occurs via a gutter and failure plane developing on
   one side of the roadway. The failure plane then propagates upwards and towards
   the centre of the roadway. The roof on the underside of the failure plane is then
   forced downwards into the roadway by horizontal stress. This mechanism is
   characterised by maximum deformation occurring on the side of the roadway
   where the failure plane and gutter initiated. This mechanism has occurred in cut
   throughs which were driven prior to the second heading. These cut throughs were
driven into solid coal rather than between existing headings.

2. The second failure mechanism occurs via a failure plane developing between the
two centre bolts. The failure plane is initiated on the same side of the roadway
centreline as described for mechanism 1. The failure propagates upwards across
the centreline and towards the top of the opposite centre bolt. This leads to the
opposite side of the roadway roof being underneath the failure plane and being
forced downward into the roadway. As a result maximum deformation occurs in
the centre and opposite side of the roadway. This mechanism occurs as a result of
Plan Showing Stress Concentration and Guttering Location Around Roadways

C/T Driven prior to 2nd Heading advance

Sequence 3 Section Proposed Failure Mechanism

C/T Driven between headings

Sequence 3 Section Proposed Failure Mechanism
a change in the local stress field when the cut throughs are driven after both headings have been driven ie. between headings rather than into solid coal. Overdriving the headings causes the local stress field to rotate closer to parallel with the headings, and therefore closer to perpendicular to the cut throughs.

Support

1.8m Celtite T-Grade Super Pin nut bolts were used for primary support in TG21. These bolts have an Ultimate Tensile Strength of 31t and a Yield strength of 19t. They are fully encapsulated with G7 85MPa Duospeed resin to allow the immediate 0.9m of roof to be pretensioned. The level of pretension which could be achieved at the time with the drill rigs was 6.5t.

The potential for improving roof conditions by modifying the rigs to increase pretension is currently being investigated. The standard pretension which is achieved at Oaky North is 10-11t.

The bolts are installed in a 27mm diameter hole. Short encapsulation pull tests (300mm) have been conducted which indicate that the load transfer capacity is 50-60t/m of encapsulation. The Celtite CVX bolt (Pony bolt) was designed for improved load transfer capacity. Tests on the Pony bolt indicate that the load transfer capacity is around 100t/m.

The standard support of 4 x 1.8m bolts per metre was increased to 6 x 1.8m bolts per metre at 16c/t. More frequent and severe guttering had been occurring outbye 16c/t compared to inbye of 16c/t where none to slight guttering was mapped. The improvement in conditions inbye of 16c/t was likely to be due to the increase in support density. An additional bolt was installed between the outside and inside bolts on each side of the heading. The change in pattern from 4 to 6 bolts effectively doubled the support density between 0-1m from the ribline where guttering had been occurring.

Long tendon / additional support was then needed in cut throughs inbye of 24c/t due to the occurrence of buckling roof conditions. Five different variations of additional support were used:

1. 2.1m T Grade centreline at 1m spacing
2. 2 x 6.1m Point Anchored GXT cable bolts at 2m spacing
3. 2 x 6.1m Point Anchored GXT cable bolts at 2m spacing with 2.1m CVX (Pony) Bolt centreline at 1m spacing
4. 2 x 6.1m Point Anchored GXT cable bolts at 2m spacing with 2.1m T Grade centreline bolts at 1m spacing
5. 2 x 4.1m fully encapsulated CVR (Pony) Coupled bolts at 2m spacing

The configuration of the bolting rigs on the miner does not allow bolts to be installed in the centre of the roadway. As this was the zone where maximum deformation was occurring the installation of a centre bolt behind the miner, or after cut through development was trialed.
A centreline of 2.1m CVX (Pony) bolts was used with GXTs instead of T Grade bolts to determine whether the higher capacity and load transfer of the Pony bolt would improve roof conditions.

The GXT cable bolt is the standard long tendon support used at Oaky No.1. It is installed with a 2m point anchor in a 27mm hole and pre-tensioned at the face to 15t. The 4.1m CVR (Pony) Coupled bolt is made from the same rebar as the CVX (Pony) bolt (see Figures 8 & 9).

The concept of a coupled roof bolt is not new and has long been surpassed by other products such as tendons. The CVR (Pony) Coupled bolt is a new version of the concept which overcomes many of the previous shortcomings. Older systems were hampered by the use of an external coupling and a conventional machine thread. This external coupling required the reaming of the drill hole, and effectively prevented the full encapsulation of the bolt. The new version no longer requires reaming of the drill hole and can be installed fully encapsulated in a 27mm hole. It is simple and quick to install at the face, has high load transfer and can be installed in low seam conditions.

The Coupled Pony bolt was first trialed in 18-19c/t D Heading TG21 where the seam split diverged from 2-5m from the roof. Installation difficulties were experienced in this area and TG21 reverted to GXT cable bolts for 18-19c/t C Heading. The initial problems with the Coupled bolt geometry were then rectified and a second trial was implemented successfully in 32c/t and 33c/t TG21.

The Coupled Pony bolt was trialed in 32c/t and 33c/t because:
1. Significant productivity gains were anticipated due to reduced installation time compared to the GXTs.
2. Improved roof stability was anticipated due to the fact that most of the softening was occurring below 4m and the Pony bolts would provide stiffer reinforcement in this zone compared to the GXTs.

Due to the significant impact of position of the monitoring point in the roadway on deformation results, the monitoring data had to be corrected for position to be able to compare the effect of different types of support. Correction factors were calculated and verified from roadways which had more than one monitoring point in different positions. Figure 10 shows the corrected displacement at 20 days for each cut through against the support installed:

A centreline of CVX Pony bolts was used in 31c/t. All other centrelines were T Grade bolts.

It is evident from the monitoring data that either geology of stress conditions became worse between 26c/t and 27c/t and again between 29c/t and 30c/t.

Taking this into account, it can be concluded that:
• A centreline of 2.1m T Grade bolts was more effective than GXTs between 24 and 26c/t
- GXTs with a centreline of 2.1m CVX Pony bolts was more effective than every other type of support including GXTs with a centreline of 2.1m T Grade bolts.
- Coupled Pony bolts performed similarly to GXTs

The effectiveness of using centrebolts reinforces the importance of bolt positioning. Targeting bolts to the zone in the roadway where maximum deformation is occurring is a very effective method of improving roof stability.

The results indicate that the use of CVX Pony bolts as a centreline reduced deformation by 30% compared to T Grade bolts. The increased load transfer capacity of the Pony bolts had a significant impact on roof stability.

Coupled Pony bolts performed similarly to GXTs. This is likely to be due to a trade off between stiffness and pre-tension. That is, the increased pre-tension applied by the GXTs (15t vs 6.5t) offset the increased stiffness provided by the Coupled Pony bolts. It should also be considered that the overall installed support capacity when using Coupled Pony bolts was less than GXTs (36t/m versus 45t/m). PANACEA - a high stiffness, high pretension, quick to install bolt all in one!!

Conclusions

The best results for managing the roof conditions in TG21 were achieved by:
- Targeting bolts to the areas in the roadway where the most deformation was occurring.
- Improving the load transfer capacity of primary support bolts.

Similar results were achieved when comparing a low stiffness but high pretension bolt with a high stiffness low pretension bolt. Benefits may be obtained if it were possible to combine high pretension with high stiffness in a support system that is still economic to install.

Disclaimer

This paper is not intended to promote any particular support products or suppliers. Reference to particular products is necessary to define the geotechnical parameters of the hardware used and to assess the impact of applying different geotechnical techniques to manage strata control problems associated with horizontal stress.

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