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Practical Investigations into Resin Anchored Roof Bolting Parameters

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PRACTICAL INVESTIGATIONS INTO RESIN ANCHORED ROOF BOLTING PARAMETERS

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ABSTRACT: Resin bolt parameters, such as back pressure and gloving, and their effect on ground support system performance, remains one of the fundamental areas of rockbolt research. The majority of previous studies into resin bolting parameters have utilised various methodologies to investigate the effect of a singular parameter. Unfortunately, due to the variability in methodologies and the relatively narrow field of study of each research project, a holistic conclusion into the exact science behind various results is unable to be drawn. It is the focus of this research project to conduct a detailed and consistent testing program, which attempts to simulate real world conditions as closely as possible, in order to provide the industry with engineered roof bolting solutions to specific underground roof properties.

Recently published studies have implemented steel piping as a simulated borehole and have reported relatively high back-pressure measurements. As part of this project, testing conducted both underground and in a cement block, have recorded back-pressures in the range of 4-10 MPa, which are substantially lower than previous tests conducted in steel piping.

Rockbolts installed in an underground coal mine using a continuous miner have been over-cored, the core has been cut into 100 mm lengths and each sample has been push tested. After push testing, the samples from the top 300 mm of each bolt were inspected for gloving. Almost all of the recovered rockbolts experienced some degree of gloving within the top 300 mm of its length. The average severity of gloving within these specimens was found to be relatively minor. It was found that gloving can reduce load transfer by 4-6 kN per 10% of gloved surface area.

INTRODUCTION

Full or partial gloving of resin anchored rockbolts was first observed in the late 1980's (Pettibone 1987), and has been an area of active research ever since. In a theoretical, perfect world, a correctly installed rockbolt will completely shred the film forming the two component resin capsule, facilitating correct mixing of the mastic and catalyst phases.

However, in reality, as the bolt is pushed up into the resin capsule, the rockbolt can act as a piston, the borehole in the rock as the cylinder and the resin as a pressurised fluid. The annulus between the rockbolt and borehole allows the pressure in the resin to be relieved as it flows down the side of the bolt. The pressure within the drill-hole is often referred to as 'resin back-pressure'. The thrust applied to the bolt causes the resin capsule to swell, split and the film to be pressed against the wall of the drill-hole, resulting in gloving of the rockbolt (Campbell and Mould 2003).

Gloving is incomplete destruction (either partial or total) of the resin capsule film at the time of rockbolt installation (Pettibone 1987). This can result in a low friction plane of weakness along the resin/rock interface, which can potentially impact on the anchorage strength of the rockbolt.

It has been theorised that resin back-pressure could lead to hydraulic fracturing of rock within the top section of the borehole during bolt installation (Evans 2015 and Campbell and Mould 2003). It has also

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been previously theorised that pressures as low as 4 MPa are capable of causing hydraulic fracturing where the minor horizontal stress is in the same order of magnitude, forcing closed joints or partings to open (Campbell and Mould 2003). The opening of these fractures could provide a route for resin to escape, leading to resin loss and reduced encapsulation. This theorised phenomena is shown in Figure 1.
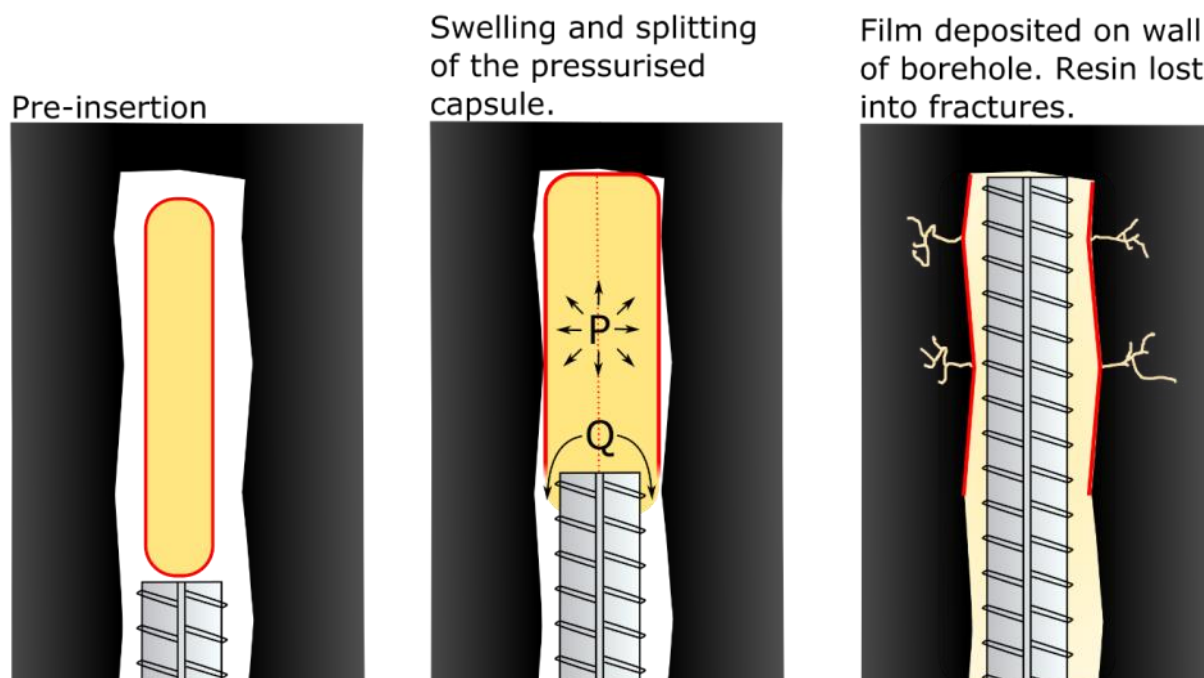

Figure 1: Theoretical pressurisation of resin capsule leading to gloving and hydraulic fracturing of the rock mass

In order to better quantify the pressures which may be required to either open existing fractures in the rockmass, or induce new fractures, reference may be made to Amadei and Stephansson (1997) equate the required magnitude of pressure of a hydraulic fracture via:

$$\sigma_1 = 3\sigma_2 + S - P_i - P_o = 3\sigma_2 - P_r, \qquad where$$

σ_1 = major principal stress
σ_2 = intermediate principal stress
S = hydraulic fracture (tensile) strength of the rock
P_i = crack initiation pressure
P_o = pore pressure
P_r = crack re-opening pressure

The orientation of the minor horizontal stress governs the orientation along which a hydraulic fracture will propagate. The majority of coal mines in Australia have a vertical principal stress (σ_v) which is relatively accurately calculated via depth. Coal mines located at a depth of 200 – 400m below the surface would be expected to have 5-10 MPa horizontal stress. Stress redistribution around mined roadways will alter the principal stresses away from their virgin condition, causing notches and reliefs in various areas (relative to the direction of the virgin stress directions with respect to the roadway). For the sake of a first pass analysis of the required pressures for crack initiation however, virgin stress conditions will be implemented in the calculations. With σ_H typically being 1.2-2.5 times the vertical stress (Nemick *et al.,* 2006) and σ_h typically being 0.6-2 times the vertical stress, and assuming tensile strengths of 1-10 MPa and pore pressure of 0 MPa (conservative), rearranging the equation above, examples of initiation pressure and reopening pressure have been calculated for real mine stress measurements, in Figure 2.

From this it can be seen that there is a vast range of pressures which can be expected to either initiate a fracture or re-open one. Whilst there is a general upward trend with depth for the pressures required for cracking, notably even at considerable depth, certain stress conditions require almost no resin pressure for crack initiation, and similarly at minimal depths certain stress conditions require considerable (20 MPa) pressures before cracks are initiated. From this it can be seen that the issue of resin pressures cannot be viewed from a 'one-size-fits-all' approach, but rather individual mines should consider their stress conditions and roof parameters before optimising the resin bolting parameters.
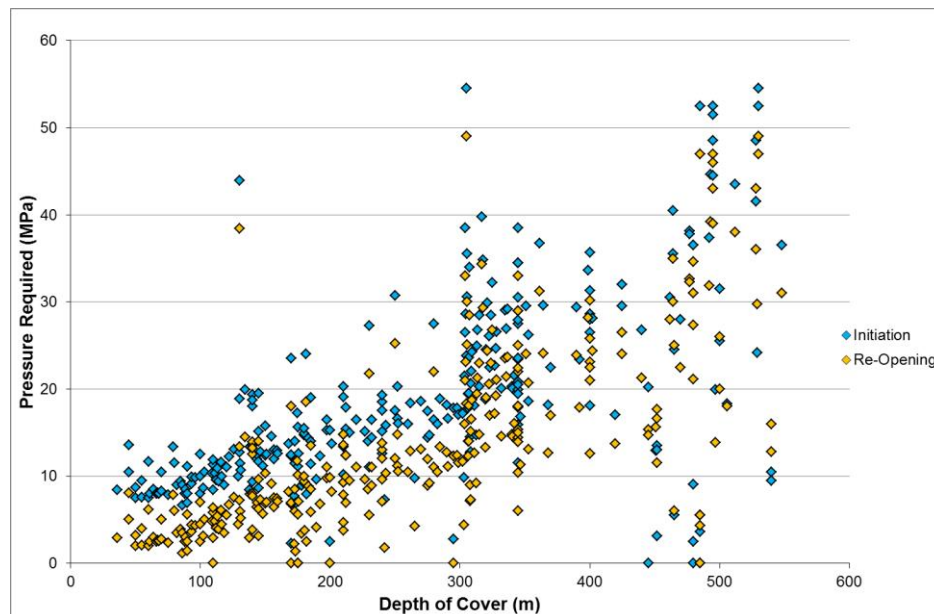

Figure 2: Pressures required for crack initiation and crack re-opening in moderate strength (5 MPa tensile) strength rock for Australian real mine virgin stress conditions

REVIEW OF PAST RESEARCH

Attempts to measure the effect of gloving on bond strength by partially over-coring gloved *in situ* bolts have shown a minimal decrease in load transfer as a result of gloving (Compton and Oyler 2005 and Craig 2012). Simulated gloving has been measured at up to an 85-90% decrease in load transfer (Mould *et al.*, 2004 and Pastars and MacGregor 2005). Different methods of measuring and expressing the severity of gloving have been used; most typically has been a general visual or external measurement of intact film longitudinally along the bolt. These range from arbitrary, qualitative assessments (Compton and Oyler 2005 and Villaescusa *et al.*, 2008), to quasi-quantitative measurements made by measuring the 'length' of gloving present (Campbell and Mould 2003). In reality, gloving interferes with the surface contact area between the resin/rock interface, and to truly quantify the amount of gloving experienced, the percentage of the surface area affected should be measured.

Recent investigations have focused on the issue of resin back-pressures and its effect on drill-hole fracturing and resin loss (Campoli *et al.,* 1999; Campbell and Mould 2003; Compton and Oyler 2005; Giraldo *et al.,* 2006; McTyer *et al,* 2014 and Evans 2015). Reported factors that affect the magnitude of resin back-pressure include drill-hole annulus, resin viscosity, cartridge film thickness (Spearing *et al.,* 2011) as well as the resin length and the rockbolt insertion speed (both upward and rotational) (McTyer *et al*, 2014). It was also suggested that the bar profile can have an impact on resin back-pressure (Evans 2015).

Several variations of testing methods have been used to investigate resin back-pressure. Rockbolt resin contains around 70% limestone particle fillers and commences reacting during bolt insertion. This makes it difficult to measure pressure using normal fluid pressure instruments. Direct pressure

measurements have been attempted by Compton and Oyler (2005) at the NIOSH testing mine in the USA using strain gauged instrumented steel pipe. Campbell and Mould (2003) reported attempts to measure fluid pressure during bolt and resin insertion into polycarbonate tube. All reported difficulty in inserting bolts into 'closed' pipes with bolts not reaching the back of the hole or polycarbonate pipes splitting during insertion. The installations do not compare to normal *in situ* rock installations. Since then, others have measured thrust force and attempted to calculate the pressure generated using the bolt or hole cross sectional area. (Giraldo *et al.,* 2006 and Evans 2015). Evans' (2015) calculations also went on to include the flow of the resin down the bolt correlated to bolt insertion speed. This research project will also seek to validate (or otherwise) the legitimacy of such calculations. A summary of previous research projects and the resultant resin back-pressure measurements is given by McTyer (2014), however importantly the measured pressures ranged from 4 to 45 MPa.

It is important to highlight that the measurements performed by researchers in the USA pushed the bolts through the resin without rotation as is standard practice in the US; rotation is only applied once the bolt is at the back of the borehole. Australian installation practice is to rotate the bolt at approx. 500 rpm during insertion up through the capsule. The cylinder pressure measurements by Compton and Oyler (2005) highlighted a sudden drop in pressure once rotation of the bolt commenced at the back of the hole. These findings raise questions as to the suitability of applying the US measurements/calculated pressures to the Australian system, as it is proposed that the absence of rotation exacerbates the pressure experienced in the resin.

To date, no researchers have directly measured the pressure generated during rockbolt and resin installation into rock.

PROPOSED TESTING METHODOLOGY AND RESEARCH OBJECTIVES

This research project attempts to provide an all-encompassing analysis of individual resin bolt system parameters (hole diameter, resin type, resin length, insertion rate and bolt profile) on key performance indicators (gloving, load transfer, rock fracture and resin loss/encapsulation) using consistent and reliable test methods, which simulate real world conditions as closely as possible; namely:

1) Measuring resin pressures of bolting systems with various parameter changes (setup shown in Figure **3**).
   a. Preparing a rockbolt by cutting a slot along the axial length of the bolt, and a recess in the top of the bolt, to accept a small load cell.
   b. Placing the load cell on the tip of the bolt.
   c. Securing cable and data logger to the drive dolly.
   d. Measuring each borehole diameter using a micrometre at 100 mm intervals.
   e. Installing the rockbolt into a ~40 MPa reinforced concrete block using a hydraulic rig (identical to that used in the underground coal industry).
   f. Directly measuring the pressure generated in the resin, during bolt installation.
   g. Logging insertion speed, rotation speed and thrust pressure of the bolting rig.
   h. Measuring encapsulation.

Figure 3: Pictures of rockbolt and pressure cell setup: a) pressure cell countersunk in head of bolt and b) wiring at foot of bolt for attachment to logger

2) Overcoring and load testing of bolting systems with various parameter changes
   a. Measuring each borehole diameter using a micrometre at 100 mm intervals.
   b. Isolating drill rig thrust hydraulic circuit and installing a pressure gauge.
   c. Performing a 'dry run' installation with no resin to determine the thrust pressure required to move the rig and rockbolt from its base position to the back of the hole.
   d. Recording drill rig thrust pressure during installation.
   e. Measuring encapsulation.
   f. Removing installed bolts and surrounding rock core from the strata by overcoring.
   g. Noting any resin migration from the borehole.
   h. Grouting the overcores into a 100 mm ID steel pipe using a TD80 cementitious grout with a seven day UCS of 60 MPa. Once cured for seven days, cut into 100 mm lengths using a band saw.
   i. Measuring push-out load of the bolt out of the rock core using an instrumented compressive load test machine.
      i. Supporting the samples on a steel block containing a 34 mm diameter hole.
      ii. Positioning the rockbolt over the hole, allowing free movement whilst providing support to the grout and rock column.
      iii. Arranging the specimen so that the applied load is oriented as if to push the rockbolt 'downwards' with respect to the in-situ orientation of the overcore.
      iv. Noting the peak load and bond strength (defined as the point on the loading graph at which the gradient drops below 20 kN/mm) of the resin for each specimen (Reynolds, 2006). The testing setup is shown in
      v. Figure 4; the cross section of a sample is shown in
      vi. Figure 5 and an example of the push test result is shown in Figure 6.
   j. Quantitatively assessing gloving.
      i. Dismantling sections using an angle grinder.
      ii. Breaking away the rock and grout with a hammer to expose the resin column.
      iii. Removing any resin that was found on the resin/rock interface and measuring its surface area.
      iv. Converting the measured surface area of film to a percentage (%) of the theoretical surface area of the 100 mm borehole length.
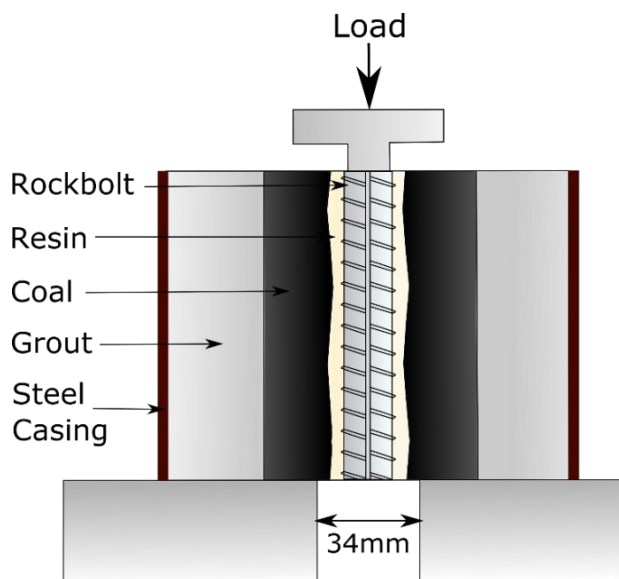

Figure 4: Push testing arrangement
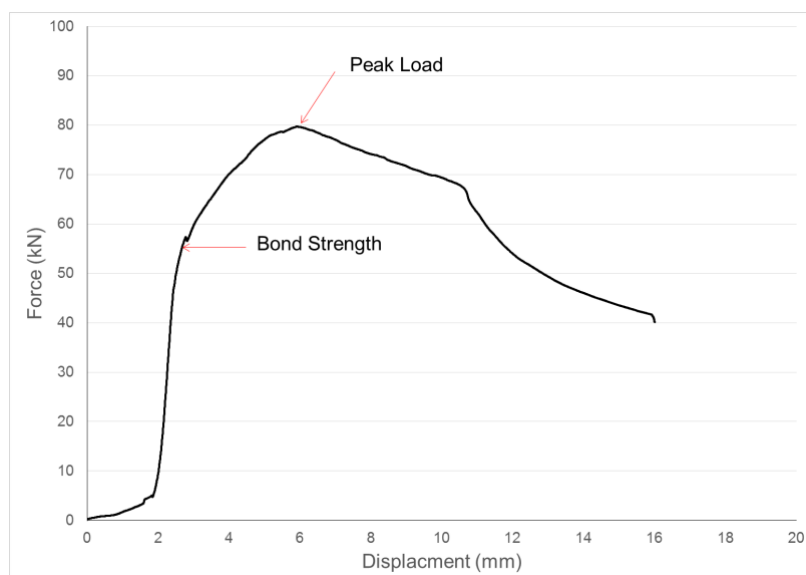

Figure 5: Push testing sample

Figure 6: Example push test result

OVER-CORE RECOVERY AND SECTION PUSH TESTS RESULTS

Over-coring has been conducted on 27 bolts at 3 test locations at Chain Valley Colliery. Location 1 and 2 had very similar roof conditions which was comprised of 1500-1600mm of coal at the bottom, and 200-300 mm of tuff at the top of the bolting horizon. Location 3 cut horizon was slightly higher, meaning that only 700-800 mm of coal was present at the bottom, and 1000-1100 mm of tuff in the top of the bolting horizon. All the installations were completed as an outbye operation, and as such, the roof in the area of the test work had time to relax, possibly producing roof separations prior to testing. This is a further area of research that needs refinement, as ideally the bolts would be installed at the face.

A 100 mm/s rockbolt insertion rate was targeted for each installation. 27, 28 and 29 mm drill bits were used as part of this investigation.

Recording of Drill Rig Thrust Pressure

Further comments regarding the correlation between the rig thrust pressure and the actual pressure in the resin are detailed later in this paper, however preliminary results from the initial 27 bolts installed indicate that:

- Larger holes didn't markedly reduce the thrust pressure required to install the bolt (see
- Figure 7)
- The thrust pressure required to install the bolt seemed to be less affected by fast insertion rate in large holes
- Resin type didn't markedly reduce the thrust pressure required to install the bolt (see Figure 8)
- Viscosity of the resin didn't markedly affect the thrust pressure required to install the bolt
- High speed insertion produces considerably higher thrust pressures than a controlled insertion rate.
- It takes approximately 40 bar (4) to raise the drill assembly, dolly and rockbolt at a controlled rate of 100 mm/s into a hole containing no resin (see Figure 9).

Considerable additional tests are required however in order to determine solid conclusions about the abovementioned parameters. Note: a) when looking at the following figures, the pressures quoted are direct hydraulic pressures for the rigs and have not been 'converted' to resin pressures, b) the graphs are for during both thrust-and-spin as well as spin at the back of the hole (with no thrust) and c) sharp

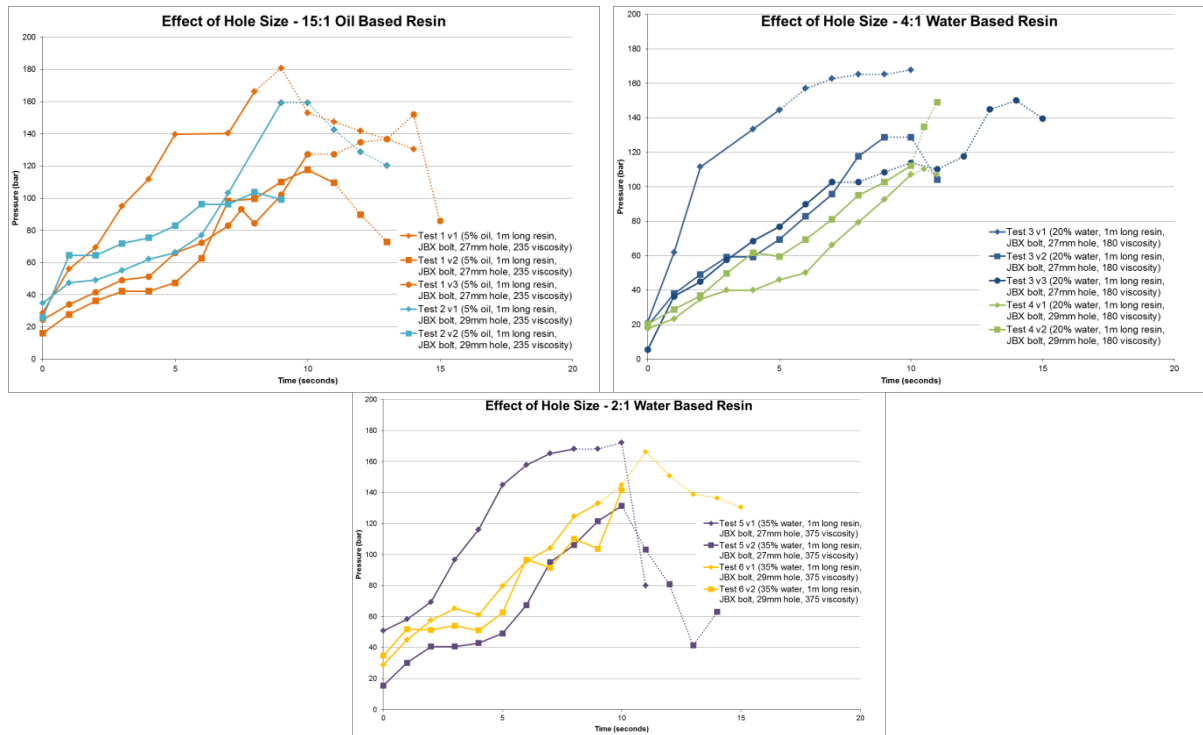rises in pressure toward the end of the installation are due to the bolt contacting the back of the borehole.

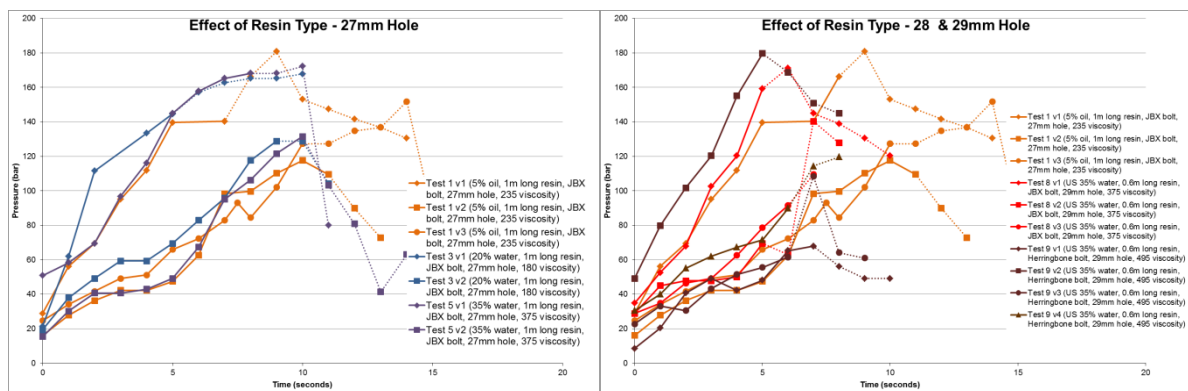

Figure 7: Effect of hole size on rig thrust pressure

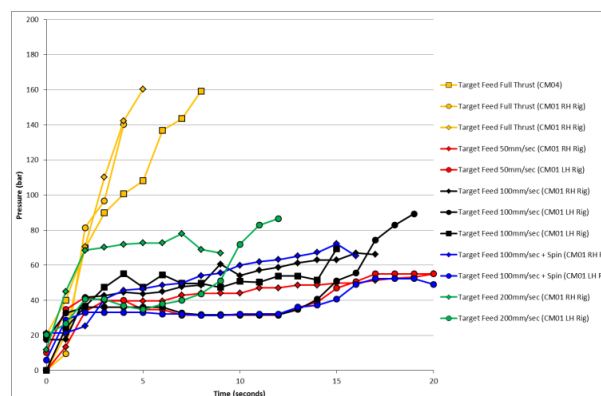**Figure 8: Effect of resin type on rig thrust pressure**

Figure 9: Results of rig thrust pressures in 'Empty Holes'

Push Tests

As almost all push tests failed at the resin to rock interface, no major conclusions can be made at this stage as to the quality of mixing or the effect of annulus size on the load transfer limit of the system, suffice to say that the 'weakest-link' of the load transfer system has seldom been found to be due to ineffective mixing or excessive annulus.

Gloving

Of the 17 bolts analysed, all but one were identified to be affected in some degree by gloving within the top 300 mm. Of the 51 x 100 mm samples inspected, 29 were identified as having some degree of gloving present.

The gloving and push test data was then collated across three horizons to measure the impact of gloving on peak load and bond strength. Specimens where rock had fallen away from the resin column during recovery of the over-core (which resulted in a grout on resin contact during preparation) were omitted from this investigation. The horizon's investigated were three 100 mm sections of the Awaba Tuff, directly above the Chain Valley Coal Seam. The peak push out loads within the tuff were around 60-65 kN, and failure was always between the resin and rock interface. The results can be seen in Figure 10, showing a consistent trend of a reduction in both peak load and bond strength with an increase in gloving percentage. Gloving was seen to reduce the peak load of a 100 mm section by 4-6 kN (8-10% of peak load) per 10% of gloving affected surface area. At this stage of the testing there is significant scatter in this data, as these results contain numerous other variations in the rockbolting system parameters (such as resin type, borehole diameter and rockbolt type), however the aim of future testing will be to gather enough quality data to obtain statistically sound relationships between the various parameters.
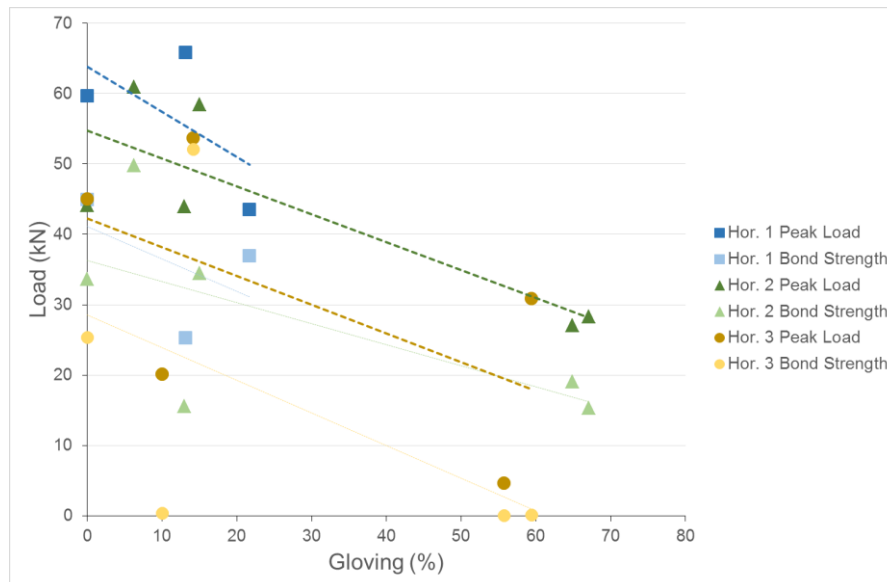

Figure 10: Peak load and bond strength vs gloving severity for horizons 1, 2 and 3

Resin Loss

An assessment of resin loss was completed for each rockbolt installed in this testing program. Resin loss was calculated by comparing the theoretical encapsulation (using borehole micrometre measurements) to the measured encapsulation.

Thrust pressures were measured to range from 40 to 180 bar. No clear relationship was found when either the peak or average drill rig thrust pressure was collated with resin loss, as seen in
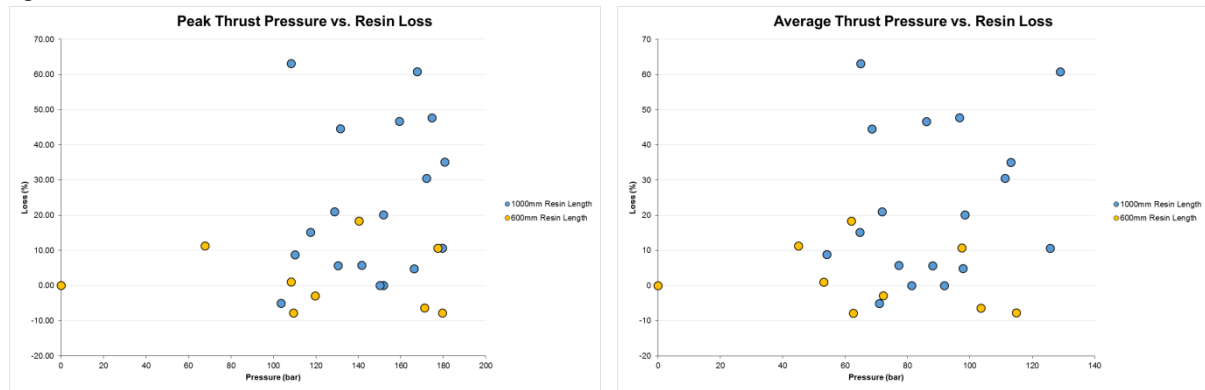
Figure **11**.

Figure 11: The effect of peak and average thrust pressures on resin loss

DIRECT RESIN PRESSURE MEASUREMENT RESULTS

Preliminary testing with this setup has provided promising results; however further testing is required to draw any meaningful conclusions. Peak pressures were observed to range from 4-10 MPa, for 'standard' resin bolt parameters (1.8m long bolts, 21.7 mm diameter, 600-1000 mm long resin cartridges, installed into holes drilled with 28-30 mm drill bits). Further testing is planned using this method and variations will be used to assess various bolt/resin/borehole combinations to identify key parameters affecting the development of back-pressure. In addition, variations in the strength of the cement block material will be used to assess the impact of rock strength on the development of back-pressure. Utilising this testing method in an underground environment would be ideal, however investigations are continuing as to how to achieve this from an electrical approval perspective.

Preliminary results show that there is minimal correlation between the hydraulic rig thrust loads and the measured resin pressure using the load cells. Example results are shown in Figure 12.
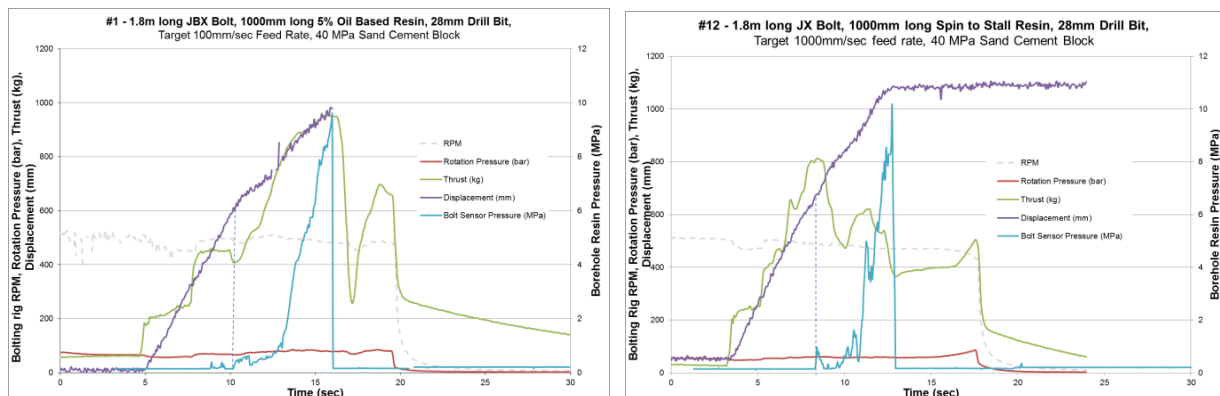

Figure 12: Results from hydraulic rig and bolt sensor loggers

Resin losses experienced in the concrete block were minimal (<10% variance from predicted encapsulation) for both 27 and 30 mm hole sizes.

CONCLUSIONS

Improved testing methodologies have been attempted in order to evaluate rockbolt performance. Measured performance parameters include not only regular load transfer capacity but also rock fracture, resin loss and gloving. The test methods have proven successful and a larger program is being developed. Some preliminary conclusions include;

- No marked reductions in drill rig thrust pressure was able to be achieved via variation of either hole size or resin type
- No correlation was able to be drawn between drill rig thrust pressure and resin loss
- Directly measured peak resin back pressures of between 4 – 10 MPa were recorded with 1.8m long Australian rock bolts and resin installed into a 40 MPa concrete block. These measurements are significantly lower than results from previous studies conducted in steel pipes.
- 16 of the 17 rockbolts recovered from underground were found to have experienced some degree of gloving within the top 300 mm of the bolt. On average, the severity of gloving was found to be relatively low.
- No correlation was able to be drawn between resin type, hole size, drill rig thrust pressure or bar profile and the severity of gloving
- A reduction in load transfer of 4-6 kN per 10% of gloved surface area was seen for push tested specimens. This equates to an 8-10% reduction of peak load per 10% of gloved surface area.

Whilst the inconclusive nature of the results so far limit the immediate benefit to the industry, they do give alternate and independent data by which to validate (or otherwise) previous claims made about various rockbolting parameters. Further research is required before the variables are able to be separated in order to decisively conclude various proposed theories.

Testing methodologies have been outlined which are suitable for the assessment of various resin anchored roof bolting parameters with the potential to provide optimisation of said parameters to suit the variety of conditions in underground coal mines.

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