KEEPING OPEN OF UNDERGROUND BLAST HOLES

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Abstract

Underground mining of base metals like copper often uses massive mining methods such as sublevel open stoping. A feature of these mining operations is the drilling of “rings” of blastholes, typically 60 – 120 mm in diameter and often over 15 m long. For convenience of equipment and personnel utilisation it is common practice for a number of “rings” to be drilled in a campaign, a long time in advance of their planned use – 12 months is not uncommon. Unfortunately in the time between drilling and charging, the holes may become distorted by changing stress fields or be blocked by dislodged rocks, both potentially preventing the hole from being charged with explosives. The blocked holes must be re-drilled at high cost and safety risk, or are abandoned, causing poor blasting results.

Recently a system has been developed to reinforce the holes after drilling so that they are less prone to blockage in the time before charging. The reinforcement consists of a thin lining of tough resin that holds loosened rock fragments in place and reduces deterioration of the hole wall. The resin is a fast-setting, two component silicate. The two components are pumped at the required ratio to a mixer and innovative head that is inserted in the hole and withdrawn at a controlled rate, while spraying the mixed resin onto the rock. It sets quickly to form a thin membrane that is strong enough to withstand deformation and spalling of the rock.

The system has been under test on an Australian nickel mine for over year, with results that indicate that the system is rugged, cost effective and suitable for the hole diameters and lengths covering all those commonly found in this industry. It will soon be made available to other users in Australia and later, in Africa.

Introduction

Many base metal mines worldwide work massive or steeply dipping orebodies that are mined by sub-level caving, sub-level open stoping or block caving. A common feature of these mining methods is the drilling of patterns of blastholes (schematically shown in Figure 1), usually with diameters in the range 60 - 120 mm. The equipment to drill holes with these diameters is large and heavy so it is preferable to drill the holes for many successive blasts in one campaign, before the drilling equipment is moved to another working place. As a result, holes may stand open for as long as 12 months until they are charged with explosives. During the interim period holes fret and close or become blocked (Figure 2) so that they cannot be charged until they are cleared. Clearing of blocked holes is labour intensive, time consuming and dangerous so mine
operators have looked for solutions. This paper describes the development of one method of alleviating the problem. It is currently (2011) under trial on mines in Australia; a number of mines in Africa have expressed interest in the system.

Figure 1. Sub-level stoping mining scheme

Figure 2. Mechanisms of hole blocking
The problem

While newly drilled blast holes are generally strong because of their circular shape, over time some holes partially collapse and can become blocked (Figure 3) to the explosives charging hoses. This is exacerbated by unfavourable geology, stress and vibration from adjacent blasting which leads to fretting of blast holes and displacement of wedges of rock, blocking the hole. Ring drilling is more susceptible to blocking than other blasting patterns such as benching, as off-vertical holes have a greater tendency to become blocked. In some instances the blockages can be quickly removed by using compressed air/water but this not always effective and usually only a little effort is expended prior to re-drilling or abandonment of the blast holes.

![Figure 3. Borehole camera view of blocked hole](image)

In either case, abandonment or re-drill, these instances will result in significant additional cost for the mine, due to disruption to normal production drilling, loss of drilling capacity or poor fragmentation and secondary blasting, in each case resulting in loss of production capacity of the mine. The clearing and-re-drilling of holes is also a strenuous operation and known to carry an increased risk of injury to workers.

The industry analysis presented below is from the Australian mining experience (Hawker 2009). The problem of closing holes is principally where mining takes place in weak or highly stressed ground and where blast causes damage. Block caving, in particular, is usually conducted in relatively weak ground, and all the undercut holes are vulnerable to damage. Undercutting is a part of mine development and not directly linked to ongoing mine production, so the problem affects relatively brief periods in the life of a mine. In the case of sub-level caving (SLC) operations, both blast damage and
stress result in very high re-drill rates. Re-drilling was estimated at around 30% (Hawker 2009), and continues at that rate for life of mine. It is thus directly influential on both production output and productivity. Additionally, many underground mines could benefit from blast-hole preservation techniques from time to time, as they mine through weak or highly stressed ground. In these cases, re-drill rate is estimated at between 5 – 15 % of originally drilled metres.

Typical drilling costs for 89 and 102 mm holes are A$25/metre and A$30/metre respectively.

The Australian underground hard-rock mining industry produces approximately 70 million tonnes of rock per year and drills approximately 7 million metres of ring holes to achieve that output. A 2009 survey of the top 18 producing mines, representing 65% of total output, gave an estimate of linear metres drilled per year of 4.6 million. Excluding the block cave mines because of their episodic drilling operations left 4.4 million metres per year being drilled in SLC and open stoping operations. Conservatively applying a factor of only 15% of primary holes being re-drilled at a cost of A$25 per metre, the estimated cost to the Australian hard-rock mining industry from blocked blast holes is A$16.5 million (R115 million). It can be expected that countries such as Canada, with similar orebodies and mining practices, will be experiencing losses of similar magnitude.

Apart from direct costs of re-drilling, blocked or collapsed blast holes are the indirect cause of other cost increases, including:

- Deviations from the designed blasting pattern and powder factor, causing under- or over-break, loss of reserves or dilution, over- or under fragmentation and secondary blasting;
- Increase in labour complement, to undertake the largely manual clearing of blocked holes. On fly-in/fly-out mines, where the mine operator bears all living and transport cost, this is a material cost.
- Increase in investment in drilling equipment, to handle the re-drilling required.

**Industry approaches to dealing with the problem**

The most common approach to alleviating the problem is adoption of a “just in time” drilling strategy. Holes are drilled for one blast only, then fired within a few days. Although this almost eliminates the need for re-drilling, it results in decreased flexibility in operational scheduling as drilling must be closely synchronised with blasting. The drilling equipment must be more frequently trammed between working places, causing reduced availability for actual drilling – and so an increase in inventory of drilling equipment.

Mines have attempted to keep drilled holes open during the drilling process by using cement slurries or additives that mix and set with the drill cuttings to bind the borehole walls but these technologies are only beneficial for down-holes and do not contribute to support of the blast hole that is often needed during the production blasting where significant vibrations lead to un-ravelling of the rock wall.
Strengthening of the hole was also recognised as a potential solution. Junction mine in Western Australia (now closed) forced slotted PVC pipes into their blastholes. This was effective in keeping the holes open. Although the cost was high, the benefits of reduced re-drilling outweighed them and the system was applied over many years; it has not found wide application however.

More recently a spray-on coating has been developed that holds the promise of offering a cost-effective, widely applicable system that is compatible with mechanised explosives loading systems.

**Spray-on rock reinforcement systems**

Since the late 1990’s, thin sprayed liners (TSL’s) have been increasingly used in the mining and tunnelling industries for protection and stability enhancement of rock surfaces (Potvin, 2002). Usually the TSL’s are applied by hand-held nozzle or robotic arm, similar to shotcreting practice. Harrison (Harrison, 2003) described remote spraying of a TSL to stabilise the rock surface of 1.5 m and 3.2 m diameter raise-bored return-air raises in the norite country rock surrounding the Cullinan Diamond Mine. Their apparatus used a rotating disk to distribute a TSL being pumped onto its periphery. An essential feature of that application was close coordination between the rates of TSL pumping and movement of the spray-disk along the length of the bore, to ensure that the desired thickness of coating was obtained. The project was successful. Remotely-applied spray linings are also used in the micro-tunnel and pipe repair industries (Trenchless World, 2011).

Minova Australia is a supplier of rock support systems (including TSL’s); it is a subsidiary of Orica Limited, a major explosives supplier to the Australian mining industry. A significant portion of explosives consumption is supplied fully pumped into the holes and Orica was therefore daily exposed to the problem of blocked holes. Recognising that the problem was one of uncontrolled rock deformation, Orica approached Minova to analyse the problem and propose a solution. The proposed solution drew on mining TSL and explosives pumping technologies: it is a “reactive” TSL (as defined by Spearing, 2002) sprayed onto the inside of the hole just after drilling. The TSL forms a tough structural membrane 2-3 mm thick that resists spalling and deterioration of the rock surface.

**The Liner System**

Prior to selection of the resin system that would be used for development work the performance criteria were defined.

The resin needed to:
- Set quickly, with an ideal set time of a 1-3 minutes post mixing.
- Gel quickly to eliminate resin loss in up-hole applications.
- Have high bond strengths to rock, generally exceeding 1 MPa in bond.
- Provide a support lining that would offer confinement to the rock surface but not be too brittle.
- Not pose significant OH&S issues for handling.
- Work in humid and wet conditions.
- Have low viscosity to enable ease of pumping, with an ideal range of 200-500mPas.
- Be compatible with a range of explosive systems available on the market.

Various resin systems were considered for use, such as Polyureas, Polyesters, Epoxies but these often matched some of the criteria but not all. The Minova range of organo-silicate resin systems was then explored as a possibility and a suitable resin system selected for initial testing. This silicate resin matched the criteria in nearly all areas and in particular it was a resin that would quickly gel, increasing from individual viscosities of 300-400 mPas to a mixed viscosity of over 90000 mPas once combined. It is a two-component resin (named Components “A” and “B”) in which the components are mixed at the spray nozzle and react rapidly to immediately form a cross-linked polymer layer.

Other properties of this resin system are shown in the table below:

Tests performed at 22°C

<table>
<thead>
<tr>
<th>Ratio A:B</th>
<th>2:1</th>
<th>1.5:1</th>
<th>1:1</th>
</tr>
</thead>
<tbody>
<tr>
<td>Punched Shear Strength (MPa)</td>
<td>3 mins</td>
<td>5</td>
<td>8</td>
</tr>
<tr>
<td></td>
<td>5 mins</td>
<td>14</td>
<td>12</td>
</tr>
<tr>
<td></td>
<td>1 hour</td>
<td>25</td>
<td>20</td>
</tr>
<tr>
<td>Gel Time (secs)</td>
<td>10</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Cure time: Penetrometer hardness test (secs)</td>
<td>33</td>
<td>26</td>
<td>24</td>
</tr>
<tr>
<td>UCS (MPa)</td>
<td>1 hour</td>
<td>70</td>
<td></td>
</tr>
<tr>
<td></td>
<td>24 hours</td>
<td>75</td>
<td></td>
</tr>
<tr>
<td>Shore D Hardness</td>
<td>2 mins</td>
<td>42</td>
<td></td>
</tr>
<tr>
<td></td>
<td>5 mins</td>
<td>55</td>
<td></td>
</tr>
<tr>
<td></td>
<td>10 mins</td>
<td>62</td>
<td></td>
</tr>
<tr>
<td></td>
<td>30 mins</td>
<td>70</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Component A</th>
<th>Component B</th>
<th>Standard</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density Kg/m²</td>
<td>1445±50</td>
<td>1150±25</td>
</tr>
<tr>
<td>Colour</td>
<td>Light brown</td>
<td>Dark brown/red tinge</td>
</tr>
<tr>
<td>Flash point ºC</td>
<td>-</td>
<td>&gt; 140</td>
</tr>
<tr>
<td>Viscosity @ 25ºC mPas</td>
<td>320-420</td>
<td>250-350</td>
</tr>
</tbody>
</table>

The advantage of the active TSL over conventional “non-reactive” TSL’s is that mixing only takes place near the nozzle so small quantities may be dispensed without material starting to set in mixers, hoses and the spray nozzle; also hardening is almost immediate.
Following the verification that the material properties would be suitable, confirmation was also required that the cured material would not react with explosive systems. A range of tests was performed on both gassed and un-gassed emulsion and also ANFO to conclude no reactivity existed between the cured resin system and these materials. (See Figures 4 and 5)

Figure 4. Compatibility with un-gassed emulsion

Figure 5. Compatibility with gassed emulsion

**Components and operation**

The development of equipment for the blast hole lining process concentrated on two key areas of a dual component pump and a spray head. Whilst a hose feeding system was also critical to the application it was anticipated that existing hose feeding systems available on the market, in particular the hose feeding systems for blast hole charging, would meet the needs of the project,
Equipment consists of:

A **two-component pump** (Figure 6) that delivers the two chemical components of the resin in a fixed ratio, to the delivery pipes. The pumping rate must match the retraction rate of the spray head to ensure a full coating of the blast hole wall and achieve a 2-3mm nominal thickness. Product consumption of one litre per blast hole metre was calculated and thus a pump with output range of 10-15 litres per minute that could deliver over 10 MPa pressure was required. The final criterion for the equipment was dual action from the pistons to ensure consistent product feed to the spray head. Trials had demonstrated that pulsing of the product feed rate would lead to gaps and inconsistency in the final liner coating. The pump is a modified version of the standard pump used for injection of resin to stabilise broken rock.

![Figure 6. Two-component pump](image)

**Spray delivery system.** The spray head needed to firstly accommodate two product lines to fit into its base but also needed to fit inside blast holes of diameters as low as 89 mm. The spray head also needed to be low weight to ensure it could easily be pushed to the back of up-holes that maybe up to 40m in length. The spray head incorporates a static mixer that mixes the product correctly in a short distance to ensure the length of the spray head was short as possible. Finally the spray tip was selected to best match the desired coating. A full fan spray tip that delivered at an angle of 120 degrees was selected. Figures 7 and 8 show the spray head assembly.
Following the selection of the required product and equipment, a series of spray trials were conducted on surface to prove the concept and that ensure a full coating at the required thickness could be achieved. A number of 90 mm PVC pipes of 25m length were installed vertically. Following this the capacity of the pump to deliver a full spray fan at 25 m height was checked then a series of trial sprayings conducted to check the spray pattern inside the pipes at various heights. During the testing it was discovered that a hole-centralizer would be of value to ensure the spray tip stayed approximately central during the spray retraction process. It was also discovered that multiple coats could be achieved to increase the liner thickness if required and that increasing the retraction speed would enable a thinner coating of approx 1 mm. Several blockages of the pipe also occurred during testing when the retraction was stopped, thus suggesting that any delay of the spray head retraction during the lining operation may cause hole blockages. An example of the spray result can be seen in Figure 9.
Mine site trials

Following completion of the surface trials and demonstration of the system to the mine customer a series of mine trials commenced. The first of these trials was the lining of 9 production holes of length 20-25 m and diameter 102 mm. Several of the lined holes would be used for visual surveying after lining to check for hole collapse whilst two holes would be used to check EP VE explosives retention. These initial trials used a nylon adaptor to connect the spray tip. This was prone to damage during the installation, which caused several spray tips to come off during trial. In the second series of mine site trials a steel adaptor was used.

The second round of trials involved the lining of 60 production up-holes again of length 20-25 m and diameter 102 mm. During these trials several other issues were noted. Firstly the use of the Orica 25mm powerhose hose pusher proved inconsistent when trying to push the spray head and product lines to 25m height. An average height of 15m was all that could be achieved. Further testing noted that the use of a larger diameter powerhose overcame the limited height and a height of 25m could be reached. The second issue was the use of a nylon body for the spray head. Although this item assisted with keeping the spray head weight low, spray pressure above 1500PSI caused leaks in the spray head and resulted in inconsistent spray coverage in several blast holes. An aluminium spray head has now been developed which enables greater working pressures to be reached. The series of photos (Figures 10, 11 and 12) show details of these spray trials.
Figure 10. Inserting the spray head

Figure 11: Lined hole
Monitoring and Results

Following completion of the initial trial it was evident from camera survey work that a full coating of resin could be achieved to the blast hole (See Figure 12). EP VE emulsion explosive retention in lined holes was not impacted and thus the 60 hole campaign was approved.

Completion of the 60 hole campaign took several days with 22 holes being lined in one 10 hour shift when there were no interruptions. Lining of one blast hole was estimated to average 5 minutes.

Monitoring of the 60 holes was initially completed with a borehole camera survey. This survey discovered some unlined sections of the borehole which were later discovered to be due to nylon spray head body pressure limit. Holes could only be lined to average height of 15m due to the limitations of the pipe pusher system.

Lined holes were gradually mined through over a 12 month period due to various production delays. Some blockages of lined holes were noted and are thought to be due to inconsistent lining in these holes. Most of the lined holes remained open and these are attributed to full coating at nominal thickness of 2 mm. Without the coating, all holes on the mine were usually closed within 6 weeks.

Trials indicated that development of an independent hose pusher and mobile vehicle would be required for full industrialisation of system; standard hose pushing equipment can be utilized if the feeder hose is of sufficient strength.
Conclusions

Initial mine site trials have demonstrated that a blast hole can be supported for a long term using a reactive resin lining post-applied to the drilled hole. To ensure ease of blast hole coating and prevent hole collapse prior to lining it is suggested that lining of the blast holes is best carried out as soon as possible after drilling. Damage of the lining was noted when inconsistent coating was applied, with a majority of this damage caused by blast vibration. Further mine trials are being planned by Minova to further develop this application technology.

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The image used in Figure 1 is by permission of Atlas Copco MCT AB, taken from the publication “Guide to underground mining methods and applications”, 1980

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Rob joined Minova Australia, previously Fosroc Mining Australia in December 1997 as Product Development Engineer. Minova Rob has been involved with a variety of projects dealing with cementitious, polyurethane and flexible membrane systems for both the metalliferous and coal mining industries. Rob spent eight years as Product Manager for the flexible membranes and injections resins product range and has been leading the Minova Australia technology team for the past 2 years.

Donald O’Connor, Director, Minova RSA

Worked from 1978 - 1981 in the SA gold mining industry in rock engineering and mine seismicity research. Since 1982 has been involved in the development, manufacture and application of explosives and rock support systems, covering resin and cement grouts, cavity filling and backfill, thin spray liners and drilling equipment.