ABSTRACT

For many years Ramp 24 at Ensham mine in Queensland, Australia has suffered substantial coal loss and damage due to the fragile nature of the coal and problematic geological conditions, which have proved very challenging. In the past various blasting techniques were applied in attempts to improve coal recovery. One approach was to use a shallow throw (cast) blast followed by a separate post-strip blast underneath. Another method was the introduction of the practice known as “baby decking” with the addition of a large buffer of material in front of the coal. However, despite increasing mining costs, none of these techniques adequately reduced coal edge loss and damage.

This paper describes how the new mining technology of Stratablast™ has been implemented at Ensham Mine to improve coal recovery and productivity. This technology employs electronic initiation systems and complex modelling to combine what were traditionally separate blast events, such as a throw blast and a stand-up blast, into a single blast event. Several layers are drilled, loaded and fired in one cycle, with a unique blast design assigned to each layer. Within each layer the blasts have different powder factors, inter-hole delays and directions of initiation. Additionally, delays of several seconds may be used between the various layers.

This technology has been used routinely in Ramp 24 at Ensham since August 2005. Several aspects of the process have been highlighted during implementation, namely; the vital importance of explosives column location, managing issues associated with the presence of large amounts of water, dealing with varying geology and ensuring blast implementation exactly followed design. All these aspects had to be well understood and rigorously controlled; on occasions when this control was inadequate some coal damage did occur.

The introduction of this technology has seen coal recovery levels increase dramatically along with providing good productivity without increasing the number of drill and blast cycles.
INTRODUCTION

Ensham Coal Mine is situated in Central Queensland, Australia. It is a large operation in the region producing around 9 Mtpa of coal. The mine removes 115 million bcm (150 million bcy) of overburden per year using four draglines and a truck and excavator fleet. While throw blasting is employed in the overburden, powder factors are relatively low. Despite this, coal loss has been a major recurring problem at the mine.

In particular, Ramp 24 at Ensham has been very prone to blast-induced coal damage and loss. Whenever throw (cast) blasting was attempted the coal edge moved. This edge movement also resulted in trenches being formed in the coal.

In Australia the extent of coal loss from mining operations varies between about 5 to 25 per cent of in-situ coal. Much of this loss occurs during blasting. Coal edge movement, ultimately into spoil, as well as blast damage to the roof of the seam and dilution with the overlying rock are frequently observed. It has been estimated that an increase in coal recovery of 1% would increase revenue by about one to three million dollars per year for most Australian mining operations (ACARP, 1995). Operations that are margin-driven are often more committed to reducing coal loss and damage as compared to mining houses which may be more volume driven and committed to high production rates. However, where both high productivity and reduced coal loss can be achieved simultaneously, the cumulative benefit is substantial to any operation.

Orica (the supplier) has developed a new mining technology that has been described previously (Goswami and Brent, 2006). It is essentially a technique of implementing multiple different blast types within a single drill, blast and excavation cycle. The method combines a throw blast with one or more stand-up blasts in a single event. The throw portions of the blasts are fired before the stand-up portions, with a separation in time in the range of 500 to 10,000 ms. This “inter-blast” delay is dictated by rock properties and the time required for the throw blast layer to fall in place. These separate sub-blasts in several layers are each loaded to a particular powder factor, separated by a column of stemming and fired on quite different initiation sequences and inter-hole delays from each other. Figure 1 shows a schematic view involving a throw blast followed several seconds later by a stand-up blast below. This is the type of blast implemented in this work to provide protection of the coal.

Usually, all movement from the throw blast has ceased well before the stand-up blasts commence. This provides a large layer of static material over the stand-up blasts to minimise any movement from these later blast portions. The technology is particularly effective at limiting coal loss, since the stand-up blasts above the coal are restricted from moving. Furthermore, the throw blasts occur in complete isolation from the coal seam, ensuring that good throw can be obtained while not allowing any coal movement. Theoretically, it would seem impossible for any coal loss to occur with this method, and in practice when designs have been well implemented no coal loss in fact results. Unfortunately however, operational realities do not always equate to idealised designs!

This paper discusses experiences with this technology in Ramp 24 at Ensham.
Previous Approaches

Various attempts have been made in the past to improve recovery of the coal from Ramp 24. Some of these are described here.

Stand-up Blasts

In order to focus on coal recovery, the mine attempted dispensing with throw blasting in favour of a more conservative approach, namely blasting only in a stand-up mode. Naturally, this resulted in extra material handling for the dragline and hence a higher cost of mining. It also adversely slowed the coal exposure rate at a critical time when market demands were high. Unfortunately, despite the productivity sacrifices, the stand-up blasts did not result in reductions in coal damage/loss.

Two-pass Overburden Removal

Another approach was to adopt a two-pass mining method for the overburden in Ramp 24. This method implemented a throw blast in the upper 80% of the overburden (35 m or 115 ft). After excavation of the throw blast a completely separate, so-called “post-strip” blast (10 - 15 m or 33 - 50 ft) was subsequently drilled and fired in the remaining overburden layer. Coal is not usually blasted in the areas considered here. The need for this second blast in the overburden once again brought about considerable increases in mining costs. Excavation of the second layer was done with a shovel and truck fleet at a considerably higher cost than could be achieved with a dragline. Figure 2 shows the blast arrangement and excavation. Furthermore, a second campaign of blasthole surveying and mark-out, drilling, loading and blasting was required for the additional blast layer. A complete additional mining cycle of drilling, blasting and excavation had been introduced. Coal exposure rates thus also decreased.

Baby Decking and Buffering

So-called “baby-decking” is a well-known technique that has been trialled with mixed success at a number of sites around Australia for many years. It is described by Kanchibotla, Laing and Grouhel (2006) amongst others. The baby-deck is a small deck of explosive just above the coal that fires on a separate delay, conventionally 50-100 ms, after the main column of explosive in the throw blast. In addition to baby-decking, a large buffer of material was constructed in front of the coal to protect it from movement during the blast. This is shown in Figure 3.

Modelling Studies

Blast models were used to compare the percentage thrown into final spoil for three different blasting methods in Ramp 24. The models, MBM and SoH, are described elsewhere (Minchinton and Lynch, 1996; Minchinton and Dare-Bryan, 2005). The blasts modelled were a baby-deck throw blast, a baby-deck throw blast with a large buffer and the new blasting method. All models were constructed keeping the bench geometry, burden, spacing and throw blast powder factor and inter-hole timing the same. Both the baby deck blasts had the same explosive loading and timing, with the bottom baby deck firing 100 ms after the top main deck of each hole.
Despite the limitations of two-dimensional models, the results compared very well to actual muckpile profiles and percentages of material thrown to final spoil. Table 1 shows the calculated percentage cast to final spoil for the three designs. From Table 1 it can be seen that the baby-decked blast (without the buffer) generates the greatest throw to final spoil, which is to be expected as the relatively short delay time for the baby deck ensures that the small bottom deck actively contributes to the throw. Conversely, with the new technology the bottom stand-up blast occurs completely separately to the throw blast with a different timing regime and hence does not contribute to throw. For the buffered baby deck blast, although the overall muckpile profile appeared to show a lot of material in the void, when the pre-existing buffer material was subtracted the actual thrown material from the blast was the least of the three blast practices modelled. Additionally of course, this material originally had to be brought into the void by the truck and shovel fleet, adding a large extra volume to the total excavation volume and making this by far the least productive option. The model showed that for the conventional cases the coal edge moved while for the new technology it did not.

Table 1. Cast to final spoil as modelled for three blast types

<table>
<thead>
<tr>
<th>Blast practice</th>
<th>Percentage cast to final spoil (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>New Technology</td>
<td>20</td>
</tr>
<tr>
<td>Baby deck (no buffer)</td>
<td>24</td>
</tr>
<tr>
<td>Buffered baby deck</td>
<td>17</td>
</tr>
</tbody>
</table>

While the exact science of coal damage and movement is not yet well understood, some general observations suggest that the following factors contribute to, or are associated with, coal damage and loss.

- Geological disturbances such as faults, folds and intrusions. These are commonplace at Ensham.

- Coal damage usually results when the material layers overlying or underlying the coal seam are weaker than the coal seam. One of the worst situations appears to be when the layers comprise saturated clay, shale or mudstone.

- The location of the control (leading) row for the throw blast. Generally the coal edge moves along the control rows, which is followed by trenches. For example, if a row-by-row throw blast timing is used parallel to the strike, the coal edge moves and trenches are formed parallel to strike. Likewise, if a “V” timing design is used then trenches are often formed at an angle to the strike direction following the orientation of the “V”.

- Blast blocks containing considerable amounts of water. It appears that energy can be transmitted between the explosive decks due to the presence of water, thereby losing control on energy release. Coal damage and movement from decks firing out-of-sequence can occur. It is also speculated that water can “lubricate” the coal seam by reducing cohesion between the coal and overburden. Coal damage has been observed to be more pronounced where coal seams are saturated by water as compared to dry sections of blasts.
• Improperly designed or inaccurately implemented stand-off distances from the coal. This is a crucial factor as explosive placed too close to the coal inevitably damages the coal. Control is made more difficult by variations in the coal or overburden as well as by blasthole “fall-back” and water ingress. Insufficient knowledge of the exact location of the top of the seam in each and every blasthole exacerbates this problem.

• Blast timing. Certain timing designs have been observed to result in increased coal movement.

• Primer location and explosive type may have an impact on the extent of damage. While these effects can be modelled, definitive field experiments have not been done in this regard.

Application in Ramp 24

The new technology was introduced into Ramp 24 at Ensham in August 2005. The first blast implemented was about 1.8 Mbcm (2.4 M cu yd) in volume and comprised over 500 blastholes of 229 mm (9 inch) diameter on a nominal 7.5 m x 10 m (25 ft x 33 ft) staggered pattern. The upper throw blast layer was about 35 m (115 ft) thick while the second, stand-up layer was around 8 m (26 ft) thick. Figure 6 shows the blast face and void. An earlier paper describes the initial blast and the highly successful outcomes (Goswami, Brent and Rutledge, 2006). The mine has since changed to 270 mm (10 5/8 inch) drill holes and patterns have also now been modified.

After the success of the first blast, this technology was routinely implemented. By September 2007, sixteen large blasts using this technology had been fired in Ramp 24.

In the majority of cases, as anticipated, the blasts produced absolutely no coal edge loss or damage, while providing productivity benefits. However, in three out of sixteen cases the operational realities conspired to ensure that theoretical perfection was not always achieved! Localised occurrences of coal damage arose in a couple of blasts where factors such as unusual or unnoticed geological formations, wet and saturated blast blocks and localised concentrations of explosives occurred, normally in combination with each other.

Typically, the bottom deck charge varies according to the design standoff distance between the bottom of the blasthole and the top of coal, as well as the rock formation (e.g. hard, soft, wet or dry) above and below the coal seam. Furthermore, each layer is blasted with a unique timing design to achieve its targeted blast result, which is different from that of the other layers. Designs in each layer differ in explosive type and deck position, powder factor, standoff distances to other layers, inter-hole and inter-row delays and direction of initiation or blast progression. Also, in order to match localised conditions and to increase or decrease blast throw, the blasthole spacing is varied in certain sections. These complexities made design and implementation challenging. It may be noted that to date no two blasts have been the same at Ensham; each has been designed to be loaded and timed differently.

In some cases the presence of a major fault dislocated and shifted the coal seam, causing the blast depth to increase to over 70 m (230 ft). The spacing in these areas had to be reduced by 30% to provide extra energy to adequately blast this particular section of the block.
The Process

The general process begins by collecting relevant information on geology, in particular the coal seam, rock layers and the presence of faults, the blast blocks, pattern, powder factors, dig rates, blast performance requirements and any environmental issues. The supplier then analyses the data and appraises the current practice, and, if required, recommendations for improvement within the general blast design are made. For example, alternative products (bulk and initiation) and delay initiation sequences (including firing direction) may well form the basis of such recommendations. The design process then commences.

Key Performance Indicators (KPIS)

Prior to designing a blast, the KPIs are usually agreed between the supplier and the mine. Generally they include the following in order of priority.

No safety or environmental incidents - vibration and over pressure levels must be within the allowable limits at particular locations.

No coal loss or damage.

Provision of a muckpile shape that maximises equipment productivity e.g. forming a ramp for dragline access and achieving sufficient blast throw to maximise machine productivity.

Maintenance of the dragline’s budget dig rates of 2011 bcm/hr (2630 bcy/hr) for the strip.

Production of a safe high wall and minimisation of material “clinging” to the wall for the dragline to clean. For Ensham mine, “Safety” is always the number one priority.

Blast Modelling

The supplier’s proprietary mechanistic blast models, MBM and SoH, are used to generate simulations on potential coal damage and on the movement of material within a blast for different blast designs. This allows the analysis of the effect of various different blast designs.

The modelling can be used to determine the effects of incremental changes in blast designs and bench geometry or more profound changes in the operation to reveal potentially more productive or cost-effective means of blasting and excavating material. Such radical changes may include changes to strip width, combinations of layers such as pre-strip with overburden or changes to wall and blasthole angles. The models generate results showing particle displacement, velocity of all particles as a function of time and final muckpile profiles as a function of time. The latter is important in optimising timing design, in particular the delay required between various blast layers.
Block Definition

All the free faces, bench surfaces and voids need to be surveyed. This information is necessary to create various surfaces used in the blast design process. The mine surveyors usually provide all details such as crest lines, the top of the coal seams visible in the exposed highwalls, voids, low wall, and relevant information of current and previous strips within the strip. The supplier’s proprietary SHOTPlus-i™ program is then used to recommend blasthole locations with appropriate burden, spacing and hole depths.

Strata Logging

Once the block is drilled, Gamma logging is carried out in a defined sequence to accurately determine rock strata changes throughout the blast block. This key process identifies material layers overlying and underlying the coal seams as well as the exact location of the top and bottom of the coal seams. Approximately 20 holes per 100 m (330 ft) of block length are logged. This information is crucial for success as it affects all design parameters.

Loading Design

A detailed design specifying each explosive and stemming deck is then generated. The various positions of each deck in the blastholes and their column lengths, explosive masses and primer locations are provided to the shotfirer. This design is then implemented, as accurately as field conditions allow, by the experienced mine and explosives teams.

Initiation Design

Information gathered from the modelling analysis, the actual as-loaded explosive and stemming deck locations, any environmental restrictions and the required muckpile shapes are then considered in developing the timing design. These complex blasts cannot be performed using signal tube systems as the system flexibility of the electronic i-kon™ initiation system is required. This includes the provision of very long and accurate delays. Accuracy and controlled timing energy release are important in achieving the desired blast results. Also, the multiple delay sequences used in the various layers can only be achieved with electronic detonators.

A key consideration in the timing design of the throw blast part of the blasts is that of environmental constraints.

Environmental Issues

Initially, airblast and vibration were not considered to be high risk factors in the design of the first blast in August 2005, as the monitoring locations were more than 5 km (3 miles) away. While ensuring no coal loss was the major blast objective, the throw blast initiation design was aimed at achieving good throw outcomes. This design used short inter-hole delays along the control row, providing timing of about 0.6 to 0.7 ms/m (0.55-0.64 ms/yd) along most of the 650 m (710 yd) long face. This blast recorded high overpressure levels, in excess of the allowable limits at the mine’s monitoring station. The blast
video showed some evidence of face bursting and stemming ejection, however, subsequent analysis of the airblast trace indicated that these were not dominant contributors to the peak level.

The supplier has since developed a new model for prediction of airblast based on the effect of blast face movement. This model is based on the blast face moving the air in front, resulting in overpressure (Blair, 2004). The overall face motion is dependent on the delays and spacing used between face holes as well as the explosive column height, rock and burden in each hole. Since its development this model has been used for all the subsequent such blasts fired at Ensham and none have exceeded allowable limits again. It was also established that the use of two face sections initiating in opposite directions in throw section could actually increase airblast levels at the monitoring station.

Field Implementation

The importance of the design steps has been emphasised, however the actual implementation of the designs in the field is equally crucial. The quality of the outcome can only be as good as the quality of implementation. The coal and rock will move according to the actual explosive charge placement, not the theoretical design!

Utmost care and attention to detail is required to implement the design successfully in the field. Control of charge masses and column lengths, air/stemming deck locations and lengths and actual stand off distances are necessary to avoid any coal damage or loss. Accurate (calibrated) stemming delivery is a key requirement.

Results

Coal Loss and Damage

One of the important benefits that this technology has brought to Ensham mine is the significant reduction of coal edge loss and the resultant trenches. In almost all the blasts with the new technology the results have been very good. An example of the ability to recover coal with localised variations is shown in Figures 7 and 8. Figure 7 shows a pre-blast model of the coal surface and Figure 8 shows the same coal surface after it was exposed by the dragline. A localised fold in the coal is visible in both figures. Figure 9 shows the general quality of coal surfaces with this technology.

Table 2 shows the comparison of recovery of the complete coal seam against the in-situ coal model for the last three strips at Ensham. The increased recovery was due to the reduced coal loss/damage as a result of this technology. This far exceeds what can be expected from conventional blasting, where 5-25% loss of coal is often experienced in Australian coalmines.

<table>
<thead>
<tr>
<th>Strip</th>
<th>25</th>
<th>26</th>
<th>27</th>
</tr>
</thead>
<tbody>
<tr>
<td>Expected coal as per mine model including % dilution/loss allowed by Ensham</td>
<td>454243 t</td>
<td>417658 t</td>
<td>388939 t</td>
</tr>
<tr>
<td>Actual recovery</td>
<td>460048 t</td>
<td>436466 t</td>
<td>391468 t</td>
</tr>
<tr>
<td>% Recovery</td>
<td>101.3%</td>
<td>104.5%</td>
<td>100.7%</td>
</tr>
</tbody>
</table>
When Things Go Wrong

Of sixteen such blasts fired so far in Ramp 24 south and north, one blast did however result in significant coal edge movement and damage. Various factors were believed to have led to this situation, including the unnoticed presence of a geological change, misinterpretation or insufficient logging data and excessive water which transmitted energy, displacing the coal seam horizontally into the low-wall. There were two other cases of some localised coal damage, again due to unusual field conditions or inaccurate implementation of design.

It is extremely important to locate any coal and geological variations, identifying locations of hard or soft layers immediately above or below the coal seam as well as the amount of water in each hole. It has been observed from video records that the overburden appears to rupture along fault lines, which can lead to disruptions in the stand-up portion of blasts and damage to the coal. This venting of energy is also believed to have caused insufficient breakage giving rise to hard digging in these regions.

When things did go wrong it was due to a combination of field circumstances. These included geological abnormalities, excessive water, restrictions on firing direction due to the presence of a nearby residence, “fall back” of drill cuttings, short holes, inaccurate backfilling and last minute changes to the firing direction due to operational requirements.

Muckpile Shape

Typical designs in Ramp 24 aim to throw the top 35 m (115 ft) of overburden to achieve maximum displacement into the void. However, due to restricted blast geometry (35 m (115 ft) instead of the full bench height of 45 m (150 ft)) the throw is usually limited to about 25%. However, the designs are done in such a way as to achieve forward displacement of the muckpile centre of mass, which helps in the initial phase of excavation; namely dozer push. Dozer push of the upper layers of the muckpile into the void can be cheaper than dragline excavation. The dozer also prepares the pad for the dragline. Figure 10 shows a muckpile during dozer push operations.

Dragline Dig Rates

One of the key performance measures for the Marion 8050 dragline (40 bcm (52 bcy) per swing capacity) is to achieve more than the mine’s budgeted dig rate, which is 2011 bcm/hr (2630 bcy/hr). Any improvement on this dig rate is directly proportional to extra dragline days made available for the mine to utilise. Extra dragline days translate into the coal exposure rate, which is taken as the best measure of productivity. The coal exposure rate is a function of both the volume to be excavated by the dragline and the actual dragline excavation rate.

It has been estimated that the opportunity-cost of lower coal exposure rates in the current market conditions, taking account of the operating and capital costs for the Marion 8050 dragline, equates to about A$11,000/hr of dragline time. This translates into a very significant opportunity-cost of approximately A$250,000/day. In strips 25, 26 and 27 the number of days saved was estimated to be 16. The total savings due to the dragline productivity alone was thus worth A$4 million just for these three
strips. In the most recent blast in strip 28, four days of dragline operation was saved after only digging half of the muckpile.

**Highwall Quality**

It was established at Ensham that the resulting highwalls were generally good in northern parts of the ramp where the major joints were parallel to the strike direction. However, in the southern parts of the ramp the face orientation was turned, changing the direction of major joints to an angle of about 30° to the strike. Less clean highwalls, with material “clinging” on to the wall, started to result in the south. This made the dragline less efficient. The problem has since been addressed by increasing the explosive energy in the last row adjacent to the presplit line and modifying the timing in the back three rows.

**Concluding Remarks**

To date over 24M bcm (31 M bcy), producing over 3 M t of coal, have been blasted in Ramp 24 with this technology. It is now a routine mining method at Ensham. Careful design and field implementation are always required to ensure success, since changes in geology or actual explosive loading can lead to coal damage or loss. The supplier and the mine continue to work together with this mining method, which has proven itself in reducing coal losses while maintaining high productivity in all aspects of mining in Ramp 24 at Ensham mine.

**Acknowledgements**

The authors thank and acknowledge many people from the mine and from Orica Australia Limited for their assistance during this work. In particular, Ensham Resources are acknowledged for their co-operation and permission to publish. Particular thanks go to Simon Wedgwood and Malcolm Christie from Ensham mine. Dane Blair and Peter Dare-Bryan are thanked for their modelling and Keith Gimbert, Dirk Smith, Trevor Byers, Alex Steciuk, Stephen Mansfield, Clive Leeds and others in the Orica Ensham team are thanked for their immense efforts in blast implementation and success.

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Figure 1. Schematic view of the new blast technology for reducing coal loss, showing a throw blast with subsequent stand-up blast.

Figure 2. Two-pass blasting with dragline and truck and shovel excavation.
Figure 3. A cast blast with baby deck and buffer to restrict coal edge movement. Despite best efforts, once again these methods did not eliminate coal edge loss. With the edge movement, trench-like damage occurred in the coal, as shown in Figures 4 and 5.

Figure 4. Edge movement into spoil and trenches in the coal with conventional blasting.
Figure 5. An example of formation of deep trenches in the coal. The trenches are filled with water in this picture. This was result of a conventional cast blast.

Figure 6. The blast face and void for the first blast with the new technology in Ramp 24.
Figure 7. A pre-blast model of the coal surface showing a fold in the coal.

Figure 8. The post-blast surface after the dragline exposed the coal, showing the recovered fold in the coal.
Figure 9. Post-blast coal surfaces after dragline exposure.

Figure 10. Dozer push in preparation for dragline excavation.