Factors affecting the quality of tunnel infrastructure at Premier Diamond Mine

by M.S. Smith*

Paper written on project work carried out in partial fulfilment of B.Eng (Mining Engineering) degree

Synopsis

The condition of tunnels at Premier has deteriorated to such an extent that an objective viewpoint on what leads to poor condition was required. Poor and untimely development results in schedule deviation, rehabilitation, and increased working costs. The project focused primarily on quality, and not on efficiency.

The project gave management guidelines that need to be focused on in order to produce quality infrastructure. The conclusions were reached by doing literature searches, underground site investigations, benchmarking and computer simulations.

From the results we can see that, in order to produce good quality tunnels, we have to have the basics working. This was found to be one of the major problems. Firstly, tunnel profiles are not to the design specification and no fixed blast designs existed. After revising tunnel profile statistics, overbreak increased to 10% (previously 7%) and development metres were 50 m short of target. Blasting was not monitored and designs not optimized for the conditions. Furthermore, geological factors are the main contributor to tunnel failure, although this process is initiated and aided by poor development (overbroken tunnels, incorrect profiles, direction and location) and blast damage. Continuing to develop tunnels in the current manner will cost R 1.5 million in additional costs for loading and supporting overbreak, whilst an additional 1 800 man-shifts will be required.

Blast patterns were designed as a starting point for trials, while the implementation of a blast management system, development statistical database, tunnel hand-over procedures and a revision of the bonus system were proposed.

Introduction

Mine background and general information

The Premier kimberlite pipe is located some 37 kilometres northeast of Pretoria. It is the largest known kimberlite pipe in South Africa, and is one of 11 kimberlite diatremes found in the Cullinan-Rayton area. Since 1902 a total of 326 million tons of ore has been mined, yielding 113 million carats at an average grade of 35 carats per hundred tons.

The Premier pipe is an elongated oval shape, with the east-west axis approximately 900 metres long and the north-south axis 450 metres on surface. It has a surface area of 32 hectares, decreasing progressively with depth.

The mining method is also known as mechanized panel retreat block caving. In this method, a drilling level is developed to allow the orebody to be undercut by drilling and blasting. When a sufficiently large area has been undercut, continuous caving initiates. Mining operations on the undercut level also include long-hole drilling and charging, blasting and tramming of limited amounts of ore.

A production level is situated 15 metres below the undercut level, and tunnels into the orebody are developed on this level. Out of these tunnels, drawpoints are developed. These drawpoints are then raise-bored, and a drawbell developed so that the caved ore falls into the drawbell and flows into the drawpoint. The ore is then loaded by load haul dumpers (LHD) and trammed to passes outside the orebody. On the extraction level, 43% of the rock is extracted to create the drawbells, production tunnels, and crosscuts needed for mining.

Two blocks are currently mined, BA5 and BB1E. Block BA5's undercut level is 615 m below surface, with its production level at 630 m below surface. Block BB1 East's undercut is 717 m below surface and the production level is 732 m below surface. Because of severe stresses on some of the production tunnels on 732 Level, some of the tunnels have been collapsing. In order to mine these areas, another production level, 747 Level, was introduced.

Project background

Premier mine has conducted many projects on development and support efficiencies and their outputs, but never a project focusing on quality and the integration of the two.

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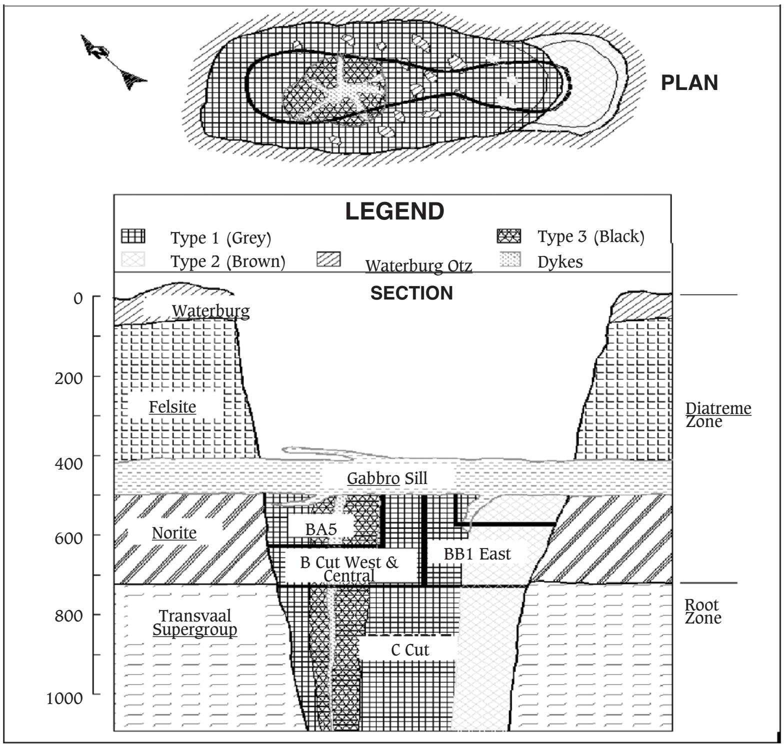

Figure 1— Generalized geology of Premier pipe

Poor and untimely underground tunnel infrastructure translates into re-work, schedule deviation, and incorrect design specification for succeeding processes, which leads to additional costs. The planning/development/support interface is mismatched, resulting in tunnels being developed too early and support being installed too late, which leads to tunnel failure and increased rehabilitation costs.

The quality of the underground tunnel infrastructure at Premier Mine has receded to such an extent that an objective viewpoint is required on the problems of quality. The deliverables from the project will guide and focus management on the critical areas of tunnel infrastructure. The driver of the project is quality and not efficiency, unless an improvement in efficiency will drastically improve the quality. Finally, this report will present a cost analysis on these factors.

Scope of the study

The scope of the project is to primarily look at the design and implementation of tunnel infrastructure. On the second level, the project specifically focuses on the development of the tunnels, whilst using various departments (e.g. geotechnical, survey etc.) as leverage points.

Methodology

A brief literature study was done focusing on other mines and civil excavations, which have experienced similar problems, and the means by which they measured and controlled these problems.

A number of underground site investigations provided the author with a good understanding of the conditions and the processes, which are used in order to develop tunnel infrastructure. Measurements and photographs were taken from the development cycle, with the focus being on before and after effects. The major factors that were observed were typically marking, drilling, charging and loading, with slightly less focus on support installation. This data was then reconciled and analysed in order to identify problems and make appropriate recommendations.

A statistical analysis on the data recorded underground, as well as the data received from survey, allowed numerous graphs and tables to be drawn up, which visually interpreted problem areas. The areas identified were location of development, rock types, overbreak, and underbreak.

Computer simulations were used in order to test the blast designs, both those used underground as well as the new designs proposed. These simulations were done in consultation with AEL's blast consult team.

Observations, measurements and data collection

Geotechnical conditions

A monthly geotechnical survey is carried out in order to assess and monitor the geotechnical condition of each tunnel. This provides management with information on problem areas and the major influencing factors.

The geotechnical conditions are described using four parameters, namely, overall condition, water, stress, and LHD damage. Each is described in Tables I to IV (Singleton, 2002).

Figure 2 shows the average rating (June to November 2002) for each of the mining levels over a period of six months. The survey has only been done for six months and in this time there is no noticeable change in the condition of

Table I

Overall condition rating

Condition rating	Condition	Description
1	Good	Area has no apparent areas of concern
2	O.K.	Shotcrete cracking
3	Medium	Shotcrete falling, small pieces falling away
4	Problem area	Shotcrete breaking—movements
5	Poor	Area at point of collapse

Table II

Water condition rating

Water rating	Condition	Description
1	Good	No water present
2	O.K.	Damp areas of shotcrete
3	Medium	Wet shotcrete
4	Problem area	Dripping
5	Poor	Heavy dripping
6	Very poor	Water flow leading to erosion of area

Table III

Stress condition rating

Stress rating	Condition	Description
1	Zero	No deformation of tunnel
2	Low	Shotcrete cracking
3	Medium	Large movements
4	High	Collapse

Table IV

LHD damage rating

LHD damage rating	Condition	Description
1	Low	Surface damage
2	Medium	ARMCO deformed, pulled away
3	High	Major deterioration of the bull nose and camel back

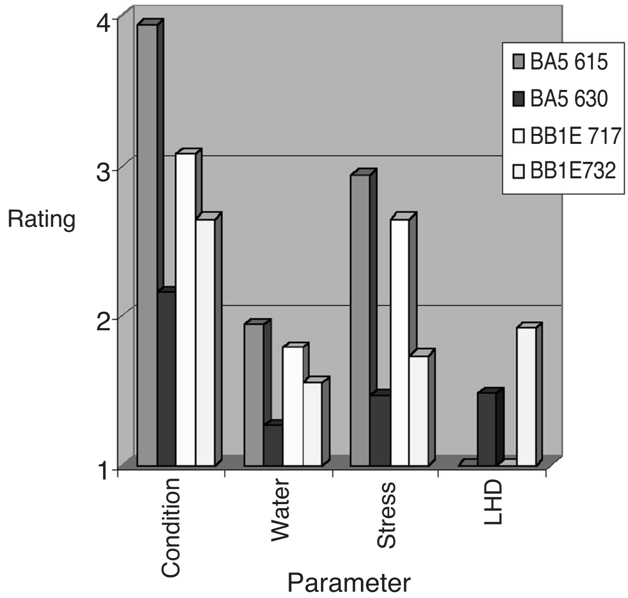

Figure 2—Results from geotechnical survey

the tunnels and therefore the averages are displayed. The survey will be continued and once enough data has been collected, the results should allow for the prediction of the life of tunnels under various conditions.

The major geotechnical factors, namely water, changing stress conditions and weathered rock, are the primary cause of tunnel failure at Premier, although this process may be facilitated by poor development.

Survey statistics 2002

From the information given in Table V it appears that development conforms with mine targets and overbreak limits.

The mine makes use of an 'equivalent metre system' (Equation [1]). This system compares development from any size tunnel to that of a 4 x 4 metre tunnel. This allows bonuses to be calculated from a standard system.

Table V

Development summary 2002

Development (m)	Blue	724
	Rim	806
	Total	1530
Sliping (m^3)	Blue	658
	Rim	2064
	Total	2722
Tons	Blue	42154
	Rim	45335
	Total	87489
Overbreak		7%
Equivalent metres	Advance	1867
	Sliping	181
	Total	2048
Target		2007

$$L_1 \times \left(A_1 / 15.04\right) = L_2 \quad [1]$$

Example

A 4.2 x 4.2 metre tunnel is advanced 10 metres. The area of the tunnel is 16.68 m^2.

By using Equation [1]

10 m x (16.68/15.04) = 11.1 m

By using volume to equate the tunnels, the equivalent advance on a 4 x 4 m tunnel is 11 metres.

Overbreak calculation

Overbreak is defined as the volume of rock broken divided by the volume of rock planned less 100%. This value defines how much larger the excavation is than the planned excavation. The allowable overbreak is ten per cent.

Revised development statistics

After reviewing the development statistics, two errors were found: firstly, there was an error in the calculation of the equivalent 4 × 4 m tunnel, and secondly, the profiles developed underground were not the same as those used by the survey department.

The calculation error was a result of multiplying the height and width (giving the area of a square profile) to get the area and dividing this by 15.04. Although the targets are calculated correctly, the survey measurements now include the area of the chamfered corners as development metres. The variance is shown in Table VI

The error in the calculation was carried throughout the calculations, resulting in tonnages, equivalent metres and overbreak percentages being incorrect. This has further implications, resulting in the cost analysis for the section also being incorrect. Before any analysis could be done, the effect of these problems had to be taken into account in order to determine the size of the error.

In summarizing the new values, two approaches were taken: firstly, to correct only the survey calculation and, secondly, to correct both the calculation and counter for the tunnel profiles. The summary is shown in Table VI. One can see that even though the target for the year was calculated on the incorrect profiles, there is a shortfall of 49 m when the calculation is corrected. The target for the correct profiles can be equated by changing the factor to divide by 14.28 (area of 4 × 4 m horseshoe profile). It is now shown that there is a shortfall of 52 m and the overbreak has increased to 11 per cent (previously 7 per cent).

All further calculations relating to development statistics are now based on the revised values using the corrected profiles.

Overbreak summary

The summaries of the tunnel sizes are as follows:

- 52.8% normal (<10%)
- 41.7% overbreak (>10%)
- 5.6% underbreak (<0%).

Table VI

Revised development statistics

		Corrected calculation	Corrected profiles
	Advance	1530	1530
Sliping (m^3)	Blue	658	658
	Rim	2064	2064
	Total	2722	2722
Tons	Blue	39266	37879
	Rim	42979	41435
	Total	82248	79314
	Overbreak	9%	11%
	Advance	1777	1867
Equivalent meters	Sliping	181	191
	Total	1958	2058
Target		2007	2110
Variance		-49	-52

The summary shows that minimal tunnels are underbroken, while there is an exceptionally large proportion of tunnels that are overbroken

By mapping the rock type to the relevant development areas, we get the following results:

- Norite – 10% overbreak
- Grey kimberlite –10% overbreak
- Brown kimberlite –9% overbreak.

From these results it is safe to assume that there is no distinct problem in any one type of rock but rather in al. This led to an investigation into the blasting practices.

Underground site investigations—drill and blast practices

Presently on the mine there are no set drill and blast designs for the various tunnel types and profiles, although there are certain guidelines, defined by the drill and blast engineer, which the development crew must follow.

Eight development cycles were monitored. For further validation of the results miners were consulted, as well as the AEL team that carried out trials on SmartDETs (van Greunen *et al*, 2002). Figure 3 is a typical example of the blast pattern utilized on 4 × 4 m and 4.2 × 4.2 m tunnels. The illustration is taken from a photograph of a 4.2 × 4.2 metre end being developed in Norite. The number of holes in these patterns varies from 55 to 61.

Firstly the face shown is marked at a 4.6 m width, already 0.4 m wider than planned. The face has 61 holes drilled at 43 mm in diameter and 3.3 m in depth. One hole is left uncharged in the five-hole cut. The remainder of the grid holes are charged to two-thirds (approximately 3.2 kg) with Anfex. Smooth wall blasting is done to the grade line and is charged with Energex Barrel. The holes are primed with Powergel 813 (25 × 200 mm) and initiated with a capped fuse. Capped Fuses are making way for electronic detonators that are to be implemented shortly.

Analysis and evaluation of research/investigation results

Review of factors affecting overbreak and underbreak

The factors influencing overbreak and underbreak can be categorized into two major categories, namely the geological factors and the drill and blast practices.

Geological factors

These factors cannot be modified and are related to the geological properties of the rock mass, namely joint orientation and joint spacing.

Overbreak and underbreak are greatly influenced by the orientation of joints relative to the perimeter of the tunnel (Ibarra, 1996). Typically less overbreak and underbreak is found where faults and joints strike nearly perpendicular to the tunnel axis, and more are found when these features are parallel. When joint sets run almost parallel to the tunnel axis, it is found that the rock tends to break along the joints rather than the line intended. Overbreak can be expected to increase with the combination of two or more joint sets.

Joint spacing (block size) is the most important characteristic of jointing for overbreak and underbreak. Intensely jointed rocks tend to be difficult to blast, while massive rock is easier to excavate in neat lines.

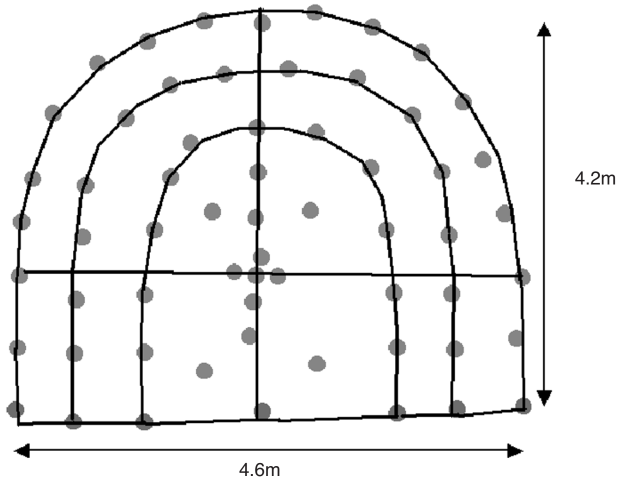

Figure 3—Typical blast pattern marked underground

Blasting factors

Poor blast results can also be a result of a poor design or poor implementation of a good design. Typically the perimeter holes cause the most amount of damage, although the cut may also cause damage but to a lesser extent. Care has to be taken when marking and drilling that this is done as accurately as possible.

Marking

Poor marking is physically marking the incorrect profile or incorrectly positioning marks for holes. Poor positioning of holes can cause holes to be overburdened, resulting in explosives not removing the required rock to create a free face for the next charge. Therefore the explosive will cause excessive fracturing of the rock around the hole.

Drilling

Where possible the operator must ensure that he drills on the marks, for the same reason as above whereby poor drill hole positioning can result in overburdening of holes. Sometimes it is not possible for the operator to drill on the marks because of the condition of the face. Sometimes with the orientation of the joints the blocks do not necessarily break perpendicular to the tunnel axis. Drill hole deviation is a problem that can be kept to a minimum if the operator takes care not to apply too much force to the bit. If the drill steel bends, the hole tends to deviate and this can cause the toe of the hole to be overburdened.

Charging

The charging crew must charge according to the charging plan in order not to overcharge the holes. Excessive use of explosives may cause damage to the surrounding rock as well as increase explosive costs. One might find that when problems are experienced underground, miners will charge the face with far more explosives to ensure that they get the maximum advance. Adherence to blast designs will minimise overbreak, whilst at the same time optimizing advance.

Overbreak history

Figure 4 shows the history of overbreak on the mine during the period 2000 to 2002. From June 2001 there has been a sharp increase in the amount of overbreak to above 10%. During this period a new drill rig was added to the fleet. This rig drills 3.3 m rounds and utilizes a 43 mm drill bit, while the older rigs use 2.7 m drill steel with 38 mm bits. It is possible that whilst implementing the new drill rig, blast patterns were not changed to match the drill bit. Therefore an additional 0.3kg/m of explosives is added to each hole. The average powder factor calculated from the statistics and those calculated from the observations underground are closely matched.

Blast Design

Improving blast designs

By recording blast results and constantly monitoring these

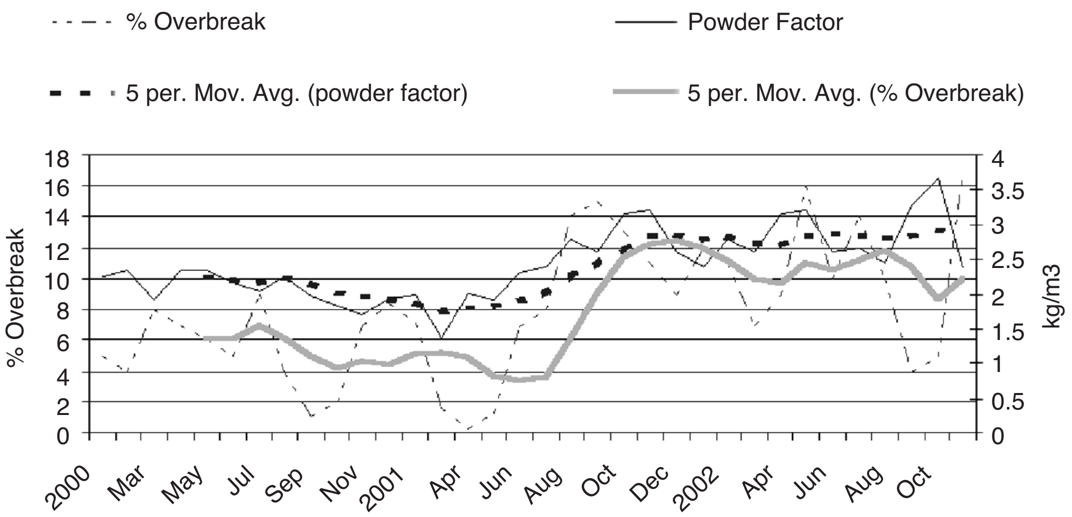

Figure 4—Percentage overbreak vs. time

results, it becomes clear when problems are encountered (Onderra, 2001). If appropriate action is taken and blasts modified accordingly to improve the results, we can expect to:

- ➤ Improve the quality of the tunnels produced, in terms of shape and size, which will reduce the amount of overbreak and backbreak, whilst also preventing underbreak
- ➤ Improve explosive and energy usage in breaking the rock
- ➤ Increase the half cast factor. (This is calculated from the remaining barrels, which represent the amount of damage to the tunnel)
- ➤ Optimize the advance
- ➤ Reduce overall working costs.

Blast simulations

Figure 5 shows the simulation (simulated with JK Simblast) of a typical 4.2 × 4.2 m face. The simulation is done with a 43 mm drill bit drilled to a depth of 3.3 m. There are a few things to note: firstly around the cut area, the distance from the cut to the inner easers shows that the holes are possibly overburdened. Secondly, the outer easers (outer row of grid holes) are overcharged causing damage to the perimeter whilst also wasting energy and explosives.

The proposed blast design (designed with AEL's Tunnel 2000) for a 4.2 × 4.2 tunnel is shown in Figure 5. This design makes use of a 9-hole cut rather than the 5-hole cut as shown on the previous pattern. This should ensure that there is a free face and that the outer easers are not overburdened. The concentration of the energy can be seen to be around the cut area, as this is where most of the explosives are required in order to break a free face. Furthermore, it can be noted that there are fewer outer easers, thus meaning that the perimeter powder factor is now reduced and that this area is not overcharged.

Table VII summarizes and compares the current design with that of the proposed design. These designs should produce better quality tunnels in terms of size and shape, as well as reducing the amount of damage to the surrounding rock mass.

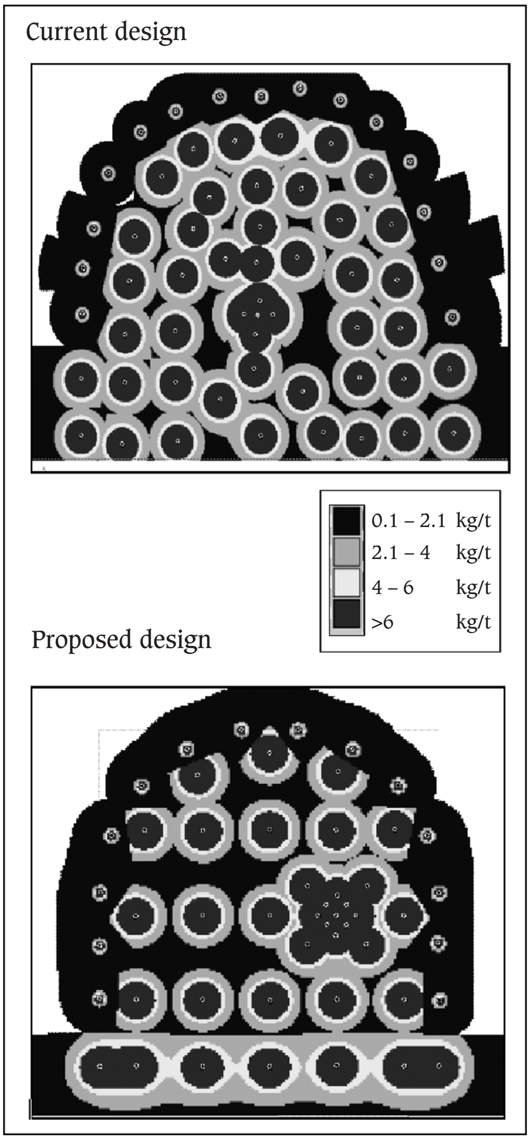

Figure 5—Blast simulation

Cost and time saving

Fewer holes in the design will reduce drilling costs and save time

Development costs per unit:

- ➤ R2.05/kg Anfex
- ➤ R0.59/cartridge Powergel 813
- ➤ R53.00 SmartDET
- ➤ R5.50 per metre Drilled

The time saving is calculated using the difference in the number of holes between the two designs. It is assumed that the time to set up the rig as well as the time to move the boom between holes will not change.

Average penetration rate	= 1.4 m/min
Charge hole	= 2 min/hole
Drilling	= (3.3 m x 10)/1.4
	= 24 min
Charging = 14 x 2	= 28 min
Total time saving	**= 52 min**

Table VII

Comparison between current and proposed design

	Current	Proposed
Holes (43 mm)	61	51
Charged holes	60	46
Overall powder factor (kg/m3)	3.44	2.8
Perimeter powder factor (kg/m3)	3.1	2.1

Table VIII

Cost analysis between current and proposed design

	Current		Proposed		Saving
	Units	Costs (R)	Units	Costs (R)	(R)
Anfex (kg)	192	393	147	301	92
Powergel 813	60	35	46	27	8
SmartDET	60	3180	46	2438	742
Drill holes	61	1107	51	925	182
Totals		4715		3692	1023

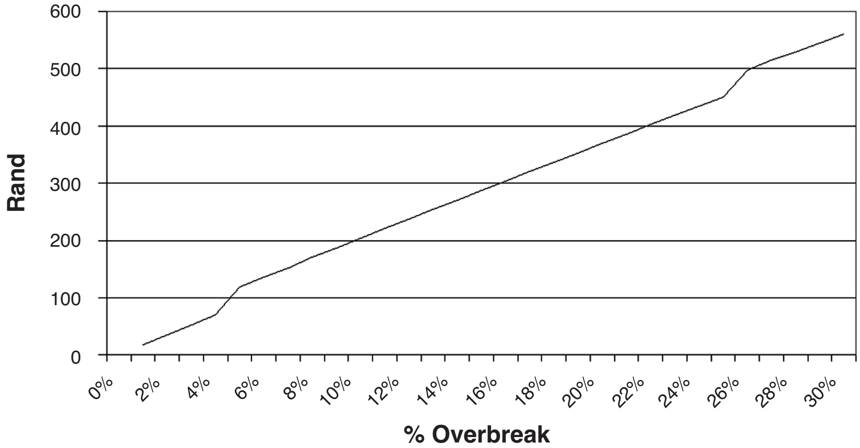

Figure 6—Cost/meter vs. per cent overbreak

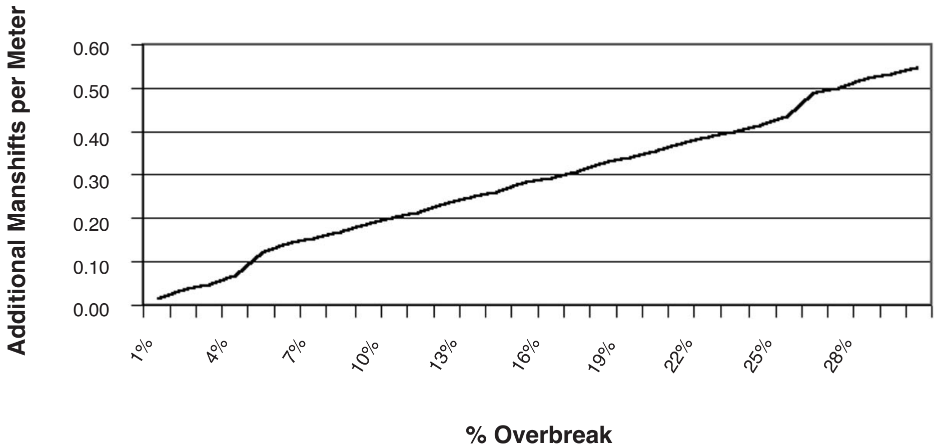

Figure 7 – Additional man-shifts required per metre vs. per cent overbreak

Forecast (2003–2006)

Forecasting was done in order to see what the implications would be of continuing to overbreak tunnels in the current manner. Figure 6 shows the function relating the theoretical additional costs required for loading and supporting overbroken areas. These costs exclude labour and consist only of vehicle running costs and materials. They also assume perfect grade control i.e. the footwall is even and does not need to be levelled. Similarly, Figure 7 shows the same relationship but between additional man-shifts required per metre vs. percentage overbreak. The man-shifts are based on the man shift efficiency calculations used by planning. The graphs are based on a 4 x 4 m tunnel, with a half circle profile. The step function in the graphs below represents the percentage where an additional rock bolt should be added to keep the spacing at 1 m.

Making use of this relationship, a forecast was done on the B Block life of mine development. The NPV was discounted @ 15%. If the average overbreak remains at 10% it will cost approximately R 1.5 million in loading and support materials and an additional 1 800 man-shifts will be required.

Conclusions

It has been shown that an overbreak average of 10% can have significant effects on the working costs and the scheduling of the operation as the workload is increased due to additional material that is required to be loaded and the larger area that is to be supported. In order to keep these problems to a minimum, constant monitoring of the drill and blast operations needs to take place. This should be in the form of a management system and hand-over procedures. Working guidelines are not sufficient to control these problems. Blast patterns must be updated on a regular basis to account for the geotechnical conditions and the equipment utilized.

Time has to be taken in order to standardize the various tunnel sizes and shapes and ensure that each of the departments follows this. This will ensure that tunnels are developed according to the design specification and that survey will measure accordingly i.e. overbreak and underbreak must not be reported in the same figures but rather separately.

Recommendations

Implementation of a blast management system (BMS)

This will allow for the formalization of blast designs. During the project various blast designs were done in order to give a starting point; these designs need to be tested. The BMS can record the results for each and every blast. This will allow the drill and blast engineer to see what has worked in various areas, and they can be recalled should these conditions be encountered again.

Perimeter powder factor and rock quality investigations

A study on the relationship between rock quality and perimeter powder factors should be undertaken. Both Rock quality (Q) and rock mass rating (RMR) should be utilized in order to find which of the systems shows a better relationship. These investigations can further be traced to cost of overbreak and cost of underbreak, and therefore for each type of rock one should be able to find the optimum perimeter powder factor, which will reduce the overall costs.

Kwikmark templates

Once various designs have been tested and optimized, a number of templates should be generated for the various situations. This will aid the miner in marking the face accurately and quickly.

Development database

The implementation of an SQL database that records targets for the month, as well as the survey measurements (offsets, advance, etc.). This will allow for efficient tracking of tunnel accuracy, overbreak and underbreak, digital photographs, geology, and rock mass ratings.

Tunnel hand-over procedures

Tunnel hand-over procedures should be implemented between the various processes. This will identify any specific problem areas and ensure a smooth changeover from predecessors and successors.

Revision of the bonus system

A model should be compiled that will allow for bonus systems to be measured not only on quantity but also on quality.

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Upgrades enhance caterpillar underground mining machines*

Three Caterpillar underground mining loaders from the Caterpillar Elphinstone stable–models R2900G, R1600G and R1300G—have been upgraded for improved performance across several key areas.

The R2900G features advanced electrical systems on all variants, including the R2900G XTRA (20 metric tonne capacity) that now also benefits from larger 35/65 R33 tyres for prolonged tyre life when operating at maximum carrying capacity in high cycle applications.

Both the R2900G and the R1600G (as well as the R1700G) incorporate new Caterpillar ACERT™ engine technology that improves emission levels by meeting the US EPA Tier 2 regulations on diesel exhaust emissions. Manufacturers are required to improve emission/economy levels in three stages or 'tiers' to meet set EPA guidelines by 2007.

By incorporating the new Tier 2 technology, Caterpillar Elphinstone's customers benefit immediately from the latest improvements in the drivetrain system.

The R1300G now features a more efficient electrical system, improved operator station ergonomics, as well as increased ground clearance at the rear of the unit. The carrying capacity of the R1300G has been increased from 6.5 to 6.8 metric tonnes.

'These upgrades effectively place us way ahead of the South African requirement to reduce emissions,' says Andy Watt, product manager: underground for Barloworld Equipment Mining. 'This is only the beginning as Caterpillar is now moving towards Tier 3 engine certification for this model and plans to incorporate this into its units in the coming year.' ◆

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