



Blast optimization at Kriel Colliery

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Synopsis

Kriel Colliery has experienced some sub-standard blast results on the overburden. Blast results are considered poor when the fragmentation is too big, back break into the new highwall, capping is experienced and/or excess overburden material is thrown into the void. Poor blasting leads to lower productivity, equipment breakdowns, and poor drilling and blasting results on the adjacent blast, contamination of coal or loss of coal.

A simulation was developed in Microsoft Excel to model blasting results while changing various parameters. From the simulation results, it could be seen that a change of blast design could be beneficial. As an example, smaller drilling patterns were simulated (10 m x 10 m to 8 m x 8 m, and 8 m x 8 m to 7 m x 7 m in Pit 5 and Pit 6 respectively) in combination with changes in explosives (P700 emulsion vs. ANFO) and drill bit diameter (311mm vs. 251mm). The results show that each of the options has a specific scenario that will be most beneficial to the mine. These results are quite specific and subject to variable inputs. The purpose of the example is to demonstrate how the simulation can be used as a tool to assess blast design variations.

Introduction

Mine background and general information

Kriel Colliery (Anglo Coal) is situated on the northern margin of the Highveld Coalfield area in Mpumalanga, 55 km south of Witbank. The Number 4 coal seam is extracted through surface and underground mining methods. The underground operation was started in 1975 and is currently divided into six mechanized sections. The opencast was commissioned in 1978 and consists of two dragline pits and a mini-pit.

Project background

General information on blasting operations

This study was conducted in both Pit 5 and Pit 6.

The overburden in Pit 5 is drilled on a 10 m x 10 m blast hole pattern and a bench height in excess of 30 metres. The blast holes are loaded with explosives to an 8 metre

stemming. In Pit 6 a 8 m x 8 m blast hole pattern is drilled and the average bench height is 20 metres. The holes are loaded to a 6 metre stemming length.

The major issues resulting from poor overburden blasting are: poor fragmentation, bad highwall conditions, capping, and unintentional casting into the void.

Poor fragmentation

The goal of blasting is to produce manageable rock fragments for ease of handling. Smaller fragments are easier to load and transport, so larger fragments are considered 'poorly fragmented'. According to Bezuidenhout (2008), the maximum fragment size a dragline can handle efficiently is 300 mm. Figure 1 shows the difference between acceptable and poor fragmentation experienced at the mine.

According to Dlamini (2008) larger fragments have an adverse affects on dragline productivity. When large rocks are encountered in the muckpile, the bucket is subjected to increased friction, which in turn leads to excessive wear and increased power usage (Levings 2008). This is called 'hard digging' and is illustrated in Figure 2.

Poor fragmentation also affects the dragline productivity through a lower bucket fill factor (Dlamini, 2008). When large rocks fill the bucket, not all the space is utilized (Figure 3). This reduces the amount of material moved per swing and therefore lowers the material shifted per cycle. This translates to a decrease in coal exposure, which eventually leads to less coal available to be mined.

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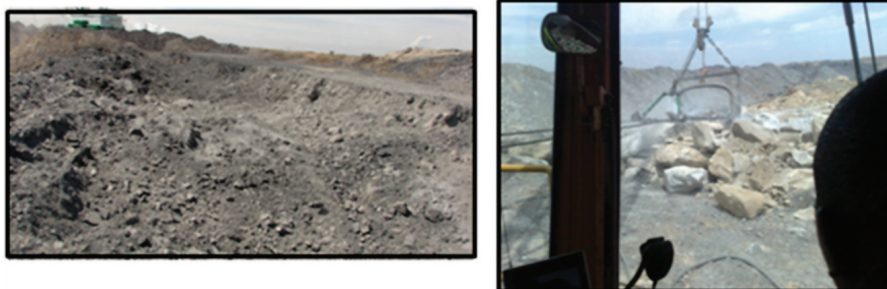


Figure 1—Good fragmentation (left) vs. poor fragmentation (right) at Kriel Colliery, (2008 photos by author)

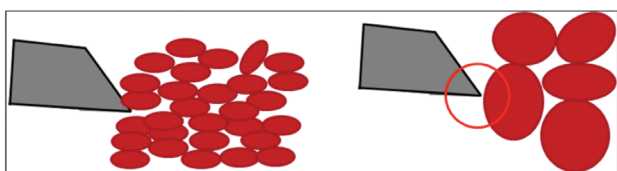


Figure 2—Hard digging as result of larger fragments (author's illustration)

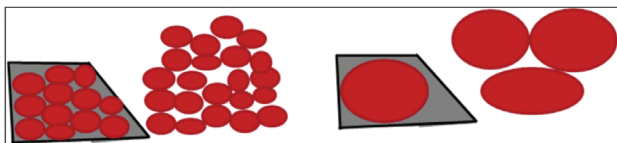


Figure 3—Bucket fill factor as influenced by fragment size (author's illustration)



Bad highwall

Figure 4—Highwall damage (Singh, 2008:17/12)

It can easily happen that poor blasting damages the highwall through overbreak or back damage. Figure 4 shows a damaged highwall crest. The dotted line indicates where the highwall crest should have been after the blast. Damaged crests can be hazardous and will also reduce the adjacent blast's effectiveness.

Coal capping

When the toe area of the overburden is not well fragmented it is very hard to remove by the dragline. The overburden portion is left on top of the coal and has to be removed by

another means. Cleanly removing capping is a formidable task and it is inevitable that dilution will occur in this area. This increases the contamination of the coal. Capping that cannot be removed will eventually result in coal losses. An estimated 8 000 t of coal is lost per cut (Levings 2008). Figure 5 shows capping after overburden removal.

Casting

On a few occasions the blasted overburden material was inadvertently thrown into the void. Cast blasting will lower the muckpile height compared to the bench height. The dragline must first elevate the pad to a suitable height before exposing the coal. This increases the re-handle percentage. This is undesired because the dragline has to do more work in a certain area, making its linear advance slower and therefore exposing less coal (Kok, 2008).

Problem statement

An investigation into the possible causes of poor blast results and the evaluation of possible solutions.

Objectives

- ▶ Investigate possible causes for the poor blast results
- ▶ Investigate methods to improve blast results
- ▶ Develop a model to simulate blast results
- ▶ Simulate and compare blast results by varying the individual blast parameters.

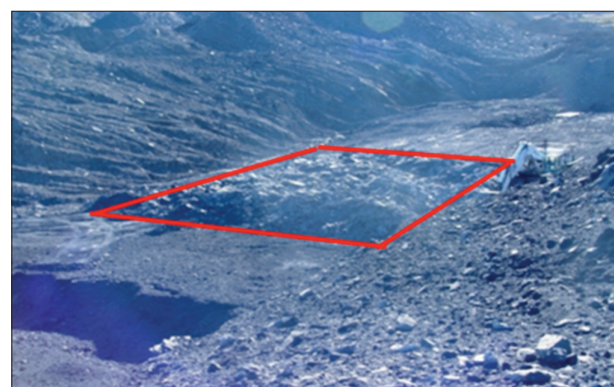


Figure 5—Coal capping (Singh, 2008:05/05)

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Methodology

- A literature study was completed on blast optimization and how to calculate the results of a blast.
- Investigating the reasons for the poor blast results and possible solutions consisted mostly of personal communication with mine employees and Dr W. Crosby, a world renowned mining consultant.
- To develop the simulation model, the formulae identified in the literature study were used. A time study was conducted on one of the drills to determine the penetration rate and manoeuvring speed. Blast reports reviews and interviews were used to establish the time associated with priming and charging. Drilling costs, explosives cost, and the explosive properties were obtained from the suppliers.
- Blast parameter variations were simulated and compared on a 200 m × 50 m standard blast. The results from the initial design, were compared to:
 - replacing the P700 explosives with ANFO,
 - replacing the 311 mm drill bit with a 251 mm drill bit and
 - substituting both the ANFO and the 251 mm drill bits. Some 20 different options for stemming length and hole spacing was simulated for each option.
- The options were varied between stemming lengths between 10 m, 8 m, 6 m and 5 m. The burden was changed between 5 m, 8 m, 10 m, 13 m, and 15 m while the spacing was kept the same as the burden.

Results from literature study

Blast optimization

Optimization of a blast, according to Thompson (2008: 4–5), can be done by determining the cost of each mine function (e.g., drill and blast, loading, hauling, etc.) related to the blast as a function of fragment size. The total cost of mining is represented by a parabola (Figure 6). From this the theoretical optimum fragmentation could be determined. In this figure higher fragmentation (more to the right of the axis) means 'better/finer' and not 'bigger'.

The optimization process described by Lopez Jimeno (c1995:325) is similar to what is discussed above:

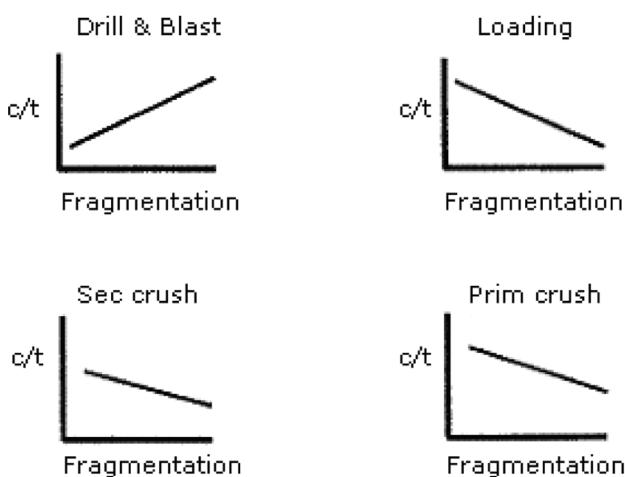


Figure 6—Blast optimization (Thompson 2008: 4–5)

- Calculate the pattern
- Predict the fragmentation
- Simulate the cost and productivity of all the processes involved with the fragmented material
- Compare this to other drill and blast patterns.

Fragmentation is not the only parameter used to assess the performance of a blast. Thompson (2008: 4–17) discusses muck pile shape, highwall toe, and fly rock as other means to measure results. In this study, fragmentation was the main measure of blast performance.

As stated earlier, during optimization different designs should be assessed. The parameters that can be changed in a design are relayed by Lopez Jimeno (2006: 179) as drill diameter, drill depth, stemming, burden, and spacing.

Predicting the results of a blast

Thompson (2008) explains the calculations of designing a blast. First the mass of explosives per metre is calculated by:

$$Mc = \frac{\pi d^2 \rho c}{4} \quad [1]$$

where: Mc = mass of explosives per linear metre
 d = drill hole diameter in metres
 ρ = explosive density
 c = coupling factor ($c = 1$ for pumpable explosives)

The burden to spacing ratio (A) and the stemming to burden ratio (Y) must be chosen. Next, if a suitable powder factor (PDF) has been decided upon, a recommended burden is calculated by:

$$B = \frac{-(A.Mc.Y) + \sqrt{(A.Mc.Y)^2 + 4(H^2 Mc.A.PDF)}}{2.H.PDF} \quad [2]$$

Where: B = burden (m)
 H = bench height (m)

The other parameters are described above.

Thompson (2008) further explains that the results of a blast can be predicted with the Rossin-Rammler equation for fragmentation distribution:

$$R = e^{-\left(\frac{x}{x_c}\right)^n} \quad [3]$$

where: R = mass fraction of fragments larger than size x
 x = fragment size (cm)
 n = Rossin-Ramler exponent (constant)
 x_c = characteristic fragment size (constant)

Rossin-Rammler exponent (n) can be calculated as follows:

$$n = \left(2.2 - 14 \frac{B}{d}\right) \left(1 - \frac{\omega}{B}\right) \left(1 + \frac{\left(\frac{1}{A} - 1\right)}{2}\right) \frac{L}{H} \quad [4]$$

where: ω = standard deviation in hole spacing
 L = charged length

The other parameters are described above.

To determine x_c it is first necessary to determine the average fragment size (x_{avg}) by:

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$$x_{avg} = A_f \left[\frac{M_h}{Vo} \right]^{-0.8} M_h^{0.167} \left[\frac{RWS}{115} \right]^{-0.63} \quad [5]$$

Where: A_f = rock factor
 M_h = mass of explosives per hole
 Vo = volume blasted per hole (m³)
 RWS = relative weight strength (relative to ANFO as 100)

Next, x_c is calculated by:

$$x_c = \frac{x_{avg}}{(0.693)^{\frac{1}{n}}} \quad [6]$$

Using these equations together is known as the KuzRam model (Cunningham, 1986). The fragmentation distribution can be graphically presented as in Figure 7.

Causes of poor blasting results

Experience loss (loss of skilled labour)

The recent commodity boom resulted in a worldwide shortage of skilled labour. Kriel Colliery was not spared in this. When commodity prices rise, higher production becomes priority and mines are prepared to offer higher remuneration for the skilled people. In these circumstances the turnover in skilled persons will be higher than normal. (Krugel, 2009)

Reaction to poor results

Initially the reaction to poor blast results is to decrease the stemming length, thereby increasing the charge length and the powder factor. When poor fragmentation occurs, the powder factor for the next blast is increased. Currently, the powder factor on the overburden is approximately 0.7 kg/BCM instead of the recommended 0.55 (Levings, 2008). This is what Thompson (2008: 4–17) calls the ‘hit harder approach’ as opposed to the more effective ‘hit smarter approach’. If the average explosive cost is in the range of R5/kg and the draglines move 850 000 BCM per month each (Olivier, 2009), the excessive 0.15 kg/BCM will translate to an overspending of R1.3 million per month.

Changing conditions

The geological conditions at Kriel Colliery vary significantly.

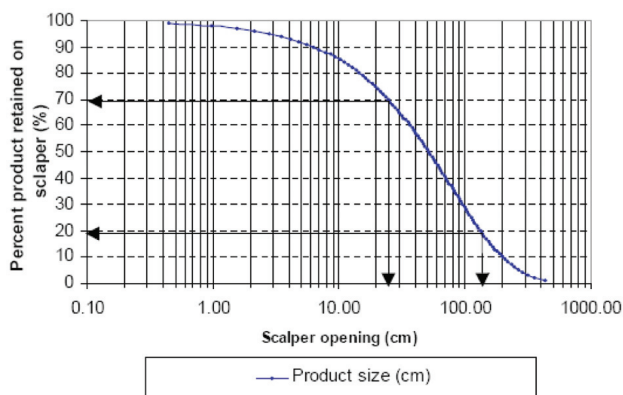


Figure 7—KuzRam model from Rossin-Rammler equation (Thompson, 2008: 4–20)

For example, the hard overburden may vary from 6 m to more than 30 m. The blast designs should be adapted with varying conditions.

Prestripping

Both Levings (2008) and Kok (2008) explained that the stemming length should be situated in the soft material zone on the top of a bench. Due to drilling difficulties it was decided to strip the layer of soft material. The stemming length is now situated in the hard zone. The stemming length was kept constant. Figure 8 illustrates a lower powder factor after prestripping, maintaining a similar stemming length.

Highwall damage

A damaged crest is not only a safety hazard, but it also adversely affects the energy distribution in the adjacent blast. The front row of blast holes may have to be moved backwards, resulting in an increased effective burden (Figure 9). Unacceptable fragmentation can be expected in this area of the blast (Levings, 2008).

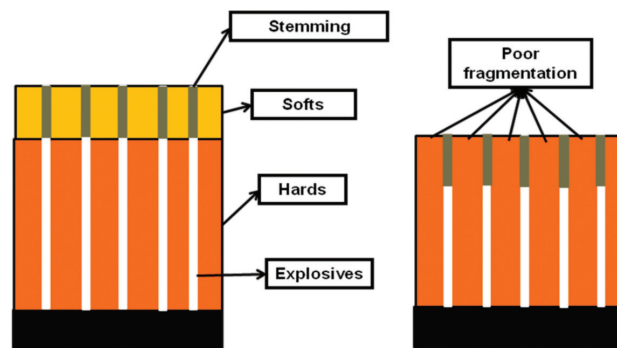


Figure 8—The effect of prestripping on fragmentation (author's illustration)

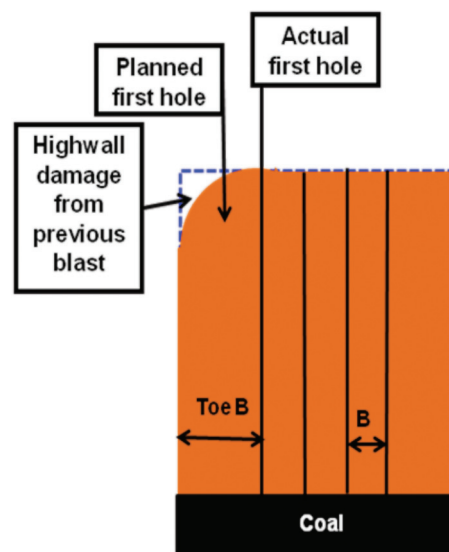


Figure 9—The effect of crest damage on the next blast hole layout. (author's illustration)

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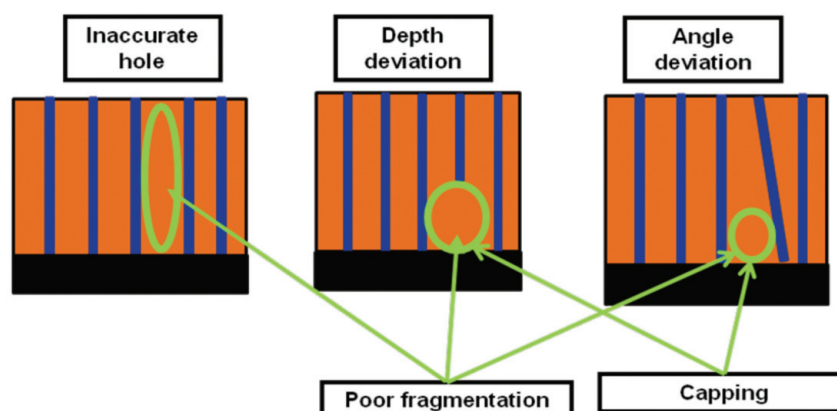


Figure 10—The effect of drilling inaccuracies on fragmentation (author's illustration)

Drilling accuracy

For any design to be successful, drilling has to be accurate. Deviations and large tolerances will lead to poor results. There may be a call for a reevaluation when, in fact, there is nothing wrong with the theoretical design. What was implemented might be a completely different design.

When a drill hole is off target (due to pegs moving, poor collaring, re-drilling, etc.) the effective spacing is increased in one area and decreased in another, changing the fragmentation accordingly (Figure 10). According to Crosby (2008), if a hole is misplaced by 0.5 m, it could reduce the explosive power at a certain point by 35%. Depth and angle deviations will also lead to zones of decreased explosive power, as shown in Figure 10.

Figure 11 shows the accuracies of the drill hole spacing at Kriel Colliery compared to New Vaal and Kleinkopje. This is indicative of a drilling placement problem.

Evaluation of solutions

Blast recording

To have optimal blasting practices, the change in geological conditions must be monitored and the design be altered accordingly. This can be done with feedback in a blast management system (LeJuge, 1996: 8). At the start of the project no data capturing systems were being utilized. Currently, all the data from every blast are recorded and analysed for continuous improvement for the adjacent strip (Levings, 2008). Proper record keeping of the blasts will assist in better decision making and improvements in blast results. A database will be available in future to evaluate changes and the effects of such changes.

Blast control

Before any design improvements can be made, it must be ensured that the current design is implemented correctly. A single design can have variable outcomes due to incorrect execution. Thus, the results from a changed design cannot necessarily be attributed to the design. Only when each hole is accurately placed (position, depth and angle), can true results be measured and different designs effectively compared.

Simulation of blast design variations—explanation of simulation

To model the blast design variations and evaluate the cost and impact, a simulation was made using Microsoft Excel. This simulation was designed to be user friendly (Figure 12). The person responsible for the blast can easily change a parameter to see what effect this will have on the fragmentation, cost, and time to drill the blast holes.

The simulation will be discussed as it will be used in practice. With each step the formulae, data and reasoning will be discussed.

E-mail communication with Smith (2009) revealed the properties (density and RWS) of the two explosives considered during the start of the project (ANFO and Emulsion P700). The density of ANFO is 900 kg/m³ and an RWS of 100. P700 Bulk emulsion has a density of 1250 kg/m³ and an RWS of 90.

Under 'drill size' one chooses between 311mm (currently used) and 251mm. According to Ball (2009), 251mm can be used on the current drill rig with a few minor changes.

The other parameters (bench height, burden, spacing, stemming length, blast zone width, and length) are also taken into consideration. The blast design method explained by Thompson (2008a) is done differently compared to the

Source data: ~ 100 holes at each pit

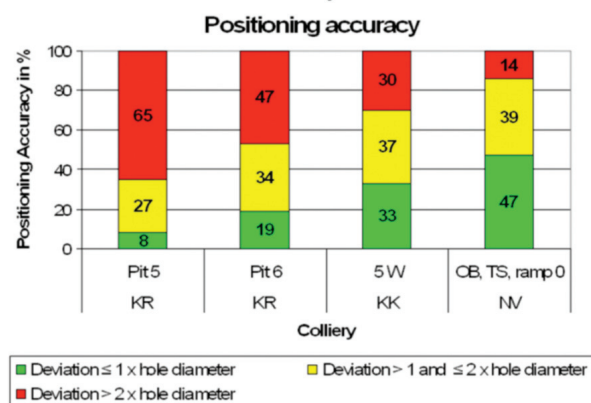


Figure 11—Drilling accuracies (Levings, Sa.)

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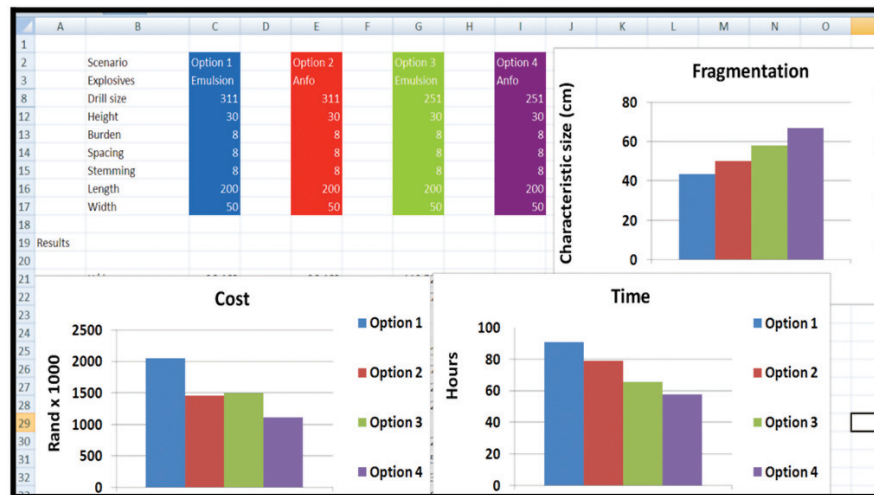


Figure 12—Simulation program developed to show the impact of design changes (Author)

model. For example, the model does not have 'powder factor' as an input, but rather burden, spacing, stemming length, etc., the things that can be changed in the field.

Once all the parameters are fed into the simulation, it calculates the cost, time, and fragmentation of the blast. The equations used are mostly those relayed by Thompson (2008) and are discussed in the literature survey. The simulation first calculates the mass of explosives per metre (Mc). The total mass of explosives per blast hole is calculated by taking the total mass per metre of charge multiplied by the charge length.

The number of holes drilled in a blast is calculated as follows. The length of the blast is divided by the spacing and rounded down to get the number of holes drilled in the length of the blast. The number of holes drilled in the width is calculated by taking the width and dividing it through the burden and rounded down. The two answers are multiplied by each other to give the total number of holes drilled for the blast.

A time study was performed on the GD3 (overburden drill) to get an indication of the walking and drilling speeds. Both penetration rate and walking speeds were rounded to 0.2 m/s. Calculating the penetration rate for the different size drill bits, the following formula was used (Thompson, 2008b: 4–20):

$$P_n = P_o \left(\frac{d_o}{d_n} \right)^{0.9} \quad [7]$$

where: P = penetration rate (m/s) or drill speed
 d = drill bit diameter (mm)
 o = old
 n = new

The penetration rates and walking speeds of the drills are used in the simulation to calculate the estimated drilling time per blast. The total drilling time is found by multiplying the number of holes by the bench height (total metres to be drilled) and dividing this by the penetration rate. The walking time is estimated by dividing the movement speed into the distance walked between the holes (approximated by multiplying the burden by the number of holes).

Previous blast reports were evaluated to obtain the average explosive charging times. It was found that the time taken per kilogram of explosives stays relatively constant (0.56 s/kg in Pit 5 and 0.49 s/kg in Pit 6) and was assumed at 0.5 s/kg. An additional 2 minutes per hole was assumed to cater for preparing the primer (Bezuidenhout, 2009).

The drilling and charging times were added to determine the total time needed for blasting a zone. This is one of the outputs on which the simulation compares blasts.

Consultation with Drilling Project Services (Ball, 2009) revealed the prices of the different drill bits available to the mine, and the anticipated life expectancy (the cost per metre could thus be calculated). It was determined that the 311 mm drill bit costs R5 per metre and the 251 mm drill bit costs R5.50 per metre. This was deemed too low an estimate. Values of R100/m and R110/m were assumed for the sake of the simulation.

According to Olivier (2008), one can assume that each hole contains one Pentolite 400g booster at R22.45 and one in-hole detonator at R31.40. Should the blast hole be deeper than 20 metres than a second primer is placed in the blast hole. For calculations in the model it is assumed that each hole requires 0.33 unit of 42 ms inter-hole delay at R25.28 and 0.66 unit of the 100 ms inter-hole delays at R23.01. The total initiation cost per hole will be R77.38 for bench heights lower than 20 metres and R131.23 where the bench height exceeds 20 metres.

The cost of the bulk explosives was obtained from Kok (2008). ANFO costs R4.42/kg and P700 costs R5.06/kg (based on January 2009 costs). The costs of the bulk explosive, initiation system, and the drilling were used to determine the cost of the blast, which is one of the final outputs of the simulation.

To model the fragmentation it was decided to use the Kuznetsov equation as it '...has been used with great success in South Africa...' (Thompson, 2008a:4-18). This equation predicts the average fragment size.

It was decided to compare results on characteristic size (X_c) instead of average size (X_{avg}). This is because X_c also gives an indication of the size distribution. For the comparative calculations, ω was assumed to be 10% of the burden. A (B:S ratio) was assumed to be 1.

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Simulation of blast design variations—simulating parameter changes

A 200 m x 50 m blast was simulated first leaving the design as it currently is as a control measure, i.e. benchmark. The next scenario was to replace the emulsion P700 explosives with ANFO. The third scenario was to use a 251mm drill bit with the original explosives. The final scenario was to use ANFO with a 251 mm drill bit.

The Excel spreadsheet was used to generate the results of 20 design changes (pattern and stemming length) for all the options. The data were plotted as cost vs. fragmentation for ease of comparison. The trend lines are shown in Figure 13 without the data points (for clarity). The lowest regression fit value (R^2) was 0.992, which means the trend line represents a very good fit of the data. The trend lines were assumed to be a good representation of the data points.

Figure 13 indicates that all the alternatives have the potential to be more cost-effective with similar fragmentation results (excluding capital costs). Utilizing a 251mm drill bit has the lowest potential of saving costs (compared to the current design), while using ANFO with the smaller drill bit has the lowest cost.

An example of exactly how a pattern varies, coupled with the abovementioned scenarios, can be beneficial; smaller patterns were simulated for both pits. Pit 5 was changed from 10 m x 10 m to 8 m x 8 m and Pit 6 from 8 m x 8 m to 7 m x 7 m. Both pits yielded similar results. The average between the pits is shown in Figure 14.

Each scenario presents pros and cons and to select an option will depend on the desired effect. Option 1 improves fragmentation drastically, but costs a lot. If costs are a constraint, the fragmentation can be improved to a lesser extent by choosing option 2 (the drilling and charging time will increase). Option 3 has little influence on all parameters and might not be worth the effort. If the goal is to be more cost-effective, option 4 will be the recommended option without major influences on the other parameters.

These results are quite specific and can fluctuate depending on a number of factors (for example, explosive costs). However, the purpose of this exercise is not to determine which options are best for a generic solution, but to demonstrate how the simulation can be used as a tool to solve blast design issues.

Simulation of blast design variations—shortcomings of simulation

When using the simulation and interpreting the results, one must understand that the simulation does have limitations and shortcomings:

- Firstly, the X_c value used is only a theoretical value and has not been correlated to practice.
- Geological changes are not simulated: the spreadsheet assumes homogeneity.
- The accuracy of the time study on the drill is debatable.
- Changing to a 251mm drill will require a different stabilizer, which could cost up to R40 000 (Ball, 2009).
- As stated earlier, a 100% change to ANFO is unpractical and an increase in ANFO use will require a dewatering unit.
- Finally, simulation cannot simulate a change of initiation and tie-up patterns.

Conclusions

The main reasons for poor blast results are as follows:

- Experience loss
- Hitting it harder, instead of smarter
- Pre-stripping
- Highwall damage
- Inaccurate drilling.

Methods to improve blast results include:

- Proper blast record keeping
- Improved control over blast design implementation
- Blast design changes.

A simulation program was developed to model blast results. By modelling changes in pattern, explosives and drill bits, it was shown how this simulator can be used as a tool to make more informed decisions about design changes.

Recommendations and suggestions for further work

- When fragmentation problems exist, the simulation program should be used to make 'smart' changes.
- Experiments with 251 mm drill bits should be performed to measure the actual difference in performance between 31 mm and 251 mm drill bits under the same conditions.
- The predicted simulation results should be correlated with actual measured results.
- Cost vs. fragmentation curves for the draglines should be quantified and used in conjunction with the cost vs. fragmentation curves for drilling and blasting. By doing this, an optimal fragmentation size can be determined.

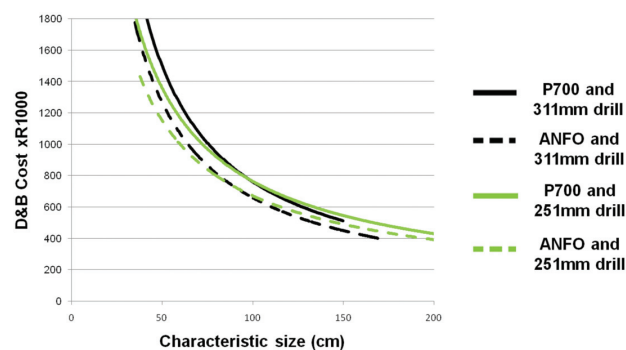


Figure 13—Cost vs. fragmentation trend lines for different options (Simulation program image)

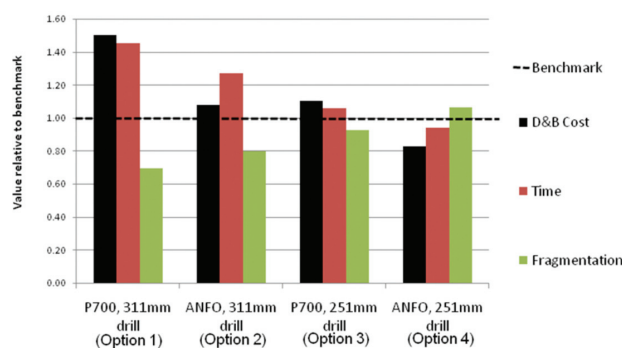


Figure 14—Results from tighter pattern simulation (Simulation program image)

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